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THE MINERAL INDUSTRY

ITS

STATISTICS, TECHNOLOGY AND TRADE

DURING

1911

FOUNDED BY RICHARD P. ROTHWELL

EDITED BY

CHARLES OF
Mining Engineer

VOLUME XX

SUPPLEMENTING VOLUMES I TO XIX

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THE STATE GEOLOGICAL SURVEYS

Of the 46 States of the Union, 35 have organised geological surveys, these comprising nearly all of the States in which the mining industry is important. The organisation of a geological survey in Massachusetts is under consideration. Certain States which have no geological survey have officials who give attention to the mining industry. Thus, California has a State mineralogist, while Idaho has a State mine inspector who collects statistics of mineral production. The States having organised geological surveys, together with the names and addresses of the respective State geologists, as of Jan. 1. 1912, are given in the following list:

STATE GEOLOGISTS.

State.	Name and Address.	State.	Name and Address.
Alabama	Eugene A. Smith, University.	New Jersey	H. B. Kümmel, Trenton.
Arizona	C. F. Tolman Jr., Tucson	New York	John M. Clark, Albany.
Arkansas	A. H.Purdue, Fayetteville.	N. Carolina	Joseph Hyde Pratt, Chapel Hill.
Connecticut	Wm. N. Rice, Hartford.	N. Dakota	A. G. Leonard, Grand Forks.
Colorado	R. D. George, Director, Boulder	Ohio	J. A. Bownocker, Columbus.
Florida	E. H. Sellards, Tallahassee.	Oklahoma	D. W. Ohern, Norman.
Georgia	S. W. McCallie, Atlanta.	Pennsylvania	H. C. Demming, Harrisburg.
Illinois	F. W. DeWolf, Director, Urbana	S. Carolina	Earle C. Sloan, Charleston.
Indiana	Edward Barratt, Indianapolis.	S. Dakota	E. C. Perisho, Vermillion.
Iowa	Geo. F. Kay, Director, Des Moines	Tennessee	A. H. Purdue, Nashville.
Kentucky	J. B. Hoenig, Frankfort.	Texas	W. B. Phillips, Austin.
Kansas	Erasmus Haworth, Lawrence.	Vermont	G. H. Perkins, Burlington.
Maine*	C. Vey Holman, Burnswick.	Virginia	Thos. L. Watson, Charlottesville.
Maryland	William Bullock Clark, Baltimore	Washington	Henry Landes, Seattle.
Michigan	R. C. Allen, Lansing.	West Virginia	I. C. White, Morgantown.
Mississippi	E. N. Lowe, Jackson.	Wisconsin	W. O. Hotchkiss, Madison.
Missouri	H. A. Buehler, Rolla.	Wyoming	C. E. Jamison, Cheyenne.
Nebraska	E. H. Barbour, Lincoln.		

STATE MINE INSPECTORS, COMMISSIONERS, ETC.

State.	Name and Address.
Alabama	C. H. Nesbit, Chief Mine Inspector, Birmingham.
Arkansas	G. B. Tucker, Commissioner, Bureau of Mines, Manufactures and Agriculture, Little Rock.
California	W. H. Storms, State Mineralogist, San Francisco.
Colorado	James Dalrymple, Coal Mine Inspector, Denver
Idaho	R. N. Bell, State Mine Inspector, Boise.
Indiana	Elliot R. Hooton, Chief Mine Inspector, Indianapolis.
Kansas	Frank Gilday, State Mine Inspector,
Kentucky	C. J. Norwood, Chief Mine Inspector, Lexington.
Minnesota	F. A. Wildes, State Mine Inspector, Hibbing.
Missouri	George Bartholomaeus, Secretary, Bureau of Mines and Mines Inspection, Jefferson City.
Montana	William Walsh, State Mine Inspector, Helena.
New Mexico	Jo E. Sheridan, Mine Inspector for the Territory of N. M., Silver City.
Ohio	George Harrison, Chief Inspector of Mines, Columbus.
Oklahoma	Ed. Boyle, Chief Inspector, Guthrie, Okla.
Oregon	H. M. Parks, Director, Bureau of Mines, Corvallis.
Pennsylvania	James Roderick, Chief, Department of Mines, Harrisburg.
South Dakota	R. L. Dougherty, State Mine Inspector, Lead.
Tennessee	R. A. Shiflett, Mine Inspector, Nashville.
West Virginia	John Laing, Chief, Department of Mines, Charleston.

* Officially abolished by the Legislature in 1911.

VALUE OF FOREIGN COINS

ESTIMATE BY DIRECTOR OF THE MINT, JAN. 1, 1911.

Country.	Standard.	Monetary Unit.	Value in Terms of U.S. Gold Dollar.	Coins.
Argentine Republic.	Gold	Peso	$0.965	Gold: argentine ($4.824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary.	Gold	Crown	.203	Gold: 10 and 20 crowns. Silver, 1 and 5 crowns.
Belgium	Gold	Franc	.193	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Gold	Boliviano	.389	Silver: boliviano and divisions.
Brasil	Gold	Milreis	.546	Gold: 5, 10 and 20 milreis. Silver: ½, 1 and 2 milreis.
British Possessions, N.A.	Gold	Dollar	1.000	.....
Central Amer. States—Costa Rica.	Gold	Colon	.465	Gold: 2, 5, 10 and 20 colons ($9.307) Silver: 5, 10, 25 and 50 centimos.
British Honduras.	Gold	Dollar	1.000	.....
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	.400	Silver: peso and divisions.
Chile	Gold	Peso	.365	Gold: escudo ($1.825), doubloon ($3.-.650), and condor ($7.300). Silver: peso and divisions.
China	Silver	Tael: Amoy	.655	.....
		Tael: Canton	.654	.....
		Tael: Cheefoo	.627	.....
		Tael: Chin Kiang	.640	.....
		Tael: Fuchau	.606	.....
		Tael: Haikwan (customs)	.667	.....
		Tael: Hankow	.613	.....
		Tael: Kiaochow	.635	.....
		Tael: Nankin	.649	.....
		Tael: Niuchwang	.615	.....
		Tael: Ningpo	.630	.....
		Tael: Peking	.639	.....
		Tael: Shanghai	.599	.....
		Tael: Swatow	.605	.....
		Tael: Takau	.660	.....
		Tael: Tientsin	.635	.....
		Dollar: Hongkong	431	.....
		Dollar: British	.431	.....
		Dollar: Mexican	.434	.....
Colombia	Gold	Dollar	1.000	Gold: condor ($9.647) and double condor. Silver: peso.
Denmark	Gold	Crown	.268	Gold: 10 and 20 crowns.
Ecuador	Gold	Sucre	.487	Gold: 10 sucres ($4.8665). Silver: sucre and divisions.
Egypt	Gold	Pound (100 piasters)	4.943	Gold: pound (100 piasters), 5, 10, 20 and 50 piasters. Silver: 1, 2, 5, 10 and 20 piasters.
Finland	Gold	Mark	.193	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Gold	Franc	.193	Gold: 5, 10, 20, 50 and 100 francs. Silver 5 francs.
German Empire	Gold	Mark	.238	Gold: 5, 10 and 20 marks.
Great Britain	Gold	Pound sterling	4.866½	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Gold	Drachma	.193	Gold: 5, 10, 20, 50 and 100 drachmas. Silver: 5 drachmas.
Haiti	Gold	Gourde	.965	Gold: 1, 2, 5 and 10 gourdes. Silver: gourde and divisions.
India (British)	Gold	Pound sterling*	4.866½	Gold: sovereign (pound sterling). Silver: rupee and divisions.
Italy	Gold	Lira	.193	Gold: 5, 10, 20, 50 and 100 lire. Silver: 5 lire.
Japan	Gold	Yen	.498	Gold: 5, 10 and 20 yen. Silver: 10, 20 and 50 sen.

NOTE.—The coins of silver-standard countries are valued by their pure silver contents, at the average market price of silver for the three months preceding January 1, 1910.

*The sovereign is the standard coin of India, but the rupee ($0.324½) is the current coin, valued at 15 to the sovereign.

VALUE OF FOREIGN COINS

Country.	Standard.	Monetary Unit.	Value in Terms of U.S.Gold Dollar.	Coins.
Liberia	Gold	Dollar	1.000	
Mexico	Gold	Peso†	.498	Gold: 5 and 10 pesos. Silver: dollar‡ (or peso) and divisions.
Netherlands	Gold	Florin	.402	Gold: 10 florins. Silver: 2½, 1 florin, and divisions.
Newfoundland	Gold	Dollar	1.014	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	.268	Gold: 10 and 20 crowns.
Panama	Gold	Balboa	1.000	Gold: 1, 2½, 5, 10 and 20 balboas. Silver: peso and divisions.
Persia	Silver	Kran‡‡	.0896	Gold: ½, 1, and 2 tomans ($3.409). Silver: ¼, ½, 1, 2 and 5 krans.
Peru	Gold	Libra	4.866⅔	Gold: ½ and 1 libra. Silver: sol and divisions.
Philippine Islands	Gold	Peso	.500	Silver peso: 10, 20 and 50 centavos.
Portugal	Gold	Milreis	1.080	Gold: 1, 2, 5 and 10 milreis.
Roumania	Gold	Leu	.193	
Russia	Gold	Ruble	.515	Gold: 5, 7½, 10 and 15 rubles. Silver: 5,10, 15, 20, 25, 50 and 100 copecks.
Servia	Gold	Dinar	1.93	
Siam	Gold	Tical	.3708	
Spain	Gold	Peseta	.193	Gold: 25 pesetas. Silver: 5 pesetas.
Straits Settlement	Gold	Pound sterling ‡	4.866⅔	Gold: sovereign (pound sterling). Silver: dollar and divisions.
Sweden	Gold	Crown	.268	Gold: 10 and 20 crowns.
Switzerland	Gold	Franc	.193	Gold: 5, 10, 20, 50 and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	.044	Gold: 25, 50, 100, 250 and 500 piasters.
Uruguay	Gold	Peso	1.034	Gold: peso. Silver: peso and divisions.
Venesuela	Gold	Bolivar	.193	Gold: 5, 10, 20, 50 and 100 bolivars. 5 bolivars.

† Seventy-five centigrams fine gold.
‡ Value in Mexico, $0.498.
‡ The current coin of the Straits Settlements is the silver dollar issued on Government account, and which has been given tentative value of $0.567758⅓.
‡‡ The Gold Kran $.1704

Russian Weights and Measures.

1 Dolia	= .6857 grs. troy	= (for gold at £3 13s per oz.) 1 1/4d.
1 Zolotnik	= 65.8329 grs. troy	= (for gold at £3 13s per oz.) 10s.
1 Funt	= 13.1665 oz. troy	= (for gold at £3 13s per oz.) £48.
1 Pood	= 526.6632 oz. troy	= (for gold at £3 13s per oz.) £1922.
1 Pood	= 36.11 lb. av.	
62.025 Poods	= 1 ton (2240 lb.).	
1 Arschin	= 2 ft. 4 in.	
1 Sasben	= 7 ft.	
1 Deciatine	= 2400 sq. sazhens	= 2.7 acres.
1 Verst	= 500 sazhens	= 1166 yd. or 2/3 mile approximately.

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INTRODUCTION

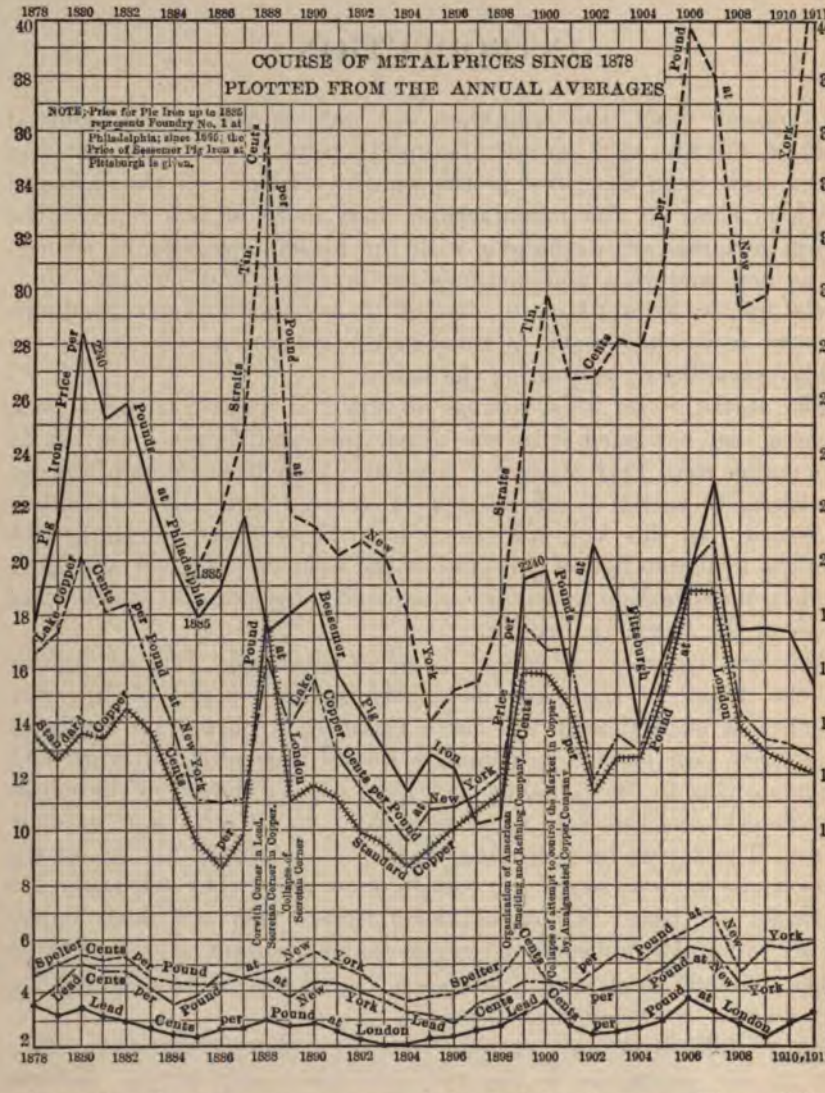
In recent years, the industries, not only of the natural mineral products, but also of those manufactured products, that are intimately related, have been described in this annual volume. As the processes by which some of the manufactures that more or less replace the natural minerals are developing with the present progress in science, and as new materials and methods are invented and put into commercial use, we cannot hope to treat all these under separate headings, but have mentioned many of them in kindred articles where they are readily accessible by consulting the index.

The statistics for this volume of MINERAL INDUSTRY have been prepared and corrected with the usual care, and figures referring to years earlier than 1911, wherever necessary, have been revised in order to conform to corrected official reports received since the last volume was issued. In this Vol. XX, therefore, will be found for the recent years the available figures accepted in the light of the latest and most complete investigation and it is consequently *important to use the* LAST volume of the MINERAL INDUSTRY.

Absolute accuracy, of course, is never attainable, but we believe that the degree of accuracy characterizing the volumes of MINERAL INDUSTRY is such that it will serve commercial purposes. Statistics for the last year cannot be complete for all olasses of minerals and products owing to the fact that they have to be collected in the course of six months and therefore many items have to be estimated to some extent. The figures for preceding years in many cases, though not all, can be made more accurate to a certain point where they will stand, for practically all time. Technical and business men will find figures that are promptly published and of reasonable accuracy of greater value in their pursuits than those statistics a year or two late, although the latter may have their final accuracy.

Many difficulties are encountered in gathering accurate data from all the producers in the very extensive and greatly divided industries. In almost every case someone reports carelessly and there is confusion as to the stage in which many products are to be reported. In a few cases there are those unwilling, for some unaccountable reason, to give their figures. Errors not only in amount produced but in the

money values lead to confusion, some producers estimating the worth of their product at the mine and others stating it in a more or less advanced stage of completion at the works, and perhaps in many cases, including


COURSE OF METAL PRICES SINCE 1878
PLOTTED FROM THE ANNUAL AVERAGES
NOTE:-Price for Pig Iron up to 1895 represents Foundry No. 1 at Philadelphia; since 1896, the Price of Bessemer Pig Iron at Pittsburgh is given.
Pig Iron Price per 2240 Pounds at Philadelphia
Lake Copper Cents per Pound at New York
Standard Copper
Tin, Straits Pound at New York
Tin, Straits Cents per Pound at London
Bessemer Pig Iron Price per 2240 Pounds at Pittsburgh
Lake Copper Cents per Pound at New York
Standard Copper
Tin, Straits Cents per Pound at New York
Tin, Straits Pound at London
Speiter Cents per Pound at New York
Lead Cents per Pound at New York
Speiter Cents per Pound at London
Lead Cents per Pound at London
Cornish Corner in Lead.
Standard Corner in Copper.
Collapse of Standard Corner
Organization of American Smelting and Refining Company
Collapse of attempt to control the Copper Metal Market by Amalgamated Copper Company
1878 1880 1882 1884 1886 1888 1890 1892 1894 1896 1898 1900 1902 1904 1906 1908 1910 1911

unintentionally treatment costs, and also freight charges. These difficulties appear not only in our statistics but also in those reported by various governments.

For many of the statistics relating to the mineral production in the

United States in 1911 and previous years, we are indebted to the United States Geological Survey; for the production of gold and silver in the United States to George E. Roberts, Director of the Mint; and for the statistics of American imports and exports, to O. P. Austin, Chief of the Bureau of Statistics of the Department of Commerce and Labor. Acknowledgment is due also to various State Geological Surveys and Statistical Bureaus for facts incorporated in this volume. Generally in individual cases footnotes or mention in the text give proper credit for information but this acknowledgment stands for any unintentional oversight. Special acknowledgment is hereby given to those heads of departments in various countries that have courteously coöperated in furnishing reports, in many cases prior to the publication of their official reports. We have indicated these in footnotes or otherwise as "officially reported," by the respective governments.

The statistics of copper, lead, zinc, and iron are compiled largely from data published in the *Engineering and Mining Journal*. Statistics as to coal and coke have been taken from *Coal Age*. Many of the reviews of the important districts have been adapted from articles in the numerous mining journals and as heretofore in each case, due credit has been given so that by following up these references more detailed information is available. The various articles have been abstracted and condensed in accordance to the importance of the subjects they treat but generally it is impossible to reproduce many excellent articles on account of lack of space, and many of these, therefore, are largely represented by bibliographies which cover the most important literature that has appeared during the year.

Many subjects have been treated by the contributors to Vol. XX. The standing of these contributors is high and the result is a noteworthy review of industrial progress, while the records of current events are supplied by thoroughly informed reporters. The progress and the mining industry of each mineral are described by specialists or by the members of State Geological Surveys, many of whom are contributors to Vol. XX as they were to Vols. XIX and XVIII. These contributors give an official character to the publication which constitutes a valuable and appreciated medium for bringing to general attention the statistical researches and commercial investigations in several states. While this is work that is most properly done by the State Geological Surveys, The MINERAL INDUSTRY affords them a means for the prompt and coördinated presentation of reports that otherwise might not become useful.

The reader of these pages will obtain the idea that 1911 was fairly prosperous in the mining industry and will, perhaps, find it difficult to understand the pessimism that prevailed. Many of the important sub-

PRODUCTION OF METALS IN THE UNITED STATES. (x)

Products.	Measures.	1910.		1911.	
		Quantity.	Value.	Quantity.	Value.
Aluminum	lb.	12,000,000	$2,700,000	28,600,000	$5,720,000
Antimony	lb.	4,630,000	365,000	3,566,000	296,500
Copper	lb.	1,086,249,983	138,366,522	1,083,856,371	134,398,190
Ferromanganese (q)	Lg. T.	224,431	9,533,829	184,717	8,496,982
Gold fine	Troy oz.	4,657,431	96,269,100	4,655,710	96,233,528
Iron, pig	Lg. T.	27,303,567	442,311,427	23,649,344	345,280,422
Lead	Sh. T.	392,704	34,809,000	400,958	35,445,000
Nickel	lb.	(e)600,000	300,000	300,000	150,000
Platinum	Troy oz.	773	25,277	929	40,058
Quicksilver (o)	Flasks.	22,418	1,054,991	21,500	1,466,000
Silver, fine	Troy oz.	57,137,900	57,137,900	57,796,117	30,805,338
Sodium	Sh. T.	(e)2,000	1,000,000	(e)2,000	1,000,000
Tin	Sh. T.	(e)5,500	3,751,000	(e)5,500	3,751,000
Zinc (y)	Sh. T.	277,065	30,587,976	295,836	34,021,000

Additional details will be found under the respective captions farther on in this volume. (e) Estimated. (o) Flasks of 75 lb. (q) Includes spiegeleisen, although the value is given as for ferromanganese. (v) Recovered from scrap metal. (w) Statistics not available. (x) Includes only metal produced from domestic ores except in case of zinc. (y) Includes zinc from foreign ore.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES.

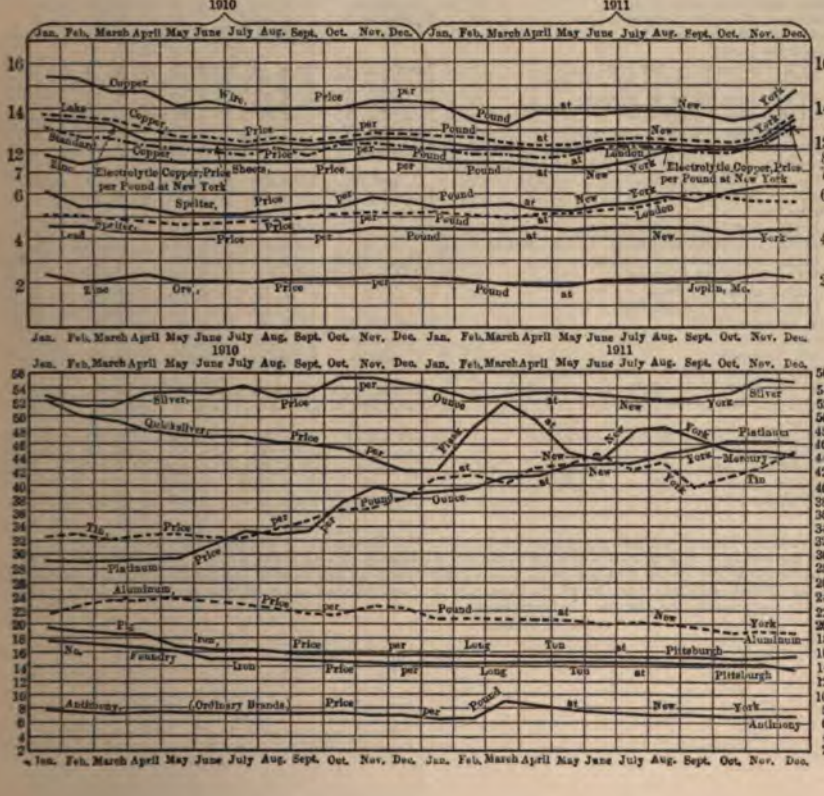
Products.	Measures.	1910.		1911.	
		Quantity.	Value.	Quantity.	Value.
Antimony ore	Sh. T.	75	$3,700	50	$2,000
Asbestos	Sh. T.	3,630	64,350	7,604	119,935
Asphaltum	Sh. T.	260,080	3,080,067	(u)360,004	3,820,751
Barytes	Sh. T.	41,720	145,187	41,057	152,732
Bauxite	Lg. T.	149,679	778,203	155,618	757,860
Borax	Sh. T.	(e)37,000	1,950,000	(u)53,330	1,569,151
Chrome ore	Lg. T.	450	6,075	120	1,629
Coal, anthracite	Sh. T.	80,389,306	160,843,081	90,490,356	174,822,183
Coal, bituminous	Sh. T.	411,038,616	471,388,557	411,830,509	485,666,177
Emery	Sh. T.	(w)1,028	15,077	770	8,810
Feldspar	Sh. T.	80,822	501,451	89,552	560,851
Fluorspar	Sh. T.	(e)46,858	270,840	47,000	270,000
Fuller's earth	Sh. T.	30,857	277,293	34,668	335,350
Garnet	Sh. T.	5,297	151,700	4,285	121,759
Graphite, amorphous	Sh. T.	1,407	39,710	1,223	32,415
Graphite, crystalline	lb.	5,625,132	340,676	4,790,000	256,050
Gypsum	Sh. T.	2,379,057	6,523,029	2,323,970	6,462,035
Iron ore	Lg. T.	53,263,553	95,881,315	43,846,476	76,731,333
Limestone flux	Lg. T.	(u)18,203,882	9,345,733	(w)	
Magnesite	Sh. T.	(e)8,000	59,780	(e)6,000	48,000
Manganese ore (d)	Lg. T.	2,258	22,892	(u)2,457	24,586
Mica sheet	lb.	(u)2,476,190	283,832	1,887,201	310,254
Mica scrap	Sh. T.	(u)4,065	53,265	3,512	45,550
Molybdenum ore	Sh. T.	*Nil.*	.....	*Nil.*	.....
Monazite	lb.	99,301	12,006	3,561	427
Petroleum, crude	Bbl. (i)	208,196,415	124,900,000	213,381,884	117,360,000
Phosphate rock	Lg. T.	2,681,938	11,961,956	3,123,584	12,237,111
Pumice	Sh. T.	23,271	94,943	(u)21,689	88,399
Pyrite	Lg. T.	223,700	830,150	261,087	1,062,914
Salt	Bbl. (k)	30,305,656	7,900,344	31,183,968	8,345,692
Sand, glass	Sh. T.	1,461,089	1,516,411	(w)	
Sand, molding	Sh. T.	3,636,167	2,431,254	(w)	
Sulphur	Lg. T.	255,545	5,622,000	206,200	4,540,000
Talc, ordinary ground and soapstone	Sh. T.	79,006	864,213	81,521	1,032,732
Talc, fibrous	Sh. T.	71,710	728,180	(e)62,500	562,500
Tin ore	Sh. T.	20	7,500	*Nil.*	
Tungsten	Sh. T.	2,130	844,526	(u)1,125	450,000
Zinc ore	Sh. T.	569,572	.....	669,305	.....

Additional details will be found under the respective captions farther on in this volume. (d) Does not include manganiferous iron ore. (e) Estimated. (i) Barrels of 42 gal. (k) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 lb. (u) Figures reported by the United States Geological Survey. (v) Includes graphite shale. (w) Statistics not yet available.

PRODUCTION OF SECONDARY MINERALS AND CHEMICALS IN THE UNITED STATES.

Products.	Measures.	1910.		1911.	
		Quantity.	Value.	Quantity.	Value.
Alundum	lb.	13,410,000	$804,600	11,116,000	$666,960
Ammonium sulphate	Sh.T.	116,000	6,449,960	127,000	7,216,360
Arsenic	lb.	2,652,000	66,300	6,162,000	154,050
Bromine	lb.	850,300	110,539	1,199,000	235,500
Carborundum	lb.	10,707,110	642,427	10,376,620	622,597
Cement, natural hydraulic	Bbl.(*g*)	1,500,000	600,000	926,091	378,533
Cement, portland	Bbl.(*h*)	76,549,951	68,205,800	78,528,637	66,248,817
Cement, puzzolan	Bbl.(*t*)	160,000	96,000	93,230	77,786
Coke	Sh.T.	36,094,769	83,314,084	33,349,754	79,340,624
Copper sulphate (*c*)	lb.	26,356,788	1,043,342	33,454,000	1,472,000
Copperas	Sh.T.	38,896	427,856	35,501	390,500
Crushed steel	lb.	825,000	57,750	800,000	56,000
Graphite, artificial	lb.	13,149,000	945,000	10,144,000	664,000
Lead, white, in oil	Sh.T.	111,572	15,027,993	(*w*)	
Lead, white, dry	Sh.T.	32,992	3,457,897	(*w*)	
Lead, red	Sh.T.	19,833	2,453,014	(*w*)	
Lead, orange mineral	Sh.T.	(*u*)823	136,082	(*w*)	
Litharge	Sh.T.	(*u*)23,766	2,689,016	(*w*)	
Mineral, wool	Sh.T.	8,408	84,012	7,514	65,500

Additional details will be found under the respective captions farther on in this volume. (*c*) Does not include sulphate made from metallic copper. (*e*) Estimated. (*g*) Barrels of 265 lb. (*h*) Barrels of 380 lb. (*t*) Barrels of 330 lb. (*u*) Figures reported by the United States Geological Survey. (*w*) Statistics not yet available.


1910
1911
Jan. Feb. March April May June July Aug. Sept. Oct. Nov. Dec.
Copper, Lake
Copper, Electrolytic
Copper, Wire, Price per Pound at New York
Copper, Sheets, Price per Pound at New York
Electrolytic Copper, Price per Pound at New York
Spelter, Price per Pound at St. Louis
Lead, Price per Pound at New York
Zinc, Grey, Price per Pound at Joplin, Mo.
Silver, Price per Ounce
Quicksilver, Price per Flask
Platinum, Price per Ounce
Aluminum, Price per Pound at New York
Pig Iron, Foundry, Price per Long Ton at Pittsburgh
Antimony, (Ordinary Brands), Price per Pound at New York

AVERAGE MONTHLY PRICES OF CHEMICALS,
(In Carload lots.

Material.	Unit.	Jan.	Feb.	Mar.	April.
Abrasives:					
Corundum	C. per lb.	8.50	8.50	8.50	8.50
Emery, Turkish flour in kegs	C. per lb.	1.875	1.875	1.875	1.875
Emery, Naxos flour	C. per lb.	1.875	1.875	1.875	1.875
Emery, Chester flour	C. per lb.	1.75	1.75	1.75	1.75
Emery, Peekskill flour, f. o. b., Easton, Penn.	C. per lb.	1.50	1.50	1.50	1.50
Garnet	$ per Sh. T.	30.00	30.00	30.00	30.00
Boric acid	C. per lb.	7.00	7.00	7.00	7.00
Borax sacks	C. per lb.	3.75	3.75	3.75	3.75
Alum, lump	$ per 100 lb.	1.75	1.75	1.75	1.75
Aluminum sulphate, com'l	$ per 100 lb.	1.45	1.45	1.45	1.45
Ammonium sulphate	$ per 100 lb.	2.875	2.875	2.875	2.875
Arsenic, white, spot	C. per lb.	2.25	2.187	2.063	1.937
Barytes, American ground	$ per. Sh. T	11.50	11.50	11.50	12.50
Chrome ore, New Caledonia, 50 per cent. ex. ship. N. Y	$ per Lg. T.	15.00	15.00	15.00	15.00
Cobalt oxide	C per lb.	82.50	82.50	82.50	82.50
Copperas, bulk	C. per 100 lb	55.00	55.00	55.00	55.00
Copper sulphate, carload lots	C per lb.	4.125	4.125	4.125	4.125
Fluorspar, lump	$ per Lg. T.	9.00	9.00	9.00	9.00
Fuller's earth, lump	C. per 100 lb.	82.50	82.50	82.50	82.50
Magnesite, Greece, crude 95 per cent	$ per Lg. T.	8.00	8.00	8.00	8.00
Magnesite, calcined, powdered	$ per Sh. T.	31.50	31.50	31.50	31.50
Gilsonite, Utah ordinary	$ per Lg. T.	36.00	60.00	48.00	45.00
Litharge, Am. powdered	C. per lb.	6.00	6.00	6.00	5.875
Red lead, American	C. per lb.	6.875	6.875	6.875	6.875
White lead, Am. dry	C. per lb.	5.75	5.75	5.75	5.75
Nitrate of soda, 95 per cent. spot	$ per 100 lb.	2.137	2.10	2.10	2.10
Zinc white, Am. extra dry	C. per lb.	5.937	5.937	5.937	5.937
Phosphate rock:					
*Florida hard, 77 per cent	$ per Lg. T.	4.75	4.75	4.75	5.875
Florida land pebble, 68 per cent	$ per Lg. T.	3.675	3.675	3.675	3.675
†Tennessee, 78 to 80 percent	$ per Lg. T.	5.25	5.25	5.25	5.25
Tennessee, 75 per cent	$ per Lg. T.	4.875	4.875	4.875	4.875
Tennessee, 68 to 72 per cent	$ per Lg. T.	4.375	4.375	4.375	4.375
‡South Carolina land rock, 60 per cent	$ per Lg. T.	3.625	3.625	3.625	3.625
Potassium cyanide, 98 and 99 per cent., 5-ton lots	C. per lb.	18.50	18.50	18.50	18.50
Kainit, bulk	$ per Lg. T.	8.50	8.50	8.50	8.50
Pyrites:†					
Dom., non-arsenical furnace, f. o. b. mines	C per Unit S. per L. T.	11.75	11.75	11.75	11.75
Dom., non-arsenical fines, f. o. b. mines		10.875	10.875	10.875	10.875
Imp., non-arsenical furnace		13.00	13.00	13.00	13.00
Imp., arsenical furnace		12.25	12.25	12.625	12.25
Imp., arsenical fines		9.75	9.75	9.75	9.75
Imp., non-arsenical fines		11.25	10.75	10.75	10.75
Soda, caustic 78/60	$ per 100 lb.	1.90	1.90	1.90	1.90
Salt cake, bulk	C. per 100 lb.	55.00	55.00	55.00	55.00
Sodium bromide	C. per lb.	20.00	20.00	20.00	20.00
Sodium cyanide, 120 to 130 per cent. KCN per 100 per cent., 5-ton lots	C. per lb.	18.00	18.00	18.00	18.00
Sodium sulphate, com'l calcined	C. per 100 lb.	75.00	75.00	75.00	75.00
Sulphur:					
Louisiana, prime to New York	$ per Lg. T.	22.00	22.00	22.00	22.00
Talc, domestic	$ per Sh. T.	16.00	16.00	16.00	20.00
Zinc chloride, gran	C. per lb.	4.125	4.125	4.125	4.125
Zinc dust	C. per lb.	6.875	6.875	6.875	6.875
Zinc sulphate	C. per lb.	2.25	2.25	2.25	2.25

* F. o. b. Florida or Georgia ports. † F, o. b. Mt. Pleasant. ‡ On vessel, Ashley River, S. C.

NOTE.—These quotations were for ordinary wholesale lots at New York unless otherwise specified and were generally subject to the usual trade discounts. In the case of some of the important minerals such as pyrites, sulphur, etc., in which there are well established markets the quotations fully represent

†25 cents deduction per ton for large lumps.

EARTHS, MINERALS, ETC., IN 1911.
at New York).

May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average, 1911.	Average, 1910.
8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50
1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875
1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875	1.875
1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75
1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50
30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00
7.00	7.00	7.00	7.00	7.00	7.00	7.00	7.00	7.00	7.00
3.75	3.75	3.75	3.75	3.75	3.75	3.75	3.75	3.75	4.166
1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75
1.45	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.60	1.679
3.125	3.125	3.125	3.125	3.125	3.125	3.125	3.125	3.042	2.846
1.937	1.563	1.563	1.875	1.937	2.125	2.125	2.313	1.99	2.449
12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.50	12.25	13.50
15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00
82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50
55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00
4.375	4.625	4.625	4.625	4.625	4.625	4.625	4.625	4.437	4.15
9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50	82.50
8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.15
31.50	30.50	30.50	30.50	30.50	30.50	30.50	30.50	30.92	31.50
45.00	43.00	43.00	43.00	43.00	43.00	43.00	43.00	45.00	36.00
5.875	5.875	7.50	7.50	7.50	7.50	7.50	7.50	6.72	6.01
6.875	6.875	6.875	6.875	6.75	6.75	6.75	6.75	6.833	6.761
5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.928
2.10	2.10	2.125	2.15	2.175	2.20	2.20	2.20	2.307	2.107
5.937	6.125	6.125	6.125	6.125	6.125	6.125	6.125	6.046	5.781
6.00	5.875	6.00	6.00	6.00	5.875	5.875	6.00	5.649	6.666
3.875	3.875	3.875	3.875	3.75	3.75	3.75	3.75	3.766	3.875
5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25
4.875	4.875	4.875	4.875	4.875	4.875	4.875	4.875	4.875	4.875
4.375	4.375	4.375	4.375	4.375	4.375	4.375	4.375	4.375	4.375
3.625	3.625	3.625	3.625	3.625	3.625	3.625	3.625	3.625	3.81
18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50
8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	9.062
11.75	12.25	12.25	12.25	12. 5	12.25	12.25	12.25	12.04	11.416
10.875	11.25	10.500	10.50	10.50	10.50	10.50	11.00	10.76	10.832
13.00	13.125	13.125	13.125	13.125	13.125	13.125	13.25	13.083	12.729
12.25	12.625	12.625	12.625	12.625	12.625	12.625	12.625	12.917	12.130
9.75	9.75	9.75	9.75	9.75	9.75	9.75	12.00	9.937	9.437
10.75	10.75	10.75	10.75	10.75	10.75	10.75	12.50	10.938	10.916
1.90	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.813	1.805
55.00	56.25	56.25	56.25	56.25	56.25	56.25	56.25	55.73	53.33
20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.50
75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00
22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00
20.00	14.00	18.50	18.50	18.50	18.50	18.50	18.50	17.75	16.00
4.125	4.125	4.125	4.125	4.125	4.125	4.125	4.125	4.125	4.219
6.875	6.875	6.875	6.875	6.875	6.875	6.875	6.875	6.875	6.663
2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25	2.25

the latter. But in the cases of some of the minor mineral products, the quotations represent what dealers asked of consumers and not what producers realized in selling their outputs as matter of private contract. The monthly figures are the averages of the quotations published in the *Engineering and Mining Journal*, Jan. 7, 1911.

stances show increased production. In many cases the outputs were the largest on record, that having occurred in the cases of copper, lead and spelter. On the other hand there were decreases in the production of coal, petroleum and iron, but they were not large. Consumption was generally large; there were some increases and some decreases. Prices ranged a little lower in 1911 than in 1910, according to best comparison about 3 per cent.

Metallurgical Production—Prices—Consumption

The statistics presented in the preceding tables cover, with but few exceptions, only the production of metals from domestic ore. In addition thereto, the United States produces a large amount of several metals derived from foreign ores, especially copper, lead, spelter, nickel, antimony, and the three precious metals—gold, silver and platinum. Data as to the total metallurgical production will be found in the following pages.

In the following pages also will be found much data as to the domestic consumption of metals and mineral substances. In general this figure is computed from the production plus imports, less exports; but in some important cases it has been possible to take into account the stocks on hand at the begining and end of the year.

In each chapter of this book all available information as to market conditions and prices has been given. For all of the important metals average, monthly or annual prices, are given for a long series of years. These prices are shown on the accompanying diagrams for the years 1910 and 1911 as well as for previous years. Unfortunately, such data are not available for all of the mineral substances. The monthly averages for many of these substances in 1911, not including the metals, are given in the table on a preceding page.

The United States was the second largest producer of gold and of silver; the largest producer of petroleum, lead, zinc and iron, and copper; it produced 60 per cent. of the copper and 60 per cent. of the petroleum, one-third of the lead, one-fourth of the zinc, one-third of the iron, and steel. We consider iron and steel industrially the most important. The commonest metal as far as distribution in the earth's crusts is concerned, is aluminium but thus far it is not obtainable from the silicates commercially.

In order to compare with other industries, it is interesting to note that the mineral production of the United States exceeds $2,000,000,000 and its intimate relation to the development of transportation facilities is shown by the fact that it accounts for nearly two-thirds of the railroad freight.

CHRONOLOGY OF MINING FOR 1911.

The following is a summary of important events recorded by the *Engineering and Mining Journal* during the year 1911. In some cases the dates present the time of publicity rather than of occurrence:

Jan. 3.—Explosion in No. 3 colliery of Nova Scotia Steel & Coal Co., at Sydney, N. S., imprisoned six men.—Fire at Rawhide mine of British Columbia Copper Compony.

Jan. 4.—Strike of 12,000 coal miners in Belgium.

Jan.5.—Stockholders of American Smelting & Refining Co. voted to increase capital stock by $15,000,000, to provide for the conversion of bonds of the American Smelting Securities Company.

Jan. 10.—Standard Oil 10,000-bbl. gusher in Kern oil field, California, took fire.

Jan. 14.—Underground fire broke out in the workings between the Modoc and Butte & Ballaklava mines, at Butte, Mont., two men being smothered.

Jan. 15.—Boston & Montana smeltery at Great Falls, Mont., shut down temporarily on account of ore freezing in bins.

Jan. 16.—Announcement of phosphate discoveries near Butte, Mont.

Jan. 17.—Powder explosion in Keating mine, Broadwater County, Mont., killed six, injured two and wrecked shaft.

Jan. 20.—Forty miners killed in fire in Casimir colliery, Sosnowice, Russian Poland.—Five killed, seven injured by explosion in Gayton mines, about 15 miles west of Richmond, Va.—Executive Council of American Federation of Labor granted charter to Western Federation of Miners on same basis as United Mine Workers of America.

Jan. 25.—Explosion at the Hughestown No. 10 colliery of Pennsylvania Coal Co., at Pittston, Penn., killed six men and seriously injured five more.

Jan. 30.—Arrival at Toronto of first bullion shipment from Porcupine.

Feb. 5.—Entrance of Bewick, Moreing & Co. into the Porcupine district, Ont.

Feb. 6.—Three men killed at the mine of the Independent Coal & Coke Co., at Kenilworth, Utah, by striking Greek miners.

Feb. 9.—Explosion at Cokedale mine of American Smelting & Refining Co., near Trinidad, Colo., killed nine men.—Organization of

international antimony convention at Paris, embracing producers of all countries of the world, the purpose being to regulate production and prices, the agreement to continue for three years.

Feb. 14.—The mill of the Veta Colorado Company, at Parral, commenced crushing.

Feb. 18.—The 40-stamp mill of the Tajo de Dolores mine of the Proprietary Mines Co. began crushing.

Feb. 23.—Fire in the Belmont mine, Tonopah, Nev., resulted in the death of 17 men.

Feb. 26.—Fire discovered in the Hazel mine of the Pittsburgh-Buffalo Coal Co., near Canonsburg, Penn.

March 6.—U. S. Circuit Court at San Francisco denied application for permanent injunction against the Montana smelteries on account of alleged smoke damages.

March 9.—Calumet & Hecla merger approved by the companies that were to go into the consolidation.—Lehigh Coal & Navigation Co. announced plan to build large power plants for transmission of power to be generated at its anthracite mines.

March 11.—Slide of bank at Norman mine of Oliver Iron Mining Co. at Virginia, Minn., killed 14 workmen.

March 13.—Announcement that Amalgamated Copper Company planned to absorb United Metals Selling Co. by purchase at $12,300,000.

March 15.—Miami mill went into operation with one section.

March 18.—Explosion in mine No. 16 of M. K. & T. Ry. Co. at West Mineral, Kan., killed one man, and five men attempting to rescue him were killed.—David H. Moffat, of Denver, died in New York.

March 20.—Ray Consolidated mill started first unit.

March 22.—Nine men killed by fall of roof in Hazel mine, of Pittsburgh-Buffalo Coal Co., at East Canonsburg, Penn.

March 23.—Ray Consolidated and Chino Copper Co. stocks listed on New York Stock Exchange.

March 28.—S. F. Emmons died at Washington.

March 30.—Completion of Copper River & Northwestern R.R. to Kennicott, Alaska.

March 31.—Dr. Rossiter W. Raymond resigned secretaryship of the American Institute of Mining Engineers.

April 1.—Strike of coal miners in Alberta and British Columbia.

April 3.—U. S. Supreme Court upheld commodity clause of the interstate commerce law which prevents a railroad from owning stock in a coal corporation along its lines.

April 6.—First shipment of copper ore from Kennicott, Alaska, via Copper River & Northwestern Railway.

April 7.—Seventy-three men killed in explosion in Pancoast coal mine at Throop, Penn.

April 8.—Explosion at Banner Coal mine, Birmingham district, Ala.; 128 men killed.

April 11.—U. S. Steel Corporation inaugurated its coke-oven plant at Gary, Indiana.

April 19.—Strike of zinc smelters at Bartlesville, Okla.—Renewal of the Chilean Nitrate Association.

April 21.—Cut of 50 cents per ton in Lake iron-ore price.

April 24.—Explosion in Ott mine, No. 20, of Davis Coal & Coke Company, near Elk Garden, W. Va., killed 10 men.

April 27.—Settlement of zinc smelters' strike.

May 5.—Seven men killed by fire in the Hartford mine of Republic Iron & Steel Co. at Negaunee, Mich.

May 8.—Leaching plant of the Warrior Copper Co. in Gila County, Ariz., destroyed by fire.

May 11.—Fire in the Boston colliery of the Delaware & Hudson Company at Larksville, Penn., suffocated five men.

May 12.—The mine tipple and buildings of the Pierson colliery near Jasonville, Ind., were destroyed by fire.

May 15.—Supreme Court ordered dissolution of the Standard Oil Co. of New Jersey.

May 19.—Brush fires at Porcupine destroyed surface plant of Hollinger and other mines in Pearl and Gilies Lake sections.

May 24.—Cut in steel prices started by Republic Iron & Steel Company.

June 1.—Fire burns bunk and boarding house at Nevada Consolidated; loss, $100,000.

June 15.—First payment on Hidden Creek mine, British Columbia, made by Granby company.

June 20.—Reopening of the St. Louis mine in the Lake Superior copper country started by Calumet& Hecla Company.

June 21.—Dissolution of the E. I. du Pont de Nemours Powder Co. ordered because of violation of the Sherman anti-trust act.

June 26.—Denial of injunction asked by G. M. Hyams against proposed Calumet & Hecla merger.

June 26.—Cunningham coal claims cancelled by the General Land Office.

June 28.—Dismissal of suit brought by minority stockholders against Greene Consolidated Copper Co., the Greene-Cananea Copper Co. and their officers and directors.

June 29.—Eighty-four wire officials and employees indicted for illegal restraint of trade.

July 1.—Opening of railway into Porcupine.

July 5.—International meeting of representatives of the steel industry at Brussels, Belgium.

July 11.—Forest fires in Porcupine district resulted in great loss of life and property.—Coniagas smeltery and coke building at Thorold, Ont., destroyed by fire.

July 15.—Explosion in mine of Cascade Coal & Coke Co., at Sykesville, Penn., killed 21 men.

July 17.—Government filed suit to compel separation of Reading railroad from Reading Coal Company.

July 21.—Government filed suit against American Smelting & Refining Co. for $2,500,000 damages, in connection with Colorado coal lands.

July 22.—Dismissal of charges against American Smelting & Refining Co., alleging a monopoly in restraint of trade.—Cessation of smelting at Balaklala works, owing to smoke trouble—Shafthouse and timbering No. 11, Calumet & Hecla, destroyed by fire.

July 25.—Announcement of sale of Gold Road mine in Arizona to United States Smelting, Refining & Mining Co. for $1,600,000.

July 29.—Strike of zinc smelters at Caney and Deering, Kan., and suspension of operations at those smelteries.

July 31.—Temporary injunction granted to minority stockholders of Osceola Consolidated Mining Co., in Calumet & Hecla merger.

Aug. 1.—Explosion in mine of Standard Pocahontas Coal Co. at Shannon, W. Va., killed three men.

Aug. 4.—Suit filed by Government at Columbus, Ohio, against six railroad companies and three coal-mining companies, charging combination in restraint of trade.

Aug. 10.—A four-deck cage dropped in the Kupp-Hannibal mine near Bochum, Germany, killing 25 men.

Aug. 14.—Announcement of purchase by the American Smelting & Refining Co. of the Tiro General mines at Los Charcos, San Luis Potosi, Mex., and of the Central Potosi R.R., for $800,000 gold.

Aug. 15.—Granby smeltery in Boundary district, British Columbia, closed for lack of fuel due to strike.

Aug. 17.—Collapse of strike at Caney and Deering, Kansas.

Aug. 19.—Cave-in at the openpit of the Buffalo & Susquehanna mine on Mesabi range, buried 12 men and caused several fatalities.

Aug. 22.—Contract executed for sale of Ray Consolidated Smeltery to American Smelting & Refining Company.

Aug. 23.—Explosion and fire at new shaft of Giroux Consolidated Company resulted in seven deaths and damage to shaft.

Aug. 30.—Confirmation of sale of Cleveland mine in Idaho to Federal Mining & Smelting Co.—Case of Fred J. Bliss against Anaconda Copper Co., based on smeltery fume troubles, reached United States Supreme Court.

Sept. 1.—Dissolution of the International Antimony Syndicate, it having proved a fiasco.

Sept. 2.—Hoisting accident at Calumet & Hecla mine destroyed hoist and engine-house.

Sept. 3.—Six men killed in a hoisting accident at Butte & Superior mine.—Fire caused total destruction of Hall smeltery at Nelson, B. C.

Sept. 4.—Two miners were killed and several injured in hoisting accident at Red Jacket shaft of Calumet & Hecla mine.—Two men were killed in hoisting accident at Daly West mine.

Sept. 5.—Second suit filed against Mason Valley Mines Co., to prevent operation of smeltery.—Butte & Ballaklava Copper Co. resumed ore shipments after a year's suspension due to litigation.

Sept. 16.—Three men, imprisoned by a cave-in in Morning Star mine, Colorado, were rescued alive.

Sept. 23.—Inauguration of strike at Mt. Lyell copper mines in Tasmania.

Sept. 30.—Superior & Boston stockholders voted favorably on plan of consolidation with the Arizona Commercial Copper Company.

Oct. 2.—Announcement of plan for reorganization of Arizona Commercial Copper Company.

Oct. 6.—Calumet & Hecla Company announced the abandonment of the merger plans.

Oct. 10.—Opening of the 101st meeting of the American Institute of Mining Engineers in San Francisco.

Oct. 18.—Explosion in Bardot coal mine, France, killed 26 men.

Oct. 19.—Raritan Copper Works instituted suit against the Government for recovery of platinum contained in gold delivered to New York Assay Office.

Oct. 20.—New Langdon shaft of Wharton Steel Company at Hibernia, N. J., flooded and 12 men drowned.

Oct. 21.—First section of Chino mill placed in operation.—Explosion in a mine at Trabonella, Italy, reported to have killed or injured 100 men.

Oct. 22.—Fire damaged Kingdon shaft of Old Dominion Company at Globe, Arizona.

Oct. 23.—Explosion in O'Gara mine No. 9, at Harrisburg, Ill., killed eight and injured ten men.

Oct. 24.—The 14th annual session of American Mining Congress opened in Chicago.—Fire destroyed Butte Reduction Works at Butte, Mont., entailing $350,000 loss, one-half insured.

Oct. 26.—Steel Corporation officially announced intention to cancel Great Northern ore lease and to reduce freight rates 20 cents per ton on its Minnesota range roads.—Government filed suit in Trenton, N. J., against Steel Corporation, its subsidiaries and officers, under Sherman anti-trust law.

Nov. 3.—Waldemar Lindgren was appointed chief geologist of the U. S. Geological Survey.

Nov. 9.—Explosion in Adrian mine, of Rochester & Pittsburg Coal & Iron Co., at Punxsutawney, Penn., killed eight men.

Nov. 11.—Cripple Creek deep-drainage tunnel encountered a flow of water which resulted in draining the lower levels of several important mines.—Settlement of strike at Mt. Lyell copper mines in Tasmania.

Nov. 15.—Freight reductions went into effect between Salt Lake City and Missouri River points.

Nov. 16.—District court, at Colorado Springs, Colo., issued temporary injunction restraining the sale of 11 Cripple Creek properties for taxes.

Nov. 17.—Canadian coal strike settled by agreement signed at Hosmer, B. C., under which the miners returned to work on Nov. 20.

Nov. 18.—Explosion in mine of Bottom Creek Coal & Coke Co., at Vivian, W. Va., killed 18 men.

Nov. 22.—Announcement that Consolidated Gold Fields of South Africa had acquired extensive interests in Alaskan dredging lands, through the operations of the Granville Mining Company.

Nov. 23.—Bankruptcy proceedings were commenced in the United States Court, in New York, against the Neveda-Utah Mines & Smelters Corporation.

Nov. 24.—John D. Ryan purchased control of Live Oak Development Co. at $30 per share, preliminary to merger of porphyry-copper properties in the Miami district, Arizona.

Nov. 25.—Governor Johnson appointed William H. Storms as state mineralogist of California.

Dec. 1.—Fifth unit of Miami concentrator started.—Standard mill of United States Reduction & Refining Co., at Colorado City, Colo., closed.

Dec. 4.—Mill of Florence-Goldfield Company, at Goldfield, Nev., destroyed by fire, entailing a $400,000 loss.

Dec. 10.—Explosion in coal mine of Knoxville Iron Co., at Briceville, Tenn., killed 84 men.

Dec. 19.—Announcement of formation of Inspiration Consolidated

Copper Co., to include the Live Oak and Inspiration Companies.—Litigation in connection with the attempted Calumet & Hecla merger was officially dismissed.—Directors of the Butte Coalition Mining Co., voted to dissolve the corporation and to distribute its assets.

Dec. 20.—Arrangements completed to distribute the assets of the Alice Gold & Silver Mining Company.

Dec. 21.—Production was resumed by the Granby Company in British Columbia.

Dec. 22.—Arrangements completed to distribute the assets of the Parrot Silver & Copper Company.

Dec. 23.—The Cross Tetley coal mine at Wigan, England, was flooded, 200 miners escaping with difficulty.

Dec. 28.—Fire destroyed a portion of Golden City, in the Porcupine district, Ontario.

Dec. 29.—Allis-Chalmers Company announced its intention to default interest due Jan. 1, 1912, on its bonds.

Dec. 31.—The firm of Wernher, Beit & Co., of London and South Africa, was dissolved by the mutual consent, the diamond branch being transferred to the firm of L. Breitmeyer & Company.

ALUMINIUM

By Jos. W. Richards

There are now 20 reduction plants producing aluminium by the Hall Heroult process, belonging to 13 different companies. They are, with the estimated maximum yearly capacity of each, in metric tons of 2204 lb. as follows:

Firm.	Locality.	Capacity. Tons.
Aluminum Company of America	Niagara Falls, N. Y., U. S. A.	
	Upper works	1,000
	Lower works	8,000
	Massena, N. Y., U. S. A.	4,000
Northern Aluminium Co	Shawenegan Falls, Canada	4,000
Aluminium Industrie Aktien Gesellschafft	Neuhausen, Switzerland	3,500
	Lend Gastein, Austria	4,000
	Rheinfelden, Germany	4,000
	Chippis, Switzerland	4,000
British Aluminium Co	Foyers, Scotland	5,000
	Loch Levan, Scotland	5,000
A/S Stangfjord Elektro-chemiske Fabbriker	Stangfjord, Norway	500
A/S Vigelands-Brug (Anglo-Norwegian Aluminium Company.)	Vennisla, Norway	2,250
Gebruder Guilini	Ludwigshafen, Switzerland	600
Société Electrometallurgique Francaise	Froges, France	8,000
Compagnie des Produits Chimiques d'Alais	Salindres, France	3,000
Soc. des Forces Motrices et Vsines de l'Arve	Chedde, Savoy, France	1,500
Soc. des Produits Electrochimiques des Pyrenees.	Auzat, Ariege, France	2,250
Soc. Electrometallurgique du Sud-Est	France	500
Société d'Electrochimie	Savoy, France	500
Soc. Italiana per la Fab. dell'Aluminio	Bussi, Italy	600

Summary of Annual Capacity.

France	15,750 tons
United States	13,000 tons
Scotland	10,000 tons
Switzerland	8,000 tons
Germany	4,000 tons
Austria	4,000 tons
Canada	4,000 tons
Norway	2,750 tons
Italy	600 tons
Total	62,100 tons
Estimated Product in 1911,	45,000 tons

As to the amounts produced in 1911, none of the works or companies will give any official figures. A reasonable estimate is that the output attained 75 per cent. of the maximum capacity. In 1910 it averaged 70 per cent., and as business was brisker in 1911, we estimate the world output at 75 per cent. of the capacity, or 45,000 metric tons. Following

are some notes upon the condition of the aluminium industry in 1911 in different countries.

AMERICA

During the year 1911 the condition of the aluminium industry in America was generally steady. In the early part of the year it still felt seriously the effects of the business depression of the previous years but the conditions improved gradually throughout the year. Low prices and heavy importations were a particularly prominent feature of the year. The prices on large quantities were generally lower than for any preceding year.

PRODUCTION, IMPORTS AND CONSUMPTION OF ALUMINIUM IN THE UNITED STATES.

	Production.			Imports.			Exports.	Consumption.
				Crude.		Mfrs.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Value.	Value.	Value.
1898..	$5,200,000	$1,690,000	$0.33	60	$ 30	13,840	$238,997	1,474,268
1899..	6,500,000	2,112,500	0.33	53,622	9,425	7,828	291,515	1,838,238
1900..	7,150,000	2,288,000	0.32	256,559	44,455	5,989	281,821	2,056,623
1901..	7,150,000	2,238,000	0.31	564,803	104,168	5,580	183,579	2,164,169
1902..	7,300,000	2,284,590	0.31	745,217	215,032	3,819	116,052	2,387,389
1903..	7,500,000	2,325,000	0.31	498,655	139,298	4,273	157,187	2,311,384
1904..	7,700,000	2,233,000	0.29	515,416	128,350	478	166,876	2,494,952
1905..	11,350,000	3,632,000	0.32	530,429	106,108	33	290,777	3,015,364
1906..	14,350,000	5,166,000	0.36	770,713	154,292	1,866	364,251	4,957,907
1907..	26,000,000	10,920,000	0.42	872,474	181,351	1,124	304,938	(b)
1908..	13,000,000	4,095,000	0.315	465,317	80,268	2,334	330,092	(b)
1909..	15,000,000	3,345,000	0.223	5,109,843	745,963	12,878	567,375	(b)
1910.	12,000,000	2,736,000	0.228	12,271,277	1,844,830	.....	949,215	(b)
1911..	(c)28,600,000	5,720,000	0.20	4,173,308	598,272	(d)63,899	1,158,603	(b)

(a) Not reported. (b) Impossible to compute accurately in the absence of information as to unsold stocks. (d) from July 1 only. (e) estimated.

PRICE OF ALUMINIUM AT NEW YORK.

(In cents per pound.)

Grade.	Dec., 1907.	July, 1908.	Dec., 1908.	July, 1909.	Dec., 1909.	July, 1910.	Dec., 1910.	July, 1911.	Dec., 1911.
99% pure.....	33	28	24	22	22	22½	23	20½	18¾
90% pure.....	32	27	23	21	21	21	21	.....	.....
No. 12 casting alloy....	32	27	22	20	20	.....	.....	.....	.....
No. 21 casting alloy....	30½	24½	22½	20½	20½	.....	.....	.....	.....
No. 31 casting alloy....	28½	22½	21½	19½	19½	.....	.....	.....	.....

The above prices were for ton lots or over; the prices for small lots were 2 to 3 cents per lb. higher.

WORLD'S PRODUCTION OF ALUMINIUM.
(In metric tons.)

Year.	Great Britain.	France.	Switzerland, Germany, Austria.	North America.	Totals.
1898.....	310	565	810	2,359	4,034
1899.....	559	763	1,300	2,949	5,571
1900.....	569	1,026	2,500	3,244	7,339
1901.....	560	1,200	2,500	3,244	7,504
1902.....	600	1,355	2,500	3,312	7,767
1903.....	(b) 650	1,570	(c)2,500	3,403	8,123
1904.....	(b) 650	1,650	(d)3,000	3,494	8,794
1905.....	2,250	4,425	3,675	6,560	16,810
1906.....	2,500	4,500	4,000	7,325	18,325
1907.....	3,700	4,700	8,000	16,329	32,529
1908.....	(b)2,000	4,681	(b)3,500	8,150	18,931
1909.....	(b)2,800	6,092	(b)5,000	(b)9,000	24,292
1910.....	(e)2,000	(e)6,000	(f)6,000	9,000	(e)23,000
1911.....	(e)7,300	11,300	(f)11,700	14,400	44,700

(a) C Le Neve Foster British Mineral Statistics for 1897. (b) Statistics of Metallgesllschaft, Frankfort am Main. (c) Includes 600 tons produced in Italy. (d) Includes 800 tons produced in Italy and 600 tons produced in Norway. (e) Estimated. (f) Estimated, including production of Italy and Norway.

No important additions were made to any of the plants of the Aluminium Company of America during the year. The increased output provided for in the years previous had been more than ample to meet the requirements of the trade. A small addition has been built to the New Kensington, Penna., plant to take care of the growing trade in cooking utensils and general manufactured articles.

The bronze powder works at Dover, N. J., has been running steadily during the year. The material produced at this plant is of a very high quality and the establishment of this domestic supply has apparently been warranted by the ready consumption of the product.

The continued small demand for steel has had a corresponding effect upon the amount of aluminium used for deoxidizing purposes.

The automobile industry has continued to be a large consumer of aluminium. This industry has continued to grow, and although the cars are sold at a lower average price each year, even the lowest priced cars use aluminium. It is now very gratifying to the manufacturers of aluminium that after a number of years of experience, despite the efforts of the motor-car makers to find a substitute for aluminium possessing better qualities than that metal, they have almost universally adopted it in those places for which it is suited. The campaign for a cheaper substitute is gradually closing; in fact, aluminium proves to be the cheapest metal in view of the ease and speed with which the castings are machined, and the ease with which sheet and wrought forms are applied. Manufacturers of motor vehicles seem to have generally come to the conclusion that aluminium is the most satisfactory metal to use in motor-car construction. It is now used in the engine construction, body construction, foot boards, running boards, and for a number of other similar parts.

The general revival of business during the year has had an influence upon construction work throughout the country. This improvement in business has had an immediate effect upon electrical power developments and electric railway construction. As a result, there has been a considerable tonnage of aluminium high-tension electrical conductors produced during the year 1911. Most of this material has gone to the southern and western sections of the United States. The largest installations of aluminium conductors have been made in North Carolina, South Carolina, Virginia, Georgia, California, Nevada, Oregon and Colorado. The numerous aluminium lines already constructed continue to give uniformly satisfactory service.

There have been very few important new uses developed for the metal during the year. There has been, however, one rather important development in regard to a new use, and this is in connection with the use of aluminium in breweries. It is too early to predict positively just the extent to which aluminium will be adopted in breweries, but from the rather extensive experiments which have already been conducted it seems that the brewing industry will be a very promising field for the future application of aluminium. One thing which has contributed materially to the possibility of this use is the improvement in welding. By means of autogenous welding, aluminium sheet can be produced in a variety of forms wholly impossible by the cruder method of riveting. The joint produced by autogenous welding, after suitable finishing, makes a practically homogeneous article. This is especially advantageous for those industries which must use vessels capable of being kept in the most sanitary condition. One seamless 600-barrel tank has been produced for a brewing concern in this country, and besides this one large vessel several others of a smaller capacity. The freedom from joints, sanitary nature of the metal, and the fact that it does not affect brewery products, particularly recommend aluminium for this industry.

The uses for plain aluminium sheet have not increased in number, although those concerns already using it have perhaps widened their scope. One large use of aluminium sheet is in vehicle-body construction. The gradual increase in price of suitable wood for this purpose, and the evident disadvantages of iron, have rendered aluminium one of the most desirable materials in body construction.

The use of extruded shapes has increased, particularly in the form of mould ng for vehicle-body construction and in numerous other forms for special purposes.

Aluminium bottle seals of a very satisfactory character are now in the market and are being widely used by fruit preservers and breweries.

The popularity of aluminium cooking utensils has created a universal

demand for these articles, and as a result several of the large manufacturers who hitherto confined their efforts to enamel, copper and iron ware, are forced to put in a line of aluminium cooking utensils. This seems to justify the claims of the original makers of aluminium cooking utensils, that no metal is superior to it for this use. The aluminium cooking utensil industry has had a constant growth throughout the year. It seems probable that if the rate of growth continues, aluminium cooking utensils will, in a few years, be essentially the standard utensil to which all others will be compared. At the present time the consumption of aluminium for this purpose amounts to a very substantial tonnage, and is rapidly increasing.

Of minor interest is the discovery at Marysvale, Utah, of a deposit of native alum, or mineralogically, of alunite. At present, all aluminium is produced from purified bauxite, a hydrous aluminium oxide, but if the alunite is as abundant as claimed, it may also figure as a source of the raw material. A vein some 10 ft. wide and traced over half a mile has been found, which may contain several hundred thousand tons of material. The mineral is a hydrous double sulphate of aluminium and potassium, which by light calcination is reduced to a mixture of potassium sulphate and alumina; the former can be washed out by water, giving the valuable potash salt in solution, while impure alumina remains.

The usual new American company has been announced in the newspapers. *Mining Science* of December 2, 1911, reports from Chattanooga, Tenn., that the American Aluminium Company has purchased 40 miles of territory on the banks of the Little Tennessee river, and is to spend $10,000,000 in development and works, and employ 1000 men. There have also been speculations as to the utilization of water power in Oregon for the same purpose, but nothing definite has crystallized from these propositions, as far as is known to the writer. The production of aluminium cheaply enough to compete at present market conditions, means such a large investment of capital, for alumina purification plant and carbon anode plant, as well as for reduction works, that a small enterprise could not exist, and a large one would have to compete actively with an already well-organized and managed company.

France

Following are some data respecting the French aluminium works, given by H. Grossmann in the *Zeitschrift für Elektrochemie*, March 1, 1912, p. 162.

The Compagnie des Produits Chimiques d'Alais et de la Camargue, founded by Henry Merle et Cie in 1865, later A. R. Pechiney et Cie, has been since 1896 a stock company with a capital of $2,000,000. Central

Works at Salindres (Gard), office at No. 12 Rue Roquepine, Paris. This firm began in 1861 to manufacture aluminium by the Sodium process of Henry Saint-Claire Deville, and for 28 years monopolized the business. In 1896 it took up the electrolytic process at the Calypso Works at Saint Michel de Maurienne. At present it also possesses works at Saint Felix de Maurienne and St. Jean de Maurienne. This company also manufactures fertilizers by the process of Serpek, which produces aluminium nitride, from which is produced alumina and ammonia. The success of this process is not yet assured.

The Société Electrometallurgique Francaise is the oldest French Electro-Metallurgical works, being founded in Froges (Isere) in 1888, to manufacture aluminium by the Heroult Process. Only 100 hp. was available; in 1894 a second works was erected at La Praz near Modane, in the neighborhood of Mt. Cenis, and an aluminium works was also erected at Gardanne (Bouches-du-Rhon). In 1903 this firm erected an aluminium works at Saint Michel-de-Maurienne, where 17,000 hp. is available. The third works was later built in Argentiere, in the Valley of Durance, near Briancon, where 35,000–40,000 hp. is available. This firm therefore uses 65,000–75,000 hp. for the larger part of the year for the manufacture of aluminium; a part is used, however, for the manufacture of aluminium alloys, electric steel, etc. The company has a capital of $3,000,000.

The Société des Forces Motrices et Usines de l'Arve, founded in 1895 with a capital of $505,000, increased in 1902 to $800,000, manufactures aluminium and aluminium alloys, chlorates and perchlorates. The waterpower available at Chedde (Upper Savoy) can produce 1500 tons of aluminium yearly.

The Société des Produits Electro-Chimique et Metallurgique des Pyrenees, founded in 1906 with a capital of $1,200,000, has an aluminium factory at Auzat (Ariege) at the foot of Montcalm, utilizing on an average 10,000–12,000 h. p. from the Viodessos, a branch of the Arieges, the fall being 420 meters. The reservoirs for storing water in winter hold eight to nine million tons of water.

United States Consul D. I. Murphy of Lyons, estimates the output of Bauxite in France as over 200,000 tons annually, and that the six plants now running in the Savoy, with two others in the Pyrenees have an estimated annual capacity of over 25,000 tons.

GERMANY

Aluminium has been regularly sold in Germany as low as 1.35 marks per kilogram, or 12 1/4 cents per pound. Germany imports an average of 10,000 tons of aluminium yearly, mostly from France and Switzerland.

Italy

The works of the Société Italiana de Elettrochemica at Bussi obtains its bauxite from Pescina, producing from it pure alumina by the Baeyer process, calcining it before use at 1000° C. Four dynamos give each a current of 7000 amperes at 100 volts, working 14 furnaces in series (average drop 7 volts per furnace). The 56 furnaces furnish 55 metric tons (121,000 lb.) of aluminium per month. Each pot contains about one-half ton of melted cryolite, takes 7000 amperes at 7 volts, or 65 hp., and produces 33 kilos of aluminium, at a current-ampere efficiency of 60 per cent. The carbon anodes are rectangular blocks, made at the works from coke, graphite dust and tar, and pressed under 300 atmospheres pressure.

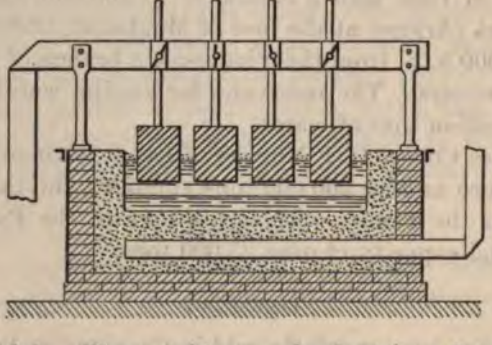
Switzerland

The Aluminum Industrie Actien Gesellschaft, at Neuhausen is also running plants in Rheinfelden, Germany, Lend-gastein, Austria, and at Chippic, near Loesch, in the valley of the Rhone, Switzerland. It has paid over 25 per cent. yearly dividend on its capital.

MANUFACTURE OF ALUMINIUM

G. Flusin of Grenoble contributes a long article upon the present methods of manufacturing aluminium to *La Houille Blanche*, October–November, 1911, from which we abstract the following items of new information:

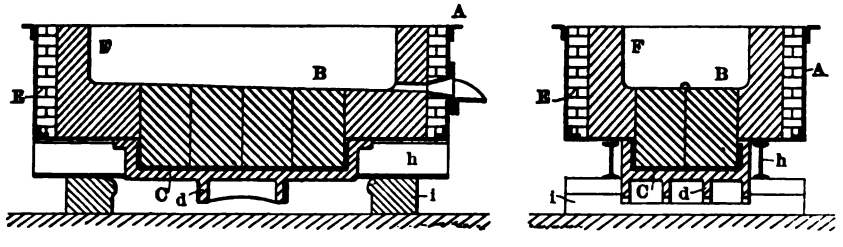
Reduction Pots.—In one of the large works of Europe, the dimensions of the pots are, 2.4 meters long, 1.2 to 1.5 meters wide, and 0.60



to 0.70 meters high. The outside case is sheet iron 8 to 10 mm. thick, strengthened with angle iron, and is placed either on bricks resting on the ground or upon iron girders. At some works the boxes rest at one

end upon screw-jacks, which enables one end to be lifted for emptying the bath. At one works the carbon lining is stamped directly into the brick-work. The inner cavity is about 30 cm. (12 in.) deep. Contact with the cathodic current conductor is made from the carbon bottom. One method uses an iron grating, embedded in the bottom with warm carbon paste. Another arrangement uses moulded carbon blocks as the bottom for the bath, set in an iron clamp into which they are keyed by set screws, serving as the current conductors; or these blocks are screwed together by iron bolts.

Anodes.—These are now preferably made in nearly cubical blocks 25 or 30 cm. (10 in. to 12 in.) high, and 30 to 50 cm. (12 in. to 20 in.) long. Their weight varies between 30 and 70 kilogrm. (66 to 154 lb.). For connection a split iron claw is embedded in the blocks upon burning, or a block is embedded in the anode having a screw thread, into which screw connection can be made. The cost of these electrodes as given



is a trifle over three cents per pound in Europe. Eight to twelve such anodes are hung in the bath; in some cases each anode being connected with a mechanical holder to adjust it separately. The current is from 8000 amperes with small baths up to 20,000 amperes in the largest.

Mr. William Clacher gives in detail the manufacture of carbon electrodes as carried out at the works of the British Aluminium Company. (*London Electrical Review*, Jan. 20, 1911; *Met. and Chem. Eng.*, March, 1911.) Interesting points are that the petroleum coke used is first calcined five hours in vertical kilns at approximately 2000° C., to remove volatile matter and increase density and conductivity. To expel 8 per cent. of volatile matter, 22 per cent. of the coke itself is consumed; a very expensive operation, since the material is worth $17.50 per ton. The calcine contains 0.5 per cent. SiO_2, 0.15 per cent. $Fe_2O_3 + Al_2O_3$, and 0.05 per cent. soluble sodium salts. This is ground, mixed with 30 per cent. of its weight of pitch, and pressed into shape. When calcined at a low temperature (as pot linings), only two-thirds of the pitch used is volatilized; when the electrodes are baked, however, in furnaces heated by

producer gas up to 1400°, for five one-half days, the volatile matt nearly all expelled.

The electrode are heavy rectangular blocks weighing 66 lb. which 57 1/2 is carbon and 8 1/2 an iron claw, moulded in, which se for electrical connection. Such electrode blocks last normally 10 140 hours in producing 140 to 200 lb. of aluminium.

Electrolytes.—The works using the Heroult Process, used first a b of cryolite alone, or with a small addition of fluorspar or sodium chlor with the alumina added between 10 and 20 per cent. The Hall bat said to be composed on the formula $Al_2F_6.6NaF.3CaF_2$, or in percenta

Cryolite	36 parts
Aluminium fluorid	44 parts
Calcium fluorid	20 parts

The French works are said to have reduced the aluminium fluoride be this, using a bath of the formula $Al_2F_6.3NaF$. The melting point of Hall bath is given as 800 to 850°, with the addition of sodium chlo it may be as low as 700°. The common salt, however, volatilizes quic so that it is not used continuously but added to bring back into reg running a bath not working well. The normal temperature of the bath 800 to 900°. The current density at the anode is 70 to 80 amperes square decimeter (65 to 75 per square feet) with a potential of seve eight volts.

Operation.—It is recommended to place small pieces of carbon tween the anode and the bottom of the bath, which serve as resista pieces and melt quickly the bath material piled around them. As s as the salt is melted more is added until the required quantity of bat obtained, the carbon blocks are then removed, alumina is added dissolves, and the decomposition commences. It takes about two d to get the bath to normal operation; this can be hastened by the addit of common salt. The top of the bath is covered with alumina, and a sm incandescent lamp, connected across from anode to cathode, shows by increased brilliancy any rise in potential across the bath. Alumin stirred into the bath when necessary, and fresh alumina placed on t Cryolite and fluorspar are only added in small quantities at several d interval. One workman is required for three to four baths, and alum ium is tapped from the baths when the casts are made, every two or th days. The anodes are from 6 to 8 cm. (2 1/2 in. to 3 in. from the cathod Instead of tapping the bath, one works uses a carbon crucible wit hole in the bottom closed by a carbon plug. This is pressed down in bath until it touches the aluminium beneath. When full the stoppe replaced and the crucible fill of metal lifted out; another device is ladle the metal out of the crucible with a small ladle. The normal curr

output is 90 to 95 per cent., but in many works the output is not over 60 to 65 per cent., corresponding to an output of 210 to 275 kilogrm. (461 to 605 lb.) per kilowatt year, or 154 to 202 kilogrm. (339 to 444 lb. per horse power year.) The anode consumption is given 60 to 65 per cent. of the weight of aluminium produced which is increased sometimes to 70 or 100 per cent. The consumption of cryolite is given as 12 to 20 per cent. of the aluminium produced, and a similar amount of calcium fluorid, but these figures are far too high.

H. K. Richardson (*Met. and Chem. Eng.*, May, 1911) describes the laboratory directions given students of State College, Pa. The bath contains 85 per cent. natural cryolite and 15 per cent. alumina; current density three amperes per square centimeter of anode used; add alumina to bath every 30 minutes. Current efficiency obtained at 1000° C., 60 to 70 per cent; at very low temperatures up to 90 per cent.; energy efficiency 20 to 25 per cent.; the rest being used to keep the bath melted.

The cost of production in Europe is given by Mr. Clacher as follows:

Electrical energy	1.25 cents per pound
Carbon electrodes	1.63 cents per pound
Alumina	7 cents per pound
General expenses	4 cents per pound
	13.88 cents per pound

Miscellaneous Propositions: U. S. Patent 961,913 claims to be able to produce aluminium carbide by reduction of alumina in an electric furnace, by the proper quantity of carbon, and then the double reaction of this carbide upon a fresh quanitity of alumina in another electric furnace. The reaction desired would be:

$$2Al_2O_3 + 9C = Al_3C_4 + 6CO$$
$$Al_4C_3 + Al_2O_3 = 6Al + 3CO$$

L. R. Keogh (*U. S. Patent* 996,094, June 27, 1911; *Met. and Chem. Eng.*, Oct. 1911) will feed into a bath of fused NaCl, in an electric furnace, a mixture of anhydrous aluminium sulphate and carbon. The mixture is supposed to form Al_2S_3, which obligingly dissolves in the bath and is electrolyzed by a very properly regulated current. Anyone who believes this will not work, should try it for himself.

Alloys

W. F. Mellen produces homogeneous alloys of aluminium with other metals by first melting the metals together with the aluminium in slight excess of the ultimately desired proportion, then adding an oxidizing flux, such as borax and potassium bichromate (1/2 lb. bichromate and 1/4 lb. borax per 100 lb. of alloy). This oxidizes some of the aluminium, pro-

ducing great heat and violent agitation, by which the ingredients of the alloy become much more intimately combined and the alloy cleaned. (*Met. and Chem. Eng.*, Apr., 1911; *U. S. Patent*, 982,218, Jan., 17, 1911.)

Alloys with Zinc.—Rosenhain and Archbutt described to the British Institute of Metals (*Met. and Chem. Eng.*, Oct., 1911) their study of these alloys in the liquid state. They showed some compounds, such as Al_2Zn_3, to exist at high temperatures and to break down into others at lower, giving off zinc crystals. The study is of scientific interest but not of great practical importance.

A. Portevin in *Revue de Metallurgie*, September, 1911, page 721 to 727, gives the result of his researches of the tensile and compressive strength of aluminium-zinc alloys. He reports tests upon the alloys cast in sand and alloys forged. The principal results are as follows:

Per cent. zinc.	Tensile strength lb. per square inch.	Relative hardness.
0.10	8,700	18.5
0.20	16,700	40
31.6	21,400	81
40.6	26,800	85
	32,300	85

With higher percentages of zinc the tensile strength and hardness decrease. It was more difficult to cast solid bars of this alloy in sand than in chill molds. Alloys with 30 per cent and over of zinc cast more easily.

The forged alloy was first cast in chill molds into bars 2 in. diameter, which were forged at about 350° Centigrade to 0.8 in. diameter. The effect of forging is marked, it increases both the tensile strength and elongation before breaking, the latter increasing ten times. Annealing after forging diminished slightly the strength but increases the elongation and reduces the elastic limit. The principal results are as follows:

Zinc Per Cent.	Treatment.	Tensile Strength.	Elastic Limit.	Per Cent. Elongation.	Contraction of Area.
5.3	Cast.	11,300	6,000	8.5	23.8
	Forged	19,400	16,400	19	63.7
	Annealed	13,700	3,600	30	74
10.2	Cast	13,300	9,300	2.5	11
	Forged	26,000	15,300	33.5	55.4
	Annealed	21,100	6,400	38	64.5
16.0	Cast	24,400	14,900	2	4.2
	Forged	30,300	25,900	23	47.4
	Annealed	33,100	10,100	28	41
21.0	Cast	26,200	24,400	1	3.9
	Forged	44,700	32,000	14	27
	Annealed	45,000	30,600	14.5	36.5

These figures show very strikingly the great advantage of forging on the mechanical qualities of this alloy. Some of these alloys have been used in aeroplanes in small cast pieces. It appears that forged alloys, particularly the 21 per cent. zinc alloy, may have considerable use in such machinery because of its high tensile strength and elastic limit.

Alloys with Magnesium.—The alloys with 3 to 10 per cent. magnesium, magnalium, are said to wear well as gasoline engine cylinders, the inside where in contact with the iron piston and piston rings taking a high mirror-like polish and wearing even better than iron, at a piston speed averaging 1000 ft. per minute. Further, the heat conductivity being seven to eight times that of iron, heat is rapidly conducted, reducing greatly the temperature of the inside surface and makes cooling easier, particularly in air-cooled engines. (Practical Engineer, Feb. 1, 1912.)

Engineer A. Wilm, inventor of the alloys known as "Duralumin" describes in *Metallurgie* for April 22, 1911, page 225, their characteristics. He states that, working upon this alloy since 1903 he has determined that alloys containing a small percentage of magnesium may be hardened and strengthened by heat treatment. This hardening process is quite different from the hardening of steel, for the alloys are stored directly after quenching, and a few hours afterward the hardness commences to increase. Even 0.5 per cent. magnesium acts in this manner, but with magnesium alone the maximum hardness obtained is less than that of bronze. The presence of copper with the magnesium increases very strikingly the resulting hardness and strength. The magnesium-copper-aluminium alloy, quenched from about 500° C. in water, has a homogenous structure, and increases materially in strength for several days.

The heat treatment adopted by Mr. Wilm is applied to alloys containing 0.5 per cent. magnesium and 3.5 per cent. copper. These are heated to 500°, quenched in water, and laid aside for several days. By this treatment not only is the hardness increased but also the tensile strength and elongation. The best results given by Mr. Wilm for these alloys are from 37,000 lb. per square inch tensile strength and 22 per cent. elongation, to 77,000 lb. tensile strength and 2.3 per cent. elongation.

These alloys thus treated are sold under the name of "Duralumin," in Germany, by the Durener Metallwerken und Aktien Gesellchaft, in England by the firm Vickers Sons and Maxim, in Austria by the firm Messingwerk C. Kulniz, Kramsach-Achenrain (Tyrol), and in America by the Duralmin Company in New York City.

Following is an analysis of the alloy sold as duralumin:

Aluminium	94.60
Copper	3.90
Iron	0.45
Magnesium	0.75
Silicon	0.21
	99.91

Tensile strength 39 tons, elastic limit 38 tons, elongation 3.5 per cent., reduction in area 10 per cent. Specific gravity 2.8.

Alloys with Copper, Etc.—The French Company "Le Ferro Nickel" of Paris patents in Germany (German Patent 231,060) the alloy consisting of:

Aluminium	95 to 98
Copper	1.5 to 4
Manganese	0.25 to 1.25
Silver	0.25 to 1.25

In each case the amount of alloying metals is between 2 per cent. and 6 per cent. The alloys are said to be more valuable than pure aluminium and finer in color; they are ductile, malleable, and can be forged or rolled, hot or cold. They can also be turned, minted, engraved, drawn into wire, or worked in any way similar to brass.

C. H. H. Claessen patents the alloy with 4 per cent. copper and 0.25 to 0.50 per cent. magnesium. (*U. S. Patent* 995,113, June 13, 1911; *Met. and Chem. Eng.*, Oct. 1911.)

It is said that 7 to 8.5 per cent. of copper, 4.5 to 5.5 per cent. of tin, and the rest aluminium, gives a strong, light, lustrous alloy which casts and finishes well. (*E. and M. J.*, May 13, 1911.)

P. H. G. Darville adds 0.05 to 5 per cent. of manganese to the 6 to 11 per cent. aluminium-copper bronzes. The alloys are said to be easy to cast sound, and to be worked. (*U. S. Patent* 1,007,548, Oct. 31, 1911; *Met. and Chem. Eng.*, Dec., 1911.)

Alloys with Cobalt.—H. Schirmeister, in Metallurgie for October 22, 1911, publishes a long article on Al—Co alloys. Groyer had previously shown (*Z. anorg. Chemie*, **57**, 140 (1908)) that these metals are mutually soluble in each other. The alloys were made by melting aluminium in a graphite crucible, and stirring upon it the necessary amount of cobalt oxide; the reaction was prompt, the alumina slag was easily removed, and the alloy was cast into pre-heated sand molds. Alloys with 3.8, 6.4, 8.8, 12.6 and 19.5 per cent. of cobalt showed no particularly useful properties. With 11.6 per cent. cobalt and 0.94 per cent. tungsten, the alloy was dense, fine-grained, forgeable, and about twice as strong as pure aluminium; similar results were obtained with 8.8 per cent. cobalt and 0.6 to 1.1 per cent. of molybdenum.

Working and Various Uses

Electrical Conductors.—At the International Electrical Congress in Turin, in 1911, a comprehensive report of the present status of aluminium conductors for electrical transmission was given by Mr. E. Huber-Stockar. On the basis of equal conductance there is a saving of 30 per cent. in cost of aluminium over copper, at present prices. Recent tests have shown that aluminium is in general the equal of copper in resisting corrosion by moist air, salt air, salt water, various gases, acids, etc. In making connections in the power-house, aluminium conductors have many advantages; on account of large radiating surface, they work cooler than copper and small conductors are stiffer than copper for a given conductance, reducing the number of necessary supports. Stranded are to be preferred to single-wire conductors. On the basisof present prices, aluminium conductors are about 20 per cent. cheaper than copper. (*Electrical World*, November 11th, 1911, p. 1180.)

Railway Motors.—London Electrician, July 21, records the use of aluminium coils in place of copper in railway motors. The weight is 50 to 55 per cent. of copper; a copper coil weighing 68.2 lb. cost $21.40, while the similar aluminium coil weighing 33 lb. cost $8.00; on renewal of the coil there is an ultimate saving of $3.60 in favor of the aluminium.

Dust Explosions.—R. J. Zinc (*Chemische Zeitschrift* 1911, page 1370) states that aluminium dust alone is not explosive, but becomes so in presence of inflammable gases in quality corresponding to one-third of the limit of inflammability of the gas. In aluminium-bronze factories explosions are due to electrical discharges from the machinery. Aluminium carbide in the aluminium dust might produce methane gas and thus an explosion. 0.1 per cent. of aluminium carbide was found in aluminium, capable of forming 4.8 liters of methane per kilogram of dust. The author suggests that the grinding mill should be closed dust-tight by a loose plate, which would be blown off if an explosion occured.

Aluminium-coated Iron.—A process is said to be working in Germany for coating black-plate on one side only with aluminium, to take the place of tin-plate. It is said to be much cheaper than ordinary tin plate, the material can be printed upon and used exactly as tin plate. (*General Trade Review*, March 2, 1911).

Corrosion of Drawn Articles.—Professors Heyn and Bauer, in the German Testing Laboratory at Gross-Lichterfield, Berlin, have studied the cause of the corrosion occurring along well-defined lines on the surface of drawn aluminium cooking utensils. (*La Nature*, Dec., 1911, p. 60). They conclude that neighboring bands of the metal have received an unequal amount of working, and are, therefore, in a different mechanical

state of strain. If these strains are not removed by a thorough annealing, the different bands will possess different electrical potential in contact with solutions, and set up closed voltaic circuits which cause quick corrosion. The harder parts will act as anode, and be the most corroded. This is also true of drawn wire. Annealing at 450° C. removes all the tendency to corrosion.

Melting.—E. A. Barnes of the Ft. Wayne Electric Works described before the American Brass Founder's Association (*Met. and Chem. Eng.*, June, 1911) his experience in re-melting aluminium for foundry purposes. When melting ingots, scrap, old runners, etc., in a furnace where the metal was exposed to contact with the flame, the loss of metal by oxidation was 20 per cent. Melting the same material in an iron crucible in a tilting furnace, the flame coming in contact with the outside of the crucible only, the loss was reduced to 2 1/2 per cent. A pyrometer, dipping into the metal, is always used, to prevent overheating. A chemical engineer is a useful man in the foundry; the foundryman has too many things to do to spend the proper time on developing new methods.

Soldering Aluminium.—J. N. Dandelin (*U. S. Patent* 989,573, Apr. 18, 1911; *Met and Chem. Eng.*, June, 1911) patents the admixture of tin 65.8 per cent., lead 17.4 per cent., zinc 12.2 per cent., copper 3.1 per cent., aluminium 0.65 per cent., phosphorus 0.55 per cent., silver 0.30 per cent. It is claimed that it runs easily, is tough and strong, and may be used on other metals than aluminium.

The solder of Mr. Joseph Richards, in commercial use since 1895, is still giving satisfaction. In 1911, over 2500 lb. of it was used in the United States, and 1000 lb. in Europe; this would solder an immense bulk or weight of aluminium articles, and the metal workers report it easy to use, and as giving strong, permanent joints.

ALUNDUM

Alundum is oxide of alumina, Al_2O_3, in crystalline form. The sole producer is the Norton Company, Worcester, Mass., whose alundum electric furnace plant is located at Niagara Falls, N. Y.

It is made by fusing the mineral bauxite in the intense heat of the electric furnace by the arc process. Chemically, bauxite is the purest form of aluminum oxide found in nature, that is, it contains less impurities such as iron and silica, than emery. Bauxite was considered infusible until the invention of the alundum electric furnace process.

In chemical composition, alundum is similar to the ruby and sapphire, the hardest natural minerals except the diamond.

Alundum is not radically different in chemical composition from the mineral corundum, but it is a purer oxide possessing superior abrasive qualities which is evidenced by the fact that there has been a constantly increasing demand for abrasive products made of alundum ever since its introduction into the abrasive field.

PRODUCTION OF ALUNDUM IN THE UNITED STATES.

Year.	Pounds.	Value.
1904	4,020,000	$281,400
1905	3,612,000	252,840
1906	4,331,233	303,186
1907	6,751,444	405,086
1908	3,160,000	189,600
1909	13,578,000	814,680
1910	13,410,000	804,600
1911	11,116,000	666,960

The bauxite used by the Norton Company is selected in the southern states of the United States and only the highest quality is used for this purpose. It passes through the processes of washing, drying and calcining to remove the water chemically combined in it before it is ready to go into the electric furnace. The electric furnaces used in the production of alundum are of a special type consisting of a crucible of 4 ft. in diameter by 5 ft. high, having two carbon electrodes 4×12 in. in cross-section. The bauxite is fed directly into the arc. Each ingot formed in the furnace weighs about two and one-half tons. It passes through many operations of crushing and grading before it is reduced to size and ready for commercial use.

The manufacture of alundum is under control which enables the production of this abrasive material in different combinations of hardness, sharpness and toughness, which makes it applicable to a wide range of uses.

A few years ago alundum was used only in the manufacture of grinding wheels. The Norton Company later began to use it exclusively in the manufacture of sharpening stones, and in grain form there is now an extensive use for it in polishing metal. Wheels made of this material have been successfully introduced into the cut glass industry. It is used in the manufacture of valve grinding compound.

A line of refractory articles for general laboratory use, electric furnace parts, etc., was introduced in the year 1910 and has met with marked success. Alundum has valuable properties for the resistance of heat and corrosive action of oxides, molten metals, slags, etc. It has a low coefficient of expansion, the melting point is approximately 2000° C. and is a non-conductor of electricity; in fact, it possesses many properties which make it valuable for use in the manufacture of many kinds of laboratory ware.

Alundum is used in combustion boats, ignition and filter crucibles, extraction thimbles, filter dishes, muffles, tubes, and cores and has been found to be successful. For melting crucibles, while somewhat porous for slag, it is especially serviceable for melting metals of high fusion point. It is stated[1] that two varieties of alundum can be obtained, one nearly pure white, the other, owing to a higher impurity, being reddish-brown. The purer variety has a specific gravity of four, conducts heat somewhat better than fire clay and resists the action of all aqueous and alkaline solutions. Experiments have also been made with alundum brick for furnace linings but lime and magnesia vapors when present are apt to cause flaking off and therefore make the increase in life over other brick less than to warrant an extra cost. The use of alundum brick in roofs where high temperatures are necessary, have shown that it will stand in cases when silica brick would be rendered useless in five or six hours.

[1] L. E. Saunders. T. A. Electrochem. Soc., XXI, p. 333.

AMMONIA AND AMMONIUM SULPHATE

By W. N. McIlravy

The production of ammonia in the United States in the year 1911 was increased by an amount estimated at 11,000 tons of sulphate, which is about the same as the increase registered for the year before. This annual increase is a fairly accurate measure of the progress of the by-product coke oven industry, which is chiefly responsible for the addition to the figures. The progress made is encouraging in that it is progress, but if it is compared with the forward strides already made by England and Germany, it can hardly be regarded as rapid.

The by-product plants that came into operation in the year 1911 include 560 ovens at Gary, Indiana, 60 ovens at Woodward, Alabama, and 110 ovens at Sault Ste. Marie, Ontario. The latter plant is geographically just over the United States boundary, but it cokes coal from this country and this coke is used in place of that formerly shipped from here. Therefore, as far as the progress of the industry is concerned, its figures may fairly be included in the United States total.

The prospects for 1912 include the placing in operation of 280 ovens at the plant of the Tennessee Coal & Iron Co., Corey, Ala., and 300 ovens for the Bethlehem Steel Co., at Bethlehem, Pa. Both of these plants are well along toward completion. They will add 580 ovens to the producing capacity, as against 830 ovens added last year.

The figures for the production of the country for the past 14 years are given in Table 1, which also gives the percentage obtained from by-product coke ovens. The total output from this source in 1911 was 95,000 tons of sulphate and its equivalent, which is 9000 tons gain over the output for 1910. It should be understood that this table includes only ammonia actually recovered as such from coal or bone carbonization, and does not refer to the so-called animal ammoniates in which nitrogen exists but has not been segregated as ammonia. The figures for the by-product coke ovens are compiled from actual reports made by the operating plants or from "Mineral Resources." Those for coal gas production are based on the latter source of information where such data are given, with due allowance for bone carbonization.

The consumption of ammonia in different forms in the United States has not, however, waited for the development of the domestic industry.

The advance has been regular and rapid, and the year just passed has emphasized this tendency by a large increase in imports, and in the selling price as well. The figures for the last 10 years are given in Table 2. According to them the imports increased from 1909 to 1910 by more than one-half, and from 1909 to 1911 were multiplied two and one-half times. The addition to the imports of the current year over the year before were greater than the whole amount received in 1909. The United States as in years past, occupies the third rank in the ammonia producing nations of the world, but her consumption is second only to that of Germany.

Market.—The price figures given are the results of averaging the regular market quotations for the year, and comparing 1910 and 1911 show an advance from $55.60 per net ton to $62.31. This figure is as high as has been attained since 1894, except that for 1905. The movement of prices for the year 1911 has not extended over a wide range, but has been decidedly upward. This is indicated by the average prices for the first and last halves, which were $3.06 and $3.15 per 100 lb. respectively. Beginning at $2.96 in January, the quotations increased to $3.15 in the middle of March, but by May 1, had fallen to $3.07 1/2 and on July 1 were $3.02 1/2. From that time on, however, the advance in prices was almost uninterrupted, and the year ended up at $3.20. The drop in prices which followed the removal of the tariff in 1909 has been obliterated, so that the effect of this measure seems to have been to obtain for the country a larger amount of foreign sulphate of ammonia. Practically all of this is used in agriculture, and is, of course, an advantage to the country, but this benefit would be considerably enhanced if the source of supply was transferred to our own coking industry which now wastes annually five-sixths of the ammonia it might recover.

England.—According to the figures given in Table 3, England shows an increase in production of 11,000 tons, or 3 per cent. over the figure for 1910. This is only a little more than the increase in the United States, and the major part of it is due to the same source, namely, the by-product coke ovens. The output from the coal gas industry has apparently reached a point where it gains only with the gain in the use of coal gas, which is not as rapid as in former years because of the development in the use of carburetted water gas. In the coke-making industry, however, approximately one-quarter of the product is made without recovery of the available ammonia, hence increase in ammonia recovery from this source should be more rapid.

England occupies the second place in the list of ammonia-producing countries, Germany having definitely passed her this year. The consumption in the United Kingdom is given as 85,500 tons for the year 1911, somewhat less than last year.

The average price for the year was £13 15s 3d, per gross ton, a gain of £1 12s 1d over the average of last year, and equivalent to $59.80 per 2000 lb. This is the highest average price monthly or yearly, that England has recorded since 1894, and the highest yearly average since 1884.

In the export trade, England is still pre-eminent, her export total for the year being 291,883 tons, the largest in her history, as against 283,772 tons for the year before. Of her customers Japan now ranks first, having taken 75,095 tons, the United States second, with 65,920 tons, Spain and Portugal third, with 56,501 tons, Java fourth, with 28,168 tons, and Italy fifth with 10,162 tons. The remaining 56,000 tons are distributed among France, Germany, Belgium, Holland, the Canaries, British Guiana, Mauritius and elsewhere in quantities under 10,000 tons. The principal feature worthy of note in this list is the forward movement of Japan which has advanced into the place the United States held last year, with about the same quota. This shows an advance in consumption for Japan, but for the United States indicates only that that country obtained the balance of her requirements elsewhere, mostly from Germany. Japan and the United States together took 141, 000 tons, or nearly half of England's total exports. As regards future requirements on the part of these two countries, there appears to be no reason to fear any serious diminution. The past record of Japan in the market, and the fact that sulphate of ammonia has proved to be an especially successful fertilizer for rice, both point the other way. As for the United States, the consumption seems liable to increase if anything, a view which the price movements since the first of 1912 decidedly reinforce. Until the domestic production assumes larger proportions than immediate prospects indicate, regular gains seem probable.

Germany.—The production of ammonia in Germany has finally passed that of England, and the former is now the largest producer in the world. (See Table IV.) Reckoning all forms as sulphate of ammonia, Germany's estimated output for the year 1911 was 418,000 tons, a gain of some 45,000 tons supplementing the regular annual gains of amounts frequently as large as this, that have been registered in years past. Such an advance, considering those that had gone before, was hardly to be expected, and emphasizes the extent to which economy in the treatment of coal has been carried in Germany.

The figures for German imports and exports are as follows:

	1911	1910
Sulphate of ammonia imported, metric tons	24,463	31,400
Sulphate of ammonia exported, metric tons	74,410	92,996
net exports	49,947	61,596

These figures indicate that in spite of the increase in production, there was less margin for export in 1911 than in 1910. The consumption

for 1911 is estimated at 370,000 tons, an increase of 20,000 tons over the year before. It is possible, however, that this amount would have been even larger but that the market was practically bare of stock in the early months of 1911. The progress of consumption for the last five years is shown by the following figures, which are to be regarded as estimates.

1907	240,000
1908	260,000
1909	274,000
1910	350,000
1911	370,000

The absorption of such large amounts is due to the energetic propaganda carried on by the Deutsche Ammoniak-Verkaufs-Vereinigung on behalf of the use of sulphate of ammonia in agriculture.

France.—The production of ammonia reckoned as sulphate in France is given in Table V for the past four years. It will be noted that the largest single item as well as the largest increase is attributed to the by-product coke oven industry. The total recovery for the year, 60,000 tons, shows about the same increase as last year.

The consumption may be approximated from the import and export figures as follows:

Imports, 1911	23,566 metric tons
Production	60,000 metric tons
	83,566 metric tons
Exports, 1911	2,040 metric tons
Available for consumption	81,526 metric tons

The amount available for consumption the past year is about 10,000 tons larger than that available for the year preceding.

Japan.—The use of chemical manures in Japan is evidently well established and this country has come to be relied on as a market for sulphate of ammonia. The amount of English sulphate taken by them in 1911 was 76,095 gross tons, a figure greater than that for 1907, the maximum hitherto. The growth of the demand since 1901 is indicated by the figures in Table VI. According to figures published in the *Zentralblatt für die Kunstdünger-Industrie*, Japan used 45,835 tons of sulphate of ammoina in 1909 and 68,813 tons in 1910, in addition to quite an impressive amount of other fertilizers, the total value of which approximated eighteen and one-half million dollars.

World's Production.—An estimate of the world's production of sulphate of ammonia and sulphate equivalent is given in Table VII, the output of each country being reduced to metric tons for convenience in tabulation. As is indicated by these figures the product of the world now amounts to nearly 1,187,000 tons.

TABLE I.—UNITED STATES AMMONIA PRODUCTION, EXPRESSED IN SULPHATE EQUIVALENT.
(Tons of 2000 lb.)

	1898.	1899.	1900.	1901.	1902.	1903.	1904.
By-product coke ovens	3,600	6,000	13,800	15,279	18,483	24,098	32,653
Coal gas and bone carbonising works	13,400	(a)13,500	(a)13,800	(a)14,000	(a)17,641	17,775	22,011
Total	17,000	19,500	27,600	29,279	36,124	41,873	54,664
Percentage from coke ovens	21	31	50	52	51	57	60

	1905.	1906.	1907.	1908.	1909.	1910.	1911.
By-product coke ovens	41,864	.....	62,700	50,073	75,000	86,000	95,000
Coal gas and bone carbonising works	23,432	.....	36,609	33,327	(a)31,500	(a)30,000	(a)32,000
Total	65,296	(a)75,000	99,309	83,400	106,500	116,000	127,000
Percentage from coke ovens	64	.....	62	60	70	74	75

(a) Estimated.

TABLE II.—UNITED STATES AMMONIA CONSUMPTION, EXPRESSED IN SULPHATE EQUIVALENT(a).
(In tons of 2000 lb.)

Year.	1901.	1902.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Imports	14,486	18,146	16,777	16,667	15,288	9,182	32,669	34,274	40,192	63,178	103,743
Total Consumption	43,765	54,270	58,650	71,331	80,584	84,182	132,000	121,874	149,192	179,178	230,743
Average price	$55.16	$59.90	$62.10	$61.71	$62.92	$62.33	$61.93	$59.90	$56.04	$55.60	$62.21

(a) The figures for consumption and price are for the calendar year, while those for imports are for the fiscal year ending June 30th.

TABLE III.—AMMONIUM SULPHATE AND SULPHATE EQUIVALENT PRODUCED IN THE UNITED KINGDOM.
(Tons of 2240 lb.)

Year.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Gas works	150,957	155,957	157,160	165,474	165,218	164,276	167,820	169,500
Iron works	19,568	20,376	21,284	21,024	18,131	20,228	20,139	20,000
Shale works	42,486	46,344	48,534	51,338	53,628	57,048	59,113	60,000
Coke ovens	20,848	30,732	43,677	53,572	64,227	82,886	92,665	129,000
Producer gas and carbonizing works	12,880	15,705	18,736	21,873	24,024	24,705	27,850	
Total	245,990	269,114	289,391	313,281	325,228	349,143	367,587	378,500

These figures are from the Alkali Inspector's reports, except for 1911, which are estimates given out by Messrs. Bradbury & Hirsch.

TABLE IV.—PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT FROM BY-PRODUCT COKE OVENS AND GAS WORKS IN GERMANY.
(In metric tons of 2204.6 lb.)

Year.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Coke ovens	120,000	152,000	168,000	.....	257,000	.....	270,000	313,000	358,000
Gas works	20,000	21,000	22,000	.....	30,000	.....	60,000	(e)60,000	(e)60,000
Total	140,000	173,000	190,000	235,000	287,000	313,000	330,000	373,000	418,000

For 1903–4–5, see *Zeit. f. angew. Chem.*, Sept. 14, 1906.
Deutsche Ammoniak-Verkaufs-Vereinigung, 1906–07–08–09–10–11.
(e) Estimated.

TABLE V.—AMMONIUM SULPHATE AND SULPHATE EQUIVALENT PRODUCED IN FRANCE.
(Tons of 2204.6 lb.)

	1909.	1910.	1911.
Gas works, Paris and vicinity	14,000	17,000	14,700
Provincial gas works	4,000		4,300
By-product coke ovens	22,800	26,500	28,500
Sewage disposal	11,300	11,500	11,200
Shale works	900	1,000	1,300
Various sources	500		
Total	53,600	56,000	60,000

TABLE VI.—EXPORTS OF SULPHATE OF AMMONIA FROM THE UNITED KINGDOM TO JAPAN.
(Tons of 2240 lb.)

1901.	1902.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
1,290	2,429	3,612	14,981	33,861	33,237	64,270	38,745	49,275	57,360	76,095

TABLE VII.—WORLD'S SULPHATE OF AMMONIA PRODUCTION.
(In metric tons of 2204.6 lb.)

Country.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
England	250,050	273,550	294,170	318,400	330,450	354,747	373,590	384,680
Germany	173,000	190,000	235,000	287,000	313,000	330,000	373,000	418,000
United States	49,600	59,250	68,000	90,120	79,500	96,600	105,143	115,245
France (b)	43,000	47,300	49,100	52,700	52,600	53,600	56,000	60,000
Belgium and Holland (b)	(a)39,000	24,200	30,000	(a)55,000	(d)35,000	(d)40,000	(d)43,000	(d) 43,000
Spain (b)	.....	10,000	10,000	(c)12,000	.....	12,000	9,000	
Italy (b)	48,000	4,500	5,000	11,000	(d)80,000	12,000	12,000	(d) 166,500
Other Countries (b)	.....	40,500	40,000	65,000	.....	(d)73,000	(d)79,000	
Total	602,650	649,300	731,270	891,200	890,550	971,947	1,050,733	1,187,425

(a) Including Norway, Sweden and Denmark.
(b) Estimates from l'Engrais.
(c) Including Portugal.
(d) Estimates from Deutsche Ammoniak-Verkaufs-Vereinigung, 1908–9–10–11.

ANTIMONY

There was no production of antimony ore in the United States during the year 1911 and antimony continued to be a distinctly minor metal. The production in 1908 was 360 tons; in 1909, 95 tons and in 1910, 75 tons. This ore was used only for experimental purposes. The continuance of a duty of one and one-half cents per pound on metallic antimony and one cent per pound on the ore, could not have any stimulating effect on production, as the antimony was selling at only eight cents per pound.

The antimony imported in the form of ore, regulus or metal was 10,975,844 lbs. valued at $573,564 in 1911 as compared with 9,899,514 lbs. valued at $572,031 in 1910. The duty on the antimony contents in ore is one per cent. per lb., and on regulus or metal is one and one-half cents per lb. The relative proportion of ore and matte not containing more than ten per cent. lead and the regulus or metal imported is shown by the figures for the fiscal year to June 30, 1911, as 1063 tons containing 1,609,916 lbs. of antimony as compared with the antimony in the regulus which was 7,988,844 lbs. for the same period.

Markets.—As shown in the accompanying statistical table, the January price of Cookson's was one-half cent over the United States brand with an average price of 8 1/4 cents for the month. While the table shows that the market was generally dull due to the lack of demand as there were no large purchases by the railroads, the so-called International Antimony Syndicate put prices to an artificial level for a while, but they collapsed in the month of September. About 70 per cent. of the metallic antimony is imported mostly in the form of Cookson's brand but the largest users of antimony claim that there is a great advantage in using the purest metal on account of drossing and also on account of the exact knowledge of the composition of the resultant alloys. Much antimony is used in the form of antimonial lead all of which finds a ready market without the necessity of separating the metals. There is also a small production of secondary antimony by the scrap smelters. In the United States antimonial lead is produced by the American Smelting and Refining Company, the United States Metals Refining Company and the Balbach Smelting and Refining Company. There is also a considerable importaton of needle antimony from China of which the production, so far, has not been interferred with by the revolution, although it has been stated that the operations were not as successful in China as at first anticipated. This was indicated by a slight fall in the production.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK. (a)

(Cents per pound.)

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1903.													
Cookson's	8.25	8.25	8.25	8.25	8.00	7.50	7.44	7.15	7.00	7.00	6.56	6.75	7.53
Hallett's	7.00	7.00	6.87	6.87	6.75	6.69	6.50	6.40	6.34	6.25	6.25	6.35	6.69
Others	6.75	6.62	6.50	6.50	6.50	6.44	6.25	6.19	6.00	6.00	6.00	5.95	6.31
1904.													
Cookson's	6.938	7.594	7.875	7.875	7.531	7.200	7.188	7.188	6.913	6.984	7.592	8.388	7.439
Hallett's	6.250	6.781	6.825	6.750	6.578	6.438	6.485	6.688	6.537	6.578	7.328	8.160	6.783
Others	5.688	6.203	6.475	6.406	6.203	5.961	5.969	6.062	6.015	6.172	7.204	8.088	6.371
1905.													
Cookson's	8.375	8.375	8.375	8.219	8.406	11.025	12.625	14.500	13.700	13.000	12.500	14.000	11.100
Others	8.063	8.063	7.638	8.125	8.406	10.175	11.875	13.500	12.900	12.000	11.250	12.750	10.400
1906.													
Cookson's	15.0	16.0	17.5	21.31	25.25	26.0	25.25	25.0	24.5	25.2	26.14	26.25	22.78
Hallett's	14.0	15.0	16.5	20.81	24.38	25.0	24.25	24.0	24.0	24.81	25.25	25.24	21.94
Others	13.5	14.25	16.15	20.25	23.31	24.0	23.19	22.75	22.25	23.63	24.50	24.70	21.73
1907.													
Cookson's	25.906	25.062	24.90	24.125	21.937	15.75	11.875	10.906	10.75	11.75	11.00	9.662	16.969
Hallett's	25.219	24.062	23.75	21.344	18.562	13.812	10.50	9.687	10.00	10.406	9.937	9.05	15.527
Others	24.156	23.437	23.025	20.875	17.75	12.65	10.125	9.375	9.65	10.047	8.906	8.088	14.840
1908.													
Cookson's	9.344	9.266	9.000	8.969	8.875	8.734	8.594	8.313	8.234	8.284	8.640	8.200	8.704
Hallett's	9.031	9.016	8.650	8.672	8.625	8.531	8.375	8.150	7.922	7.922	8.075	8.063	8.419
Others	8.344	8.406	7.988	8.297	8.250	8.094	8.125	7.850	7.609	7.625	7.775	7.688	8.004
1909.													
Cookson's	8.202	8.125	8.047	8.250	8.387	8.312	8.375	8.525	8.687	8.537	8.437	8.437	8.360
U. S.	8.075	8.000	7.843	8.031	8.150	8.062	7.875	8.125	8.125	8.012	7.937	7.937	8.015
Others	7.675	7.531	7.500	7.718	7.887	7.893	7.375	7.625	7.506	7.500	7.687	7.687	7.466
1910.													
Cookson's	8.500	8.469	8.359	8.438	8.438	8.241	8.175	8.278	8.313	8.263	7.922	7.625	8.252
U. S.	7.988	7.969	7.938	7.938	7.938	7.938	7.938	7.938	7.938	7.900	7.656	7.438	7.876
Others	7.738	7.578	7.313	7.438	7.438	7.438	7.400	7.328	7.313	7.400	7.188	7.063	7.386
1911.													
Cookson's	8.25	8.38	9.56	9.54	9.50	8.75	8.50	8.44	8.31	8.14	7.97	7.78	8.59
U. S.	7.74	8.02	9.13	9.13	8.97			7.94	7.84	7.71	7.61	7.50	8.16
Others	7.5	7.65	9.00	8.52	7.99	7.42	7.35	7.36	7.26	6.96	6.92	6.82	7.54

(a) From *Eng. and Min. Journal.*

ANTIMONY STATISTICS OF THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Imports.		Exports.		Production.			Consumption.
	Metals or Regulus.	Ore.	Metal or Regulus.	Ore.	In Hard Lead.	From Domestic Ore.	From Imported Ore. (a)	
	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons
1898	1,013	1,863	13	17	2,118	250	738	4,106
1899	1,580	1,991	*Nil.*	*Nil.*	1,586	234	796	4,196
1900	1,816	3,018	21	*Nil.*	2,476	151	1,207	5,638
1901	1,837	866	*Nil.*	25	2,235	50	336	4,458
1902	2,871	840	37	104	2,904	*Nil.*	294	6,032
1903	2,563	1,337	40	*Nil.*	2,552	*Nil.*	535	5,610
1904	2,028	1,245	16	214	2,515	*Nil.*	412	4,939
1905	2,869	988	*Nil.*	*Nil.*	2,561	*Nil.*	395	5,825
1906	3,950	1,124	12	*Nil.*	2,358	150	450	6,866
1907	4,331	1,380	24	6	2,240	105	552	7,204
1908	4,057	1,640	1	5	2,621	180	656	7,513
1909	4,826	1,693	3	*Nil.*	2,546	38	694	8,055
1910	4,950	*Nil.*	*Nil.*	*Nil.*	2,277	30	*Nil.*	7,257
1911	5,479	*Nil.*	*Nil.*	*Nil.*	1,783	*Nil.*	.....	.....

(a) Estimated at 40 per cent. extraction from net imports of ore.

PRODUCTION OF ANTIMONY METAL IN FOREIGN COUNTRIES.
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908	1909	1910
Austria.....	271	153	114	24	14	36	90	*Nil.*	207	162	*Nil.*	*Nil.*
China(b)....	.....	.....	.....	.....	.....	.....	3,353	3,829	2,316	9,356	5,762	(c) 6,812
France.....	1,499	1,600	1,786	1,725	2,748	2,116	2,396	3,433	3,945	3,850	5,444	4,640
Hungary(a)	940	846	706	683	732	1,007	756	1,322	841	670	695	782
Italy.....	581	1,174	1,721	1,574	905	836	327	537	610	345	59	*Nil.*
Japan.....	229	349	429	528	434	321	190	627	248	198	157	120

(a) Regulus. (b) Exports of regulus and refined metal. (c) Metal only.

Metallurgy.—There seems to be a crop of suggestions for a simple metallurgy of antimony which would be good if they would be certain to work. Pending the arrival of some workable improved solution, the method of smelting sulphides to a regulus or reduction by iron or similar agents, and the reduction of oxides previously volatilized and collected, continues to be the standard practice. A stimulating effect upon the consumption of antimony will be that of the anti-phosphorous bill if enacted by Congress.

The smelting of lead antimony ores in Queretaro, Mexico, was continued. The cost of this operation was about $12 per ton, not including the marketing of the product which was expensive on account of a haulage costing $7.25 per ton. The process was quite efficient, probably recovering 70 per cent. on the average. The success of the process was no doubt due to the control by bedding the ores so as to have uniform mixture with as little sulphur as possible.

Antimonial lead is now very extensively used in the manufacture of type metals of all kinds, babbitt metals and metal novelties. As obtained in the refining of silver, antimonial lead is quite impure with arsenic as the chief impurity. It has been found somewhat difficult to recover arsenic from antimonial lead and 5 or 6 per cent. in the past was not uncommon. A process for the removal of arsenic has recently been patented by Mr. Leland E. Wemple of the Hoyt Metal Company of St. Louis, Missouri.[1] This process makes use of the fact that molten caustic soda, when in contact with molten antimonial lead, will remove the arsenic without appreciably removing the antimony. The impure antimonial lead is melted in an iron kettle and is allowed to flow in drops or a small stream, into a second kettle containing the molten caustic. As the antimonial lead passes through the molten caustic soda, a large surface is exposed, thus removing the arsenic. After a time, the caustic soda becomes so contaminated with arsenic that it is replaced with a fresh supply. Any caustic soda that enters the refined lead kettle is removed by the addition of a small quantity of hydrogen sodium sulphate (niter

[1] *Brass World*, December, 1910.

cake), the resulting mixture being neutral. The best method of using the niter cake is to force pieces under the metal and as the resulting slag rises to skim it off. If the action of the caustic soda on the antimony is to be retarded, it is recommended to use iron scale, a portion of about 20 per cent. When it is desired to use this process to remove either tin or bismuth and antimony from the lead, an intensifier such as sulphur, niter or sodium peroxide is used with the idea of making the reaction quicker.

Antimony.—Black antimony, the sulphide used in medicine, condition powders for horses, etc., is much adulterated, as no standard of purity exists. It should be free from arsenic.

Antimony Oxide.—The use of antimony oxide as a substitute for the oxide of tin in the ceramic and enamel industries, has been favored by the high price of tin during 1911. The Harshaw, Fuller & Goodwin Co. was, until recently, the sole manufacturer, but owing to increased demand, a larger plant became necessary, and the Enamel Co. of America, interested in the product, bought up and enlarged the Elyria plant of the Harshaw-Fuller company. The Enamel Co. of America has appointed as its joint selling agents the Roessler & Hasslacher Chemical Co. and the Harshaw, Fuller & Goodwin Co. It is understood that this plant operates chiefly on Chinese regulus. The imports of antimony oxide are small. The price of this product in 1911 was about 7 3/4 to 8 cents per pound.

Antimony lithophones are said to be successful in Germany. They are made by treating barium carbonate and antimony sulphite. The oxide is used for white paint; sodium metaantimonate is a valuable constituent of enamels, these being prohibited in Europe.

Electrolytic Antimony.—M. Ruthenburg, of London, has patented (Brit. pat. 19,722, Aug. 24, 1910) a cyclic process for the production of antimony. A solution of ferrous and antimonious chlorides, obtained by treating stibnite with ferric chloride, and containing 1 lb. of antimony per gallon, is electrolyzed between soluble iron anodes and insoluble cathodes until about one-half of the antimony is deposited. The remaining liquid is then oxidized—by a hot-air blast or otherwise—the resulting solution of ferric and antimonic chlorides being employed for the treatment of a further quantity of stibnite.

Auriferous Antimony Ore Treatment.—A recent process for the treatment of gold-bearing antimony ores is that of John Jones and H. S. Bohn, of Mount Morgan, Australia (U. S. pat. 984,090). According to their scheme, the ore is treated with a solution of hydrochloric acid. The solution is treated with chlorine to destroy any retained sulphuretted hydrogen, and the resulting solution, after removing the suspended

impurities, is electrolyzed for antimony. The greater part of the sulphuretted hydrogen expelled from the ore is drawn off into a gasometer and treated with chlorine evolved in the electrolysis, thereby regenerating hydrochloric acid, and precipitating sulphur. The residue from the hydrochloric-acid leaching is treated for its gold content.

At the Phoenix mine which is the first gold producer in Rhodesia, the ore contains 0.73 per cent, of antimony which was found very troublesome in extracting gold. The concentration of antimonial gold ore is very difficult owing to the tendency of antimony to slime. In the original plant the extraction of gold was no more than 30 per cent. The extraction was raised to 75 or 80 per cent. by allowing the slime to run to dams and weather there for about six months. With the sands this could not be done and attempts to jig the ore proved disappointing. As the reserves of rich ore increased, the management decided to increase the efficiency of the installation. With ten stamps set apart with two grinding pans followed by Wilfley and canvas tables, the ore assaying up to 25 dwt. in gold, an extraction of 90 per cent. was obtained. This was accomplished by finer grinding thus driving more antimony into the slimes. The plant commenced operations about December, 1910. The ore after being crushed to 1 1/2 in. was passed to 1/4 in. trommels, the coarser portion being picked, from which one-half ton to one ton per day of massive stibnite results. The remainder of the ore is ground finer in stamps. The tailing, on leaving the plates, runs into classifiers to separate the slime, and all the sand passes to five Record vanners, extracting as much of the antimony and pyrite as possible. This product containing about 70 per cent. silica is elevated to settling-tanks at the concentrate roasting plant. The tailings from the vanners pass into the grinding and amalgamating pans and are treated on blanket tables, the resulting fine concentrates are added to those from the vanners. As long as the antimony in the sand charges exceeds two-tenths of one per cent., the gold extraction drops very rapidly.

An analysis of the raw concentrate shows about 65 per cent. silica, 5 per cent. antimony and 9 per cent. sulphur, while the roasted product contains 3 per cent. antimony. The present furnace treats a matter of 11 tons per day and it is proposed to erect a larger one of 100 tons capacity. The above process is described not so much on account of its efficiency as far as antimony is concerned, but to show the possibilities of utilizing this by-product.

Arkansas.—The principal antimony veins opened are three miles east of Gillham. These extend eastward six to eight miles, to the shaft of what was formerly known as the United States Antimony Company, and are traced eastward to the May shaft of the American company at Antimony

City, Howard County. This vein is from 4 to 5 ft. thick and has produced several hundred tons of antimony ore. The shaft is about 120 ft. deep and on the foot-wall a solid vein of stibnite ore 8 to 14 in. wide was mined.

Nevada.—Although there was no production in the United States in recent years of antimony ore, the requirements for sample lots for experimental purposes, are so high that ore of 50 per cent. and 60 per cent. antimony contents is necessary. Five or six years ago, the ores shipped from Nevada from the mines owned by Charles A. Green, were sent principally to the Great Western Smelting and Refining Company and the Hoyt Metal Company and some minor concerns of junk smelters. The price then was 12 cents per pound but eventually only the highest grade ore was shipped. A great deal of 20 per cent. to 40 per cent. ore is never brought to market unless it contains silver and gold values. The possibility that such ores could be utilized is very great. In order to increase the grade of production, experiments have been made in a small reverberatory furnace liquating the sulphide from the quartz antimony ore to 71 per cent. product needle stibnite or crude and it promises to be a commercial success. There is no doubt that the mining camp in Bloody Canyon, is capable of supplying any demand in the future, but the demand does not exist at present. Formerly a small concern at Oakland, California, produced metallic antimony but is now extinct.

Antimony in Foreign Countries

Austria.—There were 225.5 tons of antimony ore mined in Austria in 1910 but no metallic antimony was produced.

In 1908, Hungary produced 1316 metric tons of antimony ore, and in 1909 1971 tons; also crude antimony, 8 tons in 1908 and 10 tons in 1909; there were 1115 tons and 685 tons of regulus produced during the same years, the prices of the latter being $137 and $113 per ton respectively. The ore produced in 1910 amounted to 1939 tons and the regulus to 782 tons.

Canada and Nova Scotia.—The West Gore Antimony Company at West Gore, did not operate its mine during the year and the production of 191 tons of concentrates made and shipped was from the milling of 2004 tons from dump.

This company has for the past two or three years been engaged in litigation with respect to titles. It is stated that a settlement has been reached and that mining operations are to be resumed during the coming year.

China.—China possesses the distinction of being first in the production of this metal. The condition of the industry has been changing rapidly of recent years. The ore comes from a number of places to the northwest of T'ung-t'ing Lake in Hunan, I-yang being the chief center. After sorting the ore, it is liquated in pots about 15 inches in diameter but the original Chinese methods are being replaced by furnaces of modern construction. The product is exported to France, Germany and the Netherlands for further treatment; also large quantities have been sent to the United States. At Hankow, Carlowitz and Company are engaged in the smelting of antimony and lead. The company which operates the smelters in the district of Hunan is increasing in importance and native metallurgy is being replaced by the modern methods. The smelter at Wuchou was established under Government auspices and, while not a success at first, is reported to be prospering. A new company has been organized during 1911.

The success of the new antimony refining plant of the Hua Chang Company, established in 1909, is indicated by the increased export of regulus. The plant is not capable of turning out more than 100 tons per month, but additional machinery has recently been ordered from the United Kingdom so as to cope with the increased demand.

Transportation was hampered during the recent revolution but production continued. About two-thirds in value of the exports of antimony are in refined form. In 1911 the total exported was 7509 short tons of metal and 7701 short tons of ore. Of the metal and regulus the United States takes very little, Great Britain one-half and France one-third. Of the ore Great Britain takes one-half and the United States one-tenth. The product is sold by South Chinese firms in Wuchow through Hongkong middlemen. The selling price is reported as $7.68 gold per picul.

Antimony in the form of ware was exported to the United States from Japan in 1910 to the value of $63,800 and in 1911, $66,918.

France.—Antimony worth $72,733 and $58,829 was exported to the United States in 1910 and 1911 respectively, from Havre, France.

Germany.—In 1910 there were 4968 metric tons of antimony produced from the Prussian Smelting & Metallurgical Works.

Italy.—The mines in Sardinia and Tuscany produced, in 1910, 2194 tons of antimony, from which 59 tons of regulus and antimony oxide were derived.

Turkey.—There is one open pit in Turkey which has yielded at the rate of 500 tons of ore per annum. It consists of complex veins and is in calcareous gneiss. This pit is located on the southwestern slope of the Kizil Dagh. Veins of antimonite in schists 60 miles southeast of Smyrna

yielded some 500 tons a few years ago. Another lode at Djin Ukay carries antimonite and pyrite in quartz and soft shale, and from one pocket 4000 tons have been mined.[1] Fifteen hundred metric tons of antimony is the estimated production of Turkey for the year 1911.[2]

Rhodesia.—An extensive stibnite lode is the Hope Fountain, near Bulawayo, Umniati and Belingwe. This is gold bearing and has been worked for gold to some extent. They are also associated with quartz. The Globe and Phoenix mine where there is a plant for roasting the ore, recovers the precious metals, both gold and silver; the process is mentioned above. The Gothic mine and the Pagamesa in the Lower Gwelo district, are of the same kind. The recorded output of stibnite in 1910, was 13.75 tons valued at £275. In the Sebakwe and Hartley districts, the gold deposits show cervantite on the surface and jamesonite below.

At the antimony convention which met at Paris on February 9, 1911, there were representatives of all producers from China, Hungary and Servia. The refineries are in England and France. The price of antimony has been up to $33 per 100 kilogram, but varied between $13 and $14 during the latter part of 1910.

[1] A. S. M. E. W. F. A. Thorne, Vol. XXXIII, p. 208.
[2] *Mining and Scientific Press*, Jan. 7, 1912.

ARSENIC

The domestic production of arsenic for the year 1911 from information furnished by producers was 3081 tons against 1326 tons in 1910 and 1008 tons in 1909. The general arsenic situation is of interest on account of the shifting of trade conditions due chiefly to new producers. Ordinarily a few years ago, arsenic was entirely an imported product and New York the distributing center. The situation has been changing and the distribution centers are moving westward although there is a tremendous expansion in manufacturing arsenic products in the east, more especially insecticides. In Mexico the Compania de Penoles is working its arsenic plant on flue dusts. In Canada, the Deloro Mining and Reduction Company, Deloro, Ont., the Canadian Copper Company, Copper Cliff, Ont., and the Coniagas Reduction Company continued to be the sole producers. In the United States, the two plants of the American Smelting & Refining Co., the Everett Smeltery at Everett, Washington, the Washoe plant at Anaconda, Montana, and the United States Smelting Company at Midvale, Utah, are the largest producers. There is only a small amount of arsenic recovered in the manufacture of sulphuric acid.

The general arsenic-products situation has not materially changed during 1911. One large factory for the manufacture of paris green has been erected at Grand Rapids, Mich., and by some, the announced reduction in the price of paris green is attributed to this competition. There is a constant increase in the amount of insecticides used, and there is considerable preparation being made to satisfy the demand. It is understood that the Grasselli Chemical Company is the largest producer of arsenate of lead in the United States with the Merrimac Chemical Company second. Lead arsenate sold at 6 1/2 to 10 cents per pound and the price of paris green was quoted at 13 cents.

Markets.—The year opened with white arsenic at 2 1/4 cents per pound which was followed by a gradual decline when in July, arsenic could be bought for 1 5/8 cents. The market then suddenly stiffened, due to the stoppage of supplies from the Compania Minera de Penoles in Mexico and the restriction of the production of two American producers, and prices rose until they now stood at 2 3/8 to 2 1/2 cents per pound. A small amount of English white arsenic was sold during a great scarcity

of spot goods. About 125 tons per annum of lead arsenic and 50 tons of metallic arsenic were also imported. Ordinarily no European white arsenic is sold in the United States except some poor quality Belgian. There is no duty on arsenic and its compounds.

Arsenic.—Thirty tons of arsenic are estimated as going out of the stack of the Washoe smeltery per day. This means 80 or 90 per cent. of the total arsenic coming into the plant and hence 10,000 tons are wasted per year from this smelter alone. In 1911 the amount recovered increased four times over 1910, chiefly because of requirement of the law. Due to a much reduced price the open supply is consumed. It is a question whether it cannot be shown by experiments that calcium sulfarsenite and arsenic sulphide, which are harmless to plants, may not be substituted thus saving the lead.

The Brinton Arsenic Smeltery.[1]

The Brinton arsenic mine and smeltery are situated at Brinton, Floyd County, Va., about 14 miles southeast of Christiansburg. The mine was discovered in 1901 and erection of the smeltery begun in March, 1902. The bricks required in building the plant were made and burned on the property, and the lumber cut there. Manufacture began in 1903 and continued intermittently for several years. In 1910 a company was organized to manufacture paris green and other insecticides at Norfolk, Va., from the products of this mine, but in March, 1911, the plant was not yet running.

From the ore pile the rock is run through a Blake crusher, from which it goes to a 10×20-in. Stuyvesant roll; from the roll it goes to a rotary calciner—a large iron tube, 50 ft. long by 6 ft. in diameter, lined with firebrick for 30 ft. from the furnace, which is fired with wood. The charge is treated 45 minutes at a temperature of about 800° F. From the calciner all the gases are drawn through a chambered flue 7 ft. high, 6 ft. broad and 319 ft. long, ending in a tall stack. At intervals of 5 ft. baffles cross the flue alternately from each wall to points within 2 ft. of the other wall. The alternate contraction and expansion and the cooling of the gas current are said to settle the arsenious oxide so completely that none reaches the last five compartments. In this form the arsenious oxide is a dirty gray powder containing sulphur, iron and probably other impurities. The draft in the flue is governed by an electric fan, controlled by a rheostat sensitive to barometric pressure.

After a sufficient amount of crude arsenious oxide has accumulated in the flue , they are opened and the oxide is removed by hand and

[1] Abstract from *Bull.* 470-E. U. S. Geol. Survey.

taken to the refining furnace. The furnace is a brick reverberatory fired with coke, and volatilizes 1400 lb. of crude arsenious oxide in two and one-half hours. The sulphur remaining with the crude arsenious oxide is converted to SO_2 and most of the other impurities are left behind as a residue. After passing through the calciner the white arsenic is caught in a flue similar to that used for obtaining the crude product. From the refiner flue the white arsenic is taken by hand to a 32-in. Waldron burstone mill, and after being ground it runs to an automatic weighing and barreling machine. The white arsenic made is said to have carried 99.975 per cent. of arsenious oxide.

Arsenic Manufacture.—Vol. XIX of MINERAL INDUSTRY gives a description of the furnaces as used at the Midvale Plant of the United States Smelting Co. in Utah. Here the fume collected from the bag-house is treated in a Brunton furnace extracting 60 per cent. from the original fume which runs 43.6 per cent. lead and 21 per cent. arsenic. The product carrying 97 to 99 per cent. As_2O_3 is refined in a reverberatory.

At this plant there are six lead furnaces, 45 × 160 in., having a capacity each of 200 tons of material in 24 hours. The lead is skimmed for a dry dross containing the copper, and then for a wet dross of antimony and arsenic. This is sent to a sweating furnace, residues being returned to the blast furnace. As stated above, the only material treated for arsenic is the fume. The temperature maintained is 900° F. Each of the two furnaces can treat about 16 1/2 tons of fume per day.

The crystalline refined product is ground in a covered mill; the barrels are filled under cover so that no arsenic escapes. The men wear respirators and suits furnished by the company.

The arsenopyrite in the ore here forms small rods which lie both across and along cleavage planes in the mica, to which it is secondary. The lack of definite crystal planes and forms also shows that it was introduced after the schist was formed. There is only a trace of gold or silver in the arsenopyrite; pyrite occurs with the arsenopyrite and sometimes in it.

Canada.—Arsenic may be classed among the non-metals, since it comes into the arts chiefly in the form of arsenious acid or white arsenic. Ontario is rich in arsenical ores, and from the mines of Cobalt a large quantity of white arsenic is yearly obtained by the smelteries during the process of recovering the silver. The output in 1909 was 1085 tons valued at $61,039 and for 1910 the quantity was 1502 tons valued at $75,328. The production of white arsenic for the year 1911 was 2097

[1] *Mines and Minerals*, June, 1910; *Mining World*, May 20, 1911.

tons. In addition, a considerable amount was exported as ore and in the speiss produced by the refineries.

China.—Five thousand tons of arsenic was produced in 1908, but only 400 tons in 1909. A recent French report estimates the annual production of orpiment and realgar in Yunnan at 600 tons. This comes from near Chao-chou.

England.—The production of arsenic in 1910 was 2837 long tons, or 28 tons less than in 1909. In the United Kingdom the production of arsenic for 1911 was 2144 while that of arsenical pryites was 60 tons in 1910 and 1170 tons in 1911.

France.[1]—During 1910 the La Bellière mine produced 8500 tons of ore, yielding 38,783 ounces of gold and 5211 ounces of silver, together with 274 tons of white arsenic. The total value of this output was £175,000. In addition to the La Bellière may be mentioned the mines of La Lucette, Chatelet, Salsigne, and Villaniere. The mineral mispickel, or arsenical pyrite, is characteristic of the gold ore in most of these cases, although at La Lucette the presence of stibnite is a distinguishing feature. One mine, for example at Bourg d'Oisans, in the Dauphiné, was worked successfully by the Greeks, the Romans, and the Saracens. This mine, called La Gardette, was operated by the Count of Provence in 1783 and by two Englishmen in 1862, but it is now idle, for the excellent reason that the vein is too poor to be profitably worked. In later times we find that Napoleon III. stimulated interest in the subject. This led to systematic efforts to exploit the quartz and mispickel veins so numerous in the Auvergne, La Vendée, Dauphiné, and Aquitaine.

Germany.—In the Liegnitz district 1318 tons of mispickel valued at $19,550 were produced in 1910. Near Altenberg eight tons of arsenic ores were mined as a by-product. In the Breslau district Reicher Trost 4386 tons valued at $23.50 per ton in 1909. Clausthal, St. Andreasberg, produced 1.46 metric tons ore worth $240.

Two properties furnished arsenic ores in Prussia in 1910. One in the Lignitz produced 1318 tons of mispickel, and the Reicher Trost mine in the Breslau district, 4462 metric tons, the treatment for both these properties being done at the latter.

Italy.—The arsenic ore produced in Italy in 1910 was 16 tons valued at 1777 lire.

White Arsenic in Spain.—In northwest Spain are several mines of arsenical pyrites from which white arsenic is now being manufactured. The principal mine, the San Jose, is located near Castro del Rey, where the manufacture is carried on as follows:

The ore is thoroughly pulverized to about 1/4-in. mesh, into a hopper

[1] *Min. Mag.*, Mar., 1911.

to the Oxland furnaces. They are revolving cylinders 29 ft. long and 5 ft. in diameter, whose interior is protected by brick, some of which project like a spiral, so the mineral must pass along the cylinder as it turns. The one end fits into a condensing chamber, which is in turn connected with a series of ten other chambers, arranged in zigzag fashion, which form a system of accumulation. The arsenious acid gases are condensed on the walls in the form of a white so-called arsenic floss. The floss is of a dark gray color on account of which it cannot be marketed before it is submitted to a second treatment in a smelting furnace similarly connected to other chambers. This second process produces the pure white arsenic, which pulverized, sifted and packed, is ready for market. The condensing chambers are connected with flues up the mountain sides about 250 ft. long according to location. The analysis of the mineral treated is: PbS, 4 per cent. Sb, 2 per cent.; mispickel, 30 to 40 per cent.; silver, 400 grm.; gold 5 grm. per ton.

WORLD'S PRODUCTION OF ARSENIC.
(In metric tons.)

Year.	Canada. (a)	Germany. (b)	Italy. (d)	Japan. (a)	Portugal. (d)	Spain. (b)	United Kingdom. (a)	United States. (a)	France. (d)
1897	*Nil.*	2,987	200	13	524	244	4,232	.....	.....
1898	*Nil.*	2,677	215	7	751	111	4,241	.....	.....
1899	52	2,423	304	5	1,083	101	3,890	.....	.....
1900	275	2,414	120	5	1,031	150	4,146	.....	2,600
1901	630	2,549	.....	10	527	120	3,416	272	4,705
1902	726	2,828	.....	12	736	71	2,165	1,226	7,491
1903	233	2,768	50	6	698	1,088	916	554	5,372
1904	66	2,829	80	4	1,370	400	992	452	6,658
1905	*Nil.*	2,535	.....	8	1,562	1,140	1,552	701	3,117
1906	*Nil.*	3,052	.....	5	1,322	1,114	1,625	754	3,627
1907	317	2,904	73	7	1,538	2,400	1,523	916	6,534
1908	649	2,822	451	20	1,655	2,004	2,007	1,301	7,900
1909	1,020	2,911	*Nil.*	8	1,420	506	2,911	914	2,381
1910	1,363	3,066	*Nil.*	12	974	444	2,187	1,203	2,141
1911	1,902	2,981	16	.....	(c)	(c)	2,178	2,800	(c)

(a) White arsenic. (b) Oxide, sulphide, etc. (c) Not yet available. (d) Ore.

STATISTICS OF WHITE ARSENIC IN THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.	Pounds.	Value.
1898	.....	.....	.....	8,686,681	$370,347	$0.04¼	8,686,681	$370,347
1899	.....	.....	.....	9,040,871	386,791	0.04¼	9,040,871	386,791
1900	.....	.....	.....	5,765,559	265,500	0.04½	5,765,559	265,500
1901	600,000	$18,000	$0.03	6,989,668	316,525	0.04½	7,589,668	334,520
1902	2,706,000	81,180	0.03	6,110,898	280,055	0.04½	8,816,898	361,235
1903	1,222,000	36,691	0.03	7,146,362	256,097	0.03½	8,368,362	292,785
1904	996,456	29,504	0.03	6,391,566	226,481	0.03½	7,388,022	255,988
1905	1,545,400	50,225	0.03¼	6,444,083	219,198	0.03⅜	7,989,483	269,425
1906	1,663,000	83,150	0.05	7,639,507	336,609	0.04½	9,302,507	419,753
1907	2,020,000	101,000	0.05	9,922,870	553,440	0.05½	11,942,870	654,449
1908	2,603,505	99,193	0.03¾	9,592,881	417,137	0.04⅜	12,196,387	516,330
1909	2,015,880	57,957	0.02⅞	7,183,644	272,493	0.03¾	9,199,524	330,450
1910	2,652,000	66,300	0.02½	8,204,123	250,115	0.03	10,856,123	316,410
1911	6,162,000	123,240	0.02	6,916,069	203,766	0.02⅞	13,078,069	327,006

Rhodesia.—In the Umtale, Gwanda, Mazoe and Victoria districts, mispickel is treated for the gold and silver that it contains. Recovery of arsenic by roasting is not attempted. In the Enterprise district, the Ceylon, Arcturas and Slate mines also contain mispickel.

Turkey.—Gold and silver bearing pockets of arsenical ores occur in the Vilayet of Aidin. The Tishua mine has produced from a pocket rich gold ore (40 oz.) with 35 to 42 per cent. arsenic contents.[1]

Electrolytically Refining Alloys.

(By Anson G. Betts, Troy, N. Y.)—The arsenic-antimony alloy is dissolved as anode in a solution containing antimony and arsenic trifluorides, an alkaline fluoride and hydrofluoric acid. When the solution is rapidly agitated arsenic deposits out faster than antimony and the cathode metal may thus contain more arsenic in proportion than is dissolved from the anode, until the solution becomes impoverished in arsenic. If now the solution is kept at rest and electrolysis continued, an alloy is produced for a time, containing a less proportion of arsenic than is dissolved from the anode. When arsenic has sufficiently accumulated in the electrolyte, the first operation with rapid agitation is repeated. In practice there should be at least two refining tanks in use, through which the solution flows continuously from one to the other, and in one of which the solution is well agitated, while in the other it is nearly at rest. The alloys deposited in the two tanks are different from each other and from the anode and are generally also free of many impurities of the anode, as silver, lead, copper, etc. About one-half of the arsenic in the anode remains in the anode slime.

Combination with iron and with manganese results by driving off some arsenic heating a higher arsenide in a stream of hydrogen. The lower arsenide of manganese has strong magnetic properties, whereas the two iron arsenides made do not show this property.[2]

Arsenic

Recovery of Arsenic from Ores.—An improvement in handling arsenical vapors is claimed in a patent recently granted to Ira F. Peck, of Providence, R. I. The arsenic-bearing material is roasted in a revolving drum and the vapors issuing therefrom are conducted through a series of tanks which contain a thick creamy mixture of lime and water. A vacuum applied at the last tank assists in causing the gas to flow through the lime mixture where the arsenic is absorbed. When the mixture in any

[1]A. I. M. E. W. F. A. Thomae, Vol. XXVIII, p. 208.
[2]Berichte, XLIV, 2,378.

tank has taken up as much arsenic as possible it is withdrawn and replaced with a new mixture. The composition of the lime mixture is about 25 lb. lime per 100 gallons of water. The recovered arsenic can be used in the manufacture of insecticides. (996,516, June 27, 1911.)

Distillation Process for Arsenic and Antimony.—Distillation at reduced pressure is one of the features of a process for treating ores containing antimony and arsenic, devised by Dr. Colin G. Fink, of the General Electric Company. It may be applied, for instance, to electrolytic slimes containing antimony, and a very pure antimony can be obtained from a slime containing 35 per cent. of antimony, the yield being almost 95 per cent. At atmospheric pressure the vaporization point of antimony is between 1500 and 1700° C. At a pressure of 5 to 6 mm. mercury it is only 735° C. At a pressure of 40 mm. mercury, as used in the author's process, it is between 1200 and 1300° C. For the treatment of stibnite, $Sl_2 S_3$, using an ore containing 62 per cent. of antimony, 11.3 parts of stibnite, 12 parts of limestone, 2 1/2 parts of coke and six parts of fluorspar are ground to 20 mesh and charged into an air-tight electric resistance furnace, the charge serving as a resistor. The temperature is raised to 600 or 700° C. and maintained there for one and a half hours to let the following reaction proceed:

$$Sb_2S_3 + 3CaCO_3 + 6C = 2Sb + 3CaS + 9CO$$

A vacuum of 40 mm. mercury is maintained by constant operation of the air pump. Then the temperature is raised to 1300° C. and maintained there for about one hour, while the antimony is distilled off and condensed in a water-cooled condenser which is in communication with the furnace chamber.

The reduction of the stibnite may also be accomplished by heating the powdered ore with finely divided iron. Two parts by weight of stibnite are mixed with one of iron. The temperature of the charge is raised to 700° C. for 20 minutes, to reduce the stibnite:

$$Sb_2S_3 + 3Fe = 3FeS + 2Sb.$$

The temperature is then raised to 1200 or 1300° C. for an hour and the antimony distilled and condensed. Instead of metallic iron, an oxide ore, as hematite or magnetite, may be used, the oxygen of the hematite serving partly to unite with the sulphur of the stibnite to form sulphur dioxide, SO_2 and the remaining sulphur combining with the reduced iron to form sulphide of iron. From two to three parts by weight of powdered carbon to one part of ore are added to the charge, to increase its conductivity and to permit the carbon monoxide formed to keep the charge porous.

Instead of an iron ore, a chrome iron ore, like chromite, may be used, $Fe\ O\ Cr_2\ O_3$. For example, about four parts of finely ground stibnite are mixed with two parts of finely ground chrome iron ore. One part of powdered coke or charcoal is added for the purposes mentioned. The finely ground mixture is heated to 900° for one or two hours, to reduce the antimony and chromium.

ASBESTOS

In the United States the asbestos industry is being put on a firm basis and although production and plants are increasing, as compared with the Canadian Industry, the output is but a small amount. The United States is undoubtedly the most important user of asbestos in the world, while the raw material is mined principally in Canada.

In 1910 activity began in the Wyoming field, which continued in 1911 and will lead to the establishment of a large industry in the west. As given by the report of J. S. Diller[1] the production in 1911 is 7604 tons of 2000 lb. valued at $119,935 or an average price of $15.77, a lower price owing to the predominance of lower grades. The imports of crude asbestos were 52797 tons in 1910 and 62418 tons in 1911 valued respectively at $1,235,170 and $1,413,541. In 1911 the manufactures imported in addition were valued at $290,098 and in 1910, $308,078. In the second half of 1911 the exports of manufactures were worth $252,550. Most grades produced held their prices in 1911 but prices of the higher grades were somewhat lower than the following schedule:

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.		
	Short Tons.	Value.	Value per Ton.	Manufactured.	Unmanufactured.	Total.
1898.....	885	$13,425	$15.17	$12,899	$287,636	$300,535
1899.....	912	13,860	15.20	8,946	303,119	312,068
1900.....	1,100	16,500	15.00	24,155	331,796	355,951
1901.....	747	13,498	18.08	24,741	667,087	691,828
1902.....	1,010	12,400	12.27	33,313	729,421	762,734
1903.....	(a) 887	(a)16,760	(a)18.90	32,058	657,269	689,327
1904.....	(a)1,480	(a)25,740	(a)17.40	51,290	700,572	751,862
1905.....	3,100	126,300	40.74	70,117	776,362	846,479
1906.....	(a)1,695	(a)28,565	(a)16.85	96,162	1,010,453	1,106,615
1907.....	950	11,700	12.32	200,371	1,104,110	1,304,481
1908.....	1,350	24,000	17.78	147,548	1,068,342	1,215,890
1909.....	4,025	87,625	21.77	240,381	993,254	1,233,635
1910.....	3,630	64,350	18.80	308,078	1,235,170	1,543,248
1911.....	(a)7,604	(a)119,935	(a)15.77	290,098	1,413,541	1,703,639

(a) Statistics of the United States Geological Survey.

Market and Prices.—Asbestos is graded so there are six products for the market known as Nos. 1, 2, 3, 4, 5 and 6. The range of prices is wide and depends largely upon the matter of making a good contract, as well as the quality of the material.

[1] United States Geological Survey.

No. 1, Crude, per short ton	$250 to $350
No. 2, Crude, per short ton	150 to 250
No. 3, Asbestos fiber, according to grading	50 to 150
No. 4, Asbestos fiber, according to grading	25 to 50
No. 5, Asbestos fines, according to grading	10 to 25
No. 6, Asbestos fines, according to grading	1 to 10

The occurrences of asbestos in the United States are described in Bulletin 470 of the United States Geological Survey.

True asbestos is a fibrous form of tremolite or actinolite, minerals that are essentially calcium magnesium silicates with a little iron in the latter; another mineral occuring in asbestos form is a magnesium iron silicate, namely anthophyllitte. The Canadian asbestos of commerce is serpentine.

Arizona.—One of the requirements as an insulating material is the absence of magnetite in the product and though this can be separated by special milling it is preferable to have no magnetite in the crude material. The asbestos in the Grand Canyon originating in limestone is known to be particularly free from iron in this form. Only trial shipments have been made. The most important deposit extends at least one mile and some development work was in progress. J. S. Diller states that the proportion of high-grade material obtainable by milling is greater than in any deposit in the United States. Water-power of Shinumo Creek could be developed but the transportation problem is difficult to solve, the canyon being very inaccessible.

Georgia.—The Salt Mountain Asbestos Co., produces an anthophyllite asbestos which is all fibrous, the grade being uniform but brittle, and the fiber short. The mill is three miles from the mine and the nearest railroad point is Clarksville 12 miles distant. Only 10 per cent. waste is incurred is the manufacture of fibrous product.

Idaho.—The Spokane Asbestos Firebrick Co. is quarrying anthophyllite asbestos 12 miles from Kamai on the railroad. This material is used in the manufacture of pipe covering and wall plaster, the plant being located at Spokane. This variety of asbestos is cheaper than the chrysotile, and its occurrence is more massive and abundant in the rock mined.

Vermont.—There is only one producing plant in the State, although there are a number of outcrops upon which some work has been done. An analysis of the chrysotile variety from the vicinity of Chrysotile, is as follows: H_2O, 13.43 per cent.; SiO_2, 38.19; Al_2O_3, 3.79; Fe_2O_3, 3.30; FeO, 1.65 and MgO, 38.19 per cent. The Lowell Lumber and Asbestos Company at Chrysotile, is producing annually about 2500 tons of all grades.

The feature of 1911 was the doubling of the output at the mines of the Chrysotile Asbestos Co., a new corporation that has taken over

the properties of the Lowell Lumber and Asbestos Co. The ore reserves have not been neglected and though a lower average value was obtained for the product the tonnage was increased 130 per cent. The Company proposes with largely increased capital to do a larger business in the future, and to install new mills, transportation facilities, etc. Some of these are already under way. Sales are increasing rapidly. A small amount of higher-grade material, No. 2 crude, was also produced in 1911. The mines at Chrysotile are managed by W. G. Gallager. The asbestos is of the cross-fiber chrysotile variety and the mines are located on the slope of Mount Belvidere, near Lowell, in the northern part of the State, the rocks of the Canadian asbestos belt crossing the border to this place.

Wyoming.—The two localities near Caspar, Natrona Co., Wyo., have been worked for asbestos of the chrysotile variety, the same as that of Vermont and Canada. Caspar mountain is the general name for the western end of the Laramie range. It consists of a granite core, cut by serpentine and diorite dikes. The general trend of the granite core is east and west. The serpentine dikes have a southwest trend and vary in width from 100 ft. to over 500 ft., the asbestos-bearing zones being well defined and showing from a streak to 50 or 60 ft. in width. The asbestos is of high tensile strength and free from iron and other injurious minerals. It is capable of being spun. The Company has a fiberizing mill on upper Smith Creek, with a capacity of 200 tons of asbestos rock per 24 hours. Work during the spring of 1910 put the quarries in shape to produce a large tonnage. The method consists in open quarrying, rough sorting the long or No. 1 crude, and sending the mill rock to the mill below. The mill rock runs from 5 to 10 per cent. fiber. The first shipment of asbestos from Wyoming was made Oct. 8, 1910. Another plant was also operated and erected in 1911 by the Northwestern Mills and Power Co., but it is understood these are now under one management of the International. The Wyoming Asbestos Company has been developing its property on Caspar mountain and is preparing to erect a plant to manufacture fireproof materials.

The Wyoming rock carries about 35 per cent. fiber and in certain localities veins 300 ft. wide may be traced for miles. The Canadian fields produce only a small proportion of No. 1 and No. 2 fiber, while much of the rock thus far taken from the Wyoming veins will carry spinning fiber. Samples from the Caspar district revealed a moisture loss of only 10.3 per cent., while the loss to the Canadian product under the same test was 12 per cent. The presence of the rock is indicated by outcroppings from Caspar mountain to Laramie peak, a distance of more than 100 miles. The quantity and quality of the fiber are such

that the region will certainly supply the demand of the western consumers.

The work on the Caspar mountain properties was carried on during 1911 to the extent that the ore house was filled with ore to the roof and the tunnels were practically filled with milling ore. It was then decided that it would necessitate an extra handling of the ore to mine any more, and the work temporarily ceased until the mill could be operated and the ore mined gotten out of the way and marketed.

Asbestos in Foreign Countries

Australia.—According to Mr. Soanes, Geologist of Perth, the Western Australia deposits are similar to the Canadian deposits. They occur in a serpentine lode 8 to 10 ft. wide and good shafts have been sunk to a depth of 140 ft. The crystallization of the chrysotile variety, according to Fritz Cirkel, occurs at right angles to the walls. The brilliant wavy luster rarely found in Canadian asbestos is characteristic. The tensile strength required does not come up to the standard and test lots sent to Europe will not fulfill the requirements for manufacturing interests. Another sample shows the crystallization of the fiber embedded in serpentine, and it goes to show that the solidification of the serpentine took place after the formation of the fiber.

New Zealand.—According to Charles H. Holland,[1] the increased transportation facilities would make the deposits of New Zealand a formidable competitor in the asbestos world. The mining operations are now carried on in the Nelson province in the Upper Takaka Valley. The distance from Nelson is 28 miles by road and an additional trail of 16 miles. There is a nearer port only 18 miles away from the mines. The asbestos occurs in serpentine or limited area and is covered in places with 30 ft. of till. The principal asbestos veins occur in elliptical sections in serpentine shells of pyroxene masses which sometimes reach a diameter of 70 ft. The following analysis of commercial asbestos was made by McLaurin:

Silica	41.12
Alumina	.68
Ferrous oxide	1.54
Manganous oxide	.35
Magnesia	42.52
Soda	.16
Potash	.16
Water	13.90

A feature of the Takaka asbestos is its continuity across the fissure, without the break which is such a conspicuous habit of the veins of Quebec. The fibers run at right angles to the walls. The quarrying is done on benches and hand and machine drills are used. Holes eight

[1] *Mining and Engineering Review*, Sept. 5, 1911.

to ten feet deep are common. The blocks produced are lifted by crane onto the trucks so as to avoid fines. After blocking a rough sorting takes place, allowing it to be transported to a cobbing-shed and for future mill treatment. The cobbing is done with 1 1/2 lb. hammers and the material containing fibers one inch long or over, is graded into No. 1 and put up in 100-lb. bags for the market. The price of this grade varies somewhat as some offers include fibers of less length. The process is not very efficient and there must be an enormous loss. The European asbestos manufacturers express satisfaction with the spinning and weaving qualities of the fiber.

Canada.—For a number of years past the annual output of asbestos has exceeded the sales. In 1911, however, the sales have been greatly increased but at considerably reduced prices. Returns received for the year 1911 show a total output of 96,299 tons as compared with 100,430 tons in 1910. The sales in 1911 are, however, reported as 100,893 tons valued at $2,922,062 or $28.96 per ton, as compared with 77,508 tons valued at $2,555,974, or an average of $32.98 per ton in 1910, an increase of 23,385 tons or 30 per cent. in quantity but only $366,088, or 14 per cent. in total value. Stocks on hand at December 31, 1911, are reported as 34,568 tons, valued at $1,509,100 as compared with 41,903 tons, valued at $1,943,171 on December 31, 1910.

The average number of men employed in mines and mills during 1911, was 2707 at a wage cost of $1,231,896.

The total quantity of asbestos rock sent to mills is reported as 1,484,691 tons which, with a mill production of 91,237 tons, shows an average estimated recovery of about 6.14 per cent.

The following tabulated statement shows the output and sales during 1911 and the stock on hand at the end of the year:

	Output.	Sales.			Stock on Hand Dec. 31.	
	Tons.	Tons.	Value.	Per Ton.	Tons.	Value.
			$	$		$
Crude No. 1	1,467.9	1,301.4	342,855	263.45	1,256	327,508
Crude No. 2	3,594.5	3,562.7	402,107	112.86	3,222.7	404,198
Mill stock No. 1	20,376	18,315	916,678	50.05	8,471	380,570
Mill stock No. 2	39,289	47,326	991,370	20.95	17,794	365,457
Mill stock No. 3	31,572	30,388	269,052	8.85	3,824	31,367
Total asbestos	96,299.4	100,893.1	2,922,062	28.96	34,567.7	1,509,100
Asbestic	.....	26,021	21,045	81	.....	.....

In the absence of a uniform classification of asbestos of different grades the above subdivisions have been adopted purely on a valuation basis;

crude No. 1 comprising material valued at $200 and upward, and crude No. 2 under $200; mill stock No. 1 includes stock valued at from $30 to $100; No. 2 from $15 to $30; No. 3 under $15.

A decision of the directors of the Amalgamated Asbestos Corporation to default payment of the interest on their consolidated bonds came as no surprise to the public at the end of 1911. The earnings of the corporation could not provide 5 per cent. on $8,000,000 and there is no other recourse than a financial reorganization. The asbestos market has been depressed, but the mines are in excellent condition.

The asbestos slate or shingle industry has been developed by the plants in Canada, and the predictions are made that in a short time 75 per cent. of all the asbestos produced in the country will be used for asbestos shingles. The asbestos slate business is only five years old, but the demand for this material has increased enormously.

Quebec.—Despite the unfavorable conditions under which the asbestos industry labored during the year 1911, the shipments of this substance, from the producers to the consumers, show a substantial increase as compared with the previous year 1910.

The average value per ton has somewhat decreased, from $33.09 in 1910 to $29.58 in 1911. This is due in a great measure to the overproduction of the previous year which glutted the market and caused a consequent fall in prices. On the whole, from the commercial and technical standpoints, the asbestos situation is satisfactory. Some producers may take exception to this statement, but on close examination it will be seen that many of the asbestos ventures, which are now closed down or stagnating, should not have installed elaborate mining plants and put up expensive mills without doing first a great deal more preliminary development work and ascertaining more thoroughly the workability of their deposits.

The application and uses of asbestos are increasing year by year, but the absorbing power of the market is limited, and this must be taken into consideration by the producers.

As to the mines themselves, more especially those which produce a fair proportion of crude products, it can only be re-stated that they are in excellent shape and the rock shows practically no change in depth.

The shipments of asbestos in 1911 reached 102,224 tons, valued at $3,026,306. On December 31, stocks on hand, unsold, were 33,751 tons valued at $1,583,076.

For the year 1910 the figures were: shipments 80,605 tons valued at $2,667,829. Stocks on hand on December 31, 1910, amounted to 41,159 tons valued at $1,921,923.

The shipments of asbestos made during the year 1911, according to grades, are as follows:

	Tons.	Value.
Crude Nos. 1 and 2	3,429	$ 668,031
Mill stock Nos. 1, 2, and 3	77,176	1,999,798
Total	80,605	$2,667,829

New concentrating mills have been put in operation at the Bell, Jacobs, and Black Lake Consolidated Mines. These can handle 800 to 1000 tons per day. The B. and A. Asbestos Company at Robertsonville and the Berlin Asbestos Company near Leeds Station have built cyclone mills operating at each property. At Chrysotile the Belmina Consolidated Asbestos Company has acquired what was the Asbestos Mining and Manufacturing Company and re-equipped the mill and mine.

The Thetford Asbestos Syndicate of Montreal has done substantial development work at Coleraine.

One of the most important developments of the asbestos industry is the establishment, by the Asbestos Manufacturing Company, of large works at Lachine, Quebec. This plant, the only one of the kind in Canada, is designed to manufacture all classes of asbestos goods. It is expected to consume about 1000 tons of fiber per month.

The principal development was at Brompton Lake where John McCaut found more asbestos. This property was worked twenty years ago. The wider veins are usually hard and brittle, but the smaller ones contain a good quality.

China.—The Chinese have long known of asbestos, which they call "stone cotton," and deposits exist about 45 miles northeast of Antung. It is reported that these mines are working with 30 men each and the irregular output is shipped to northern Manchuria. The price at Antung is said to be 60 cents (U. S. Currency) per 100 lb., but the methods being primitive this could become lower.

Cyprus.—The Cyprian Mining Company has installed a large plant and new machinery, but has not produced extensively. The asbestos mines on the Pasho River produce a chrysotile asbestos with a fine fiber and a high elasticity. It is said to employ 500 persons during the summer and picking and sorting of the rock is carried on throughout the year, employing about 80 men. The average per cent. of fiber in the rock is about 8 to 10. The capacity of the plant is about 3000 tons per year. The asbestos deposits of Cyprus are in highly decomposed condition near the surface and no explosives are used for mining. The

size of the veins varies from 1/8 to 1 1/2 in. and the analyses of serpentine are promising. The high grade asbestos contains 2.5 per cent. iron oxide, the Cyprus variety contains 4.8 per cent. to 5 per cent.; the water content is between 1.3 and 1.4 per cent. and the silica and magnesia remain within their permissible limit.

Germany—In Germany, amphibole-asbestos is used chiefly for insulating purposes, for filters, gas stoves and chemical laboratories. Another class of asbestos is the fibrous variety, which is mostly spun or woven. It is estimated that the former kind, the raw material of which comes from German, American and Russian deposits, amounted to 500 or 600 tons per year. Russia being nearer, the ground variety finds its way into the German markets more readily than that from other countries. Prices fluctuate from $14 to $31 per ton c. i. f. Hamburg. The first grade variety, finely ground, costs as much as $35 per ton. The consumption of the second variety is about 14,000 tons per year. In the beginning of 1911, the industry was reported to have met with reverses. The International Syndicate had to be dissolved on account of competition by new factories and this caused a sharp decline in prices.

Italy.—Italy has been a regular producer of serpentine quality of asbestos, but only in small quantities. The imports in the United States have fallen to only a fraction of a ton. According to G. P. Merrill, the asbestos is of the amphibole and serpentine variety, both having long fibers. The amphibole variety comes from Mont Cenis and the serpentine from Aosta.

Italy's asbestos exports were 517 metric tons valued at 258,500 lire in 1911; the value of 485 tons in 1910 was 242,550 lire, and of 527 tons in 1909, 263,700 lire. The Italian imports have been:

	1909	1910	1911
Germany	5	5.5	59
Canada	2569	1995	2141
Total	2285	2051	3892
Valued, lire	1,599,780	1,435,500	2,724,330

A. E. Marshall[1] states that in making joints between fused silica ware used as nitric or hydrochloric acid containers, asbestos is the best material. Italian asbestos is superior to the Canadian in this respect, as the latter swells considerably when brought in contact with the strong acids.

Rhodesia.—Most of the South African asbestos has come from Cape Colony, but in 1911, Natal, Transvaal and Southern Rhodesia contributed to the output.

British South Africa now produces at the rate of 1600 tons per year

[1] Am. Chem. Society, March 8, 1912.

and the prospects are that this production will increase rapidly. Rhodesia produced 1119.5 short tons valued £12,991 in 1911.

Russia.—The Russian asbestos fields are in the Urals but asbestos has also been found in other parts of the Russian Empire including Siberia and some parts of Orenburg. In the Urals in 1910, there were 21 asbestos mines worked, the production being as follows: Voznesensky mines, 5000 tons; Govorunzhinsky mines, 2450 tons; Shtshuivi mines, 3500 tons, and others, 500 tons. The production of asbestos in the Urals, says the *Viestnik Finanssoff*, in the year 1911 has greatly increased as compared with the production in 1910. The total for 1911 was 984,380 poods. This includes the production of the mines of Poklieffsky-Kosiel, 307,617 poods; Baron Girard de Sukanton, 322,555 poods; P. O. Korieff Successors, 197,352 poods; the Russo-Italian Company, 66,380 poods; and N. V. Muchanoff & Company, 54,476 poods. The increase, compared with the production in 1910, makes 270,414 poods. The largest increases in the order of their importance were from the mines of Poklieffsky-Kosiel, Girard de Sukanton, the Russo-Italian Company, and Muchanoffs. There was a decline in the production of the Korieff concern, though not a serious one. The productions of the past five years were, respectively, as follows: 1907, 543,143 poods; 1908, 663,038 poods; 1909, 814,134 poods; 1910, 677,966 poods; and 1911, 948,380 poods. Cirkel states, in reference to the new Siberian deposits, that the rock occurs in the Yenissei on the right bank of the Kamyshto river. The asbestos formation takes in the foothills of the Psajansk mountains, being crystalline limestone intersected by bedded masses of serpentine. It appears that they are Laurentian and therefore may be compared with those in the Ottawa country. The deposits are of limited extent, the veins occurring in parallel serpentine layers. The crude asbestos is rough, yellow in appearance, but the fiber when worked up is rather delicate, though ordinary and to some extent fluffy. The estimated shipping cost to London is $35 to $40 per ton.

Kryshanofsky Asbestos Deposits in the Berezowa, etc., Estates, Urals, Russia.

By H. Aurep.

The Ural asbestos deposits are situated 30 versts (1 mile = 1.0668 versts), from Bazenowa station, Perm-Tjunenskay Railway, or 85 versts northwest from Ekaterinburg. The deposits in the Berezowa estate are owned by Baron Shejeiada and those in the Kamenskaja and Manitnaja estates by the Ural (Italian-German) Corporation.

The surface is flat and poorly drained, with many bogs. Prospecting is difficult as the nearest villages are 30 versts from the asbestos deposits.

The serpentine and allied rocks form a belt two to three miles wide and running for 30 miles north and south. It lies within a large granite area. The rocks of the belt are chlorite and talc schists, diabase, porphyrite and serpentine. The schist in places carries emeralds and other gems. The serpentine is in elliptical areas, which appear at intervals and have their longer axes running north and south; that is, parallel to the length of the belt. One of the largest bodies of serpentine is about 1000 feet wide and a mile long. Some narrower portions of the belt consist of diabase and porphyrite. In places serpentine contains veins of porphyrite. The asbestos is in veins forming a sort of stockwork running in all directions through the rock. They disappear, recur and spread out in all directions from 0 to 360° and from horizontal to vertical. Veins are frequently crushed, showing movement of the rock since the veins were formed. In places the veins are 20 cm. (8 in.) wide. The fibers run at right angles to the walls and vary in color from pure white, yellow, golden, and red, rarely to bluish and darker shades. The color does not appear to indicate any difference in other qualities. The central part of the veins contain streaks of serpentine rock rich in magnetic iron ore. The fibers are closely bound together but the cohesiveness is not equal to the Canadian asbestos. Near the surface the rock is loose and disintegrated. There is also a kind of asbestos with fibers a yard long, but these are parallel to the walls coarse and unsuitable for use. Other minerals are quartz and garnet and vesuvianite.

Asbestos was found in the Ural district in 1710 and mining undertaken soon after. A process of manufacture known as the Nevjausko process was devised and textiles, socks, gloves and handbags have been made for 50 or 60 years. The industry failed owing to the limited local demand for the products and lack of transportation. The present operations date from 1883 and have been vigorously prosecuted since 1889. The mines are principally in the hands of large capitalists and equipped with steam or electric power.

Mining is carried on in open pits, the ground being cut down in benches of about 7 ft. so as to admit tramways. The rock, especially near the surface, is so weathered that explosives are little used, except to break up harder remnants of rock. Horses are often used for moving the rock in the pits. The mining is largely done by contract. The harder rock is treated in mills where it is crushed by stamps, rolled, dried and screened and the products classified by a kind of rotary classifying screen. Fiber from the softer rock is washed, dried and screened. Five grades of asbestos are distinguished according to length of fiber

from 1/2 to 4 cm. or longer. These mechanical processes for the separation of asbestos were in use in Russia before mechanical concentration was employed in America. Owing to the severe winters, work is carried on only during the summer and then only in the intervals between farming operations. Work is usually begun in May and continued through June, and during September and October. Over 15,000 laborers were thus employed for a 6000 ton production.

Siberia.—With the construction of the Siberian railway, the Minusinsk district is attracting more and more attention. Copper and coal are being worked and delivered into the town of Krassnoyarsk. Asbestos is receiving the attention of a French company, which has acquired nine claims on the Kamyushta River. The company produced 600 poods of asbestos during 1909. The mineral was sent to Paris to be tested.

Uses.—The main features of the asbestos industry in recent times are the application of lower grades and the increased use of these which were at first regarded simply as so much waste. Asbestos is getting to be a principal ingredient for coverings and also for insulating materials such as the lining of roofs. As a substitute for plaster, it has been used to make a sound-absorbing non-reflecting surface for walls. In the latter part of 1911, a large burning oil well in Louisiana was extinguished by using asbestos wicking.[1] Its combination with cements is the opening of a great future.

C. L. Norton states that the most suitable fire-resisting material in building construction must contain a fiber of some kind held together by cement. These are conclusions after investigating other materials, such as wood treated in various ways. The processes for wood are either impregnating with a substance that yields a gas incapable of supporting combustion, or with a salt, such as sodium phosphate, which renders the wood fireproof, chiefly by shortening the flame. Fibrous incombustible materials, such as mineral wool or asbestos, have been used with all the common cements to make boards and fireproof material but all are wanting in some property. A material has been developed recently called asbestos wood which answers most requirements. It is made under certain processes of mixing, pressing and curing, which make it harder than natural woods and with a strength of about two-thirds that of white pine with the grain. Is it not brittle, although its elasticity is less than natural woods. The thermal conductivity is 50 to 30 B. T. U. per square foot in 24 hours per degree Fahrenheit.

The weight of the several grades of asbestos wood 1 in. thick varies from 8 to 13 lb. per square foot, whereas white pine 1 in. thick, when dry, weighs approximately 3 lb. per square foot, and oak about twice as

[1] *Scientific American Supplement*, Sept. 16, 1911.

much. The transverse strength of boards and the minimum fiber stress is between the limits of 5000 and 10,000 lb. Test specimens 12 in. wide, 12 in. long and 1/4 in. thick, broken by center load when supported on two edges with 11-in. span, broke on an average at 200 lb. load. Under the same conditions clear white pine broke under a load of 300 lb. with grain and 20 lb. across it. In some cases the equality of strength in both directions is of great advantage.

The tendency of the material to absorb water varies between the limits of 4 per cent. and 20 per cent. This absorption is not of such a nature or amount as to cause disintegration from freezing and the material is not injured by prolonged soaking in fresh water. Sea water tends to disintegrate it after a time, the magnesium salts in the water replacing the calcium of the cement.

The coefficient of expansion varies with the temperature and also with the age of the specimen. It is small, being about 0.000004 at ordinary temperatures, diminishing rapidly at 700° Fahrenheit, and becoming negative at 950° Fahrenheit where shrinkage occurs.

The operations of boring, sawing and finishing are all somewhat more difficult with asbestos wood than with the natural woods. It withstands scraping and rubbing much better. Floors and stair treads made of the material wear well, but are liable to be slippery when wet.

Its ability to stand heat makes it available for blocks on friction brakes where a flexible material is not needed. The coefficient of friction on iron is approximately 0.3 to 0.4.

It has also been used as a mold for glass utensils during the process of manufacture, its non-conducting properties making it preferable to iron in some instances.

Under prolonged red heat the boards become weakened, but for such exposure as occurs in fires in buildings the resistance is such that a sheet of 1/4-in. asbestos wood placed over the top of an open furnace at 1700° Fahrenheit is affected as follows: The flame impinging directly on the lower surface of the sheet, in half an hour it may be taken off and, while its lower side is still white hot, plunged into cold water and then returned to the furnace for 10 minutes. It may be slightly warped and its strength will probably be diminished 25 per cent. or more, but it will still be sound, whole and free from cracks or serious weakness. Fire tests have shown the ability of the material to withstand much longer exposure, but it is not suitable for linings for furnaces or kilns which are kept constantly at red heat. In some fire tests the paint on the outside of an asbestos wood door 1 1/4 in. thick was not scorched after the fire inside had been burning at a temperature of 1700° Fahrenheit for more than an hour.

Shingles.—The plant for making asbestos shingles in Denver, Colo., has been acquired by the Northwestern Asbestos Mills and the International Asbestos Mills and Power Co. The process of the manufacturing of these shingles is quite simple, but details cannot be obtained. The shingles are made from asbestos, cement and other secret ingredients. These are mixed in a dry state on the second floor of the plant, and go down a long shoot to an intermediate floor, which is just above the main floor. At this point it goes through a wet mixer and is made into a compound resembling cracker dough. From this it goes down a shoot upon steam plates which are upon a revolving belt, and thence through rollers which roll the material out flat. There are four sets of these rollers from which it passes upon another revolving belt at the end of the rolling mill, which moves slightly faster than the belt of the rolling mill. It is then stacked upon cars fifty high and runs on miniature railroad tracks into the power press where it is subjected to a 700 ton pressure. From the power press it goes to trimming tables, and to cutting machines which stamp the sheets into shingles, and stacks them. From the cutting machines they go by cars to tanks in the basement where they are dipped into chemicals, and from there they go to the curing rooms, which are rooms into which a constant stream of steam is allowed to pass. In from 30 to 60 days the shingles are cured and ready for the market. From the sample exhibits of the shingles, there is little question but that the demand will be very great and that it will necessitate in the near future the increasing of the capacity of this plant.

The use of asbestos shingles would annually save many times the difference in cost, between that form of roofing, and of the wooden shingles, in the amount saved by fire loss. The rate of insurance would be cheaper, and while the initial cost might be slightly higher, yet in the long run, they prove a paying interest upon the extra cost. Charles Guy is the manager of the Company and the inventor of the process. One unit to start with will turn out 112 squares of shingles per day. Considering that it took the Canadian Companies 10 years to put their properties on a profit-earning basis, the success of this enterprise will reflect great credit upon the management.

Milling methods for asbestos have been discussed in the *Journal of Canadian Mining Institute*, Vol. XVII, April, 1910. There is a great divergence in opinion on milling design but mills must be adopted that are suitable for every particular class of rock. As to the machinery, rolls have proved efficient, since they preserve the quality of the fiber. The cyclone is valuable from the view of large production and low cost, but it is a question with some of preserving the valuable qualities of fiber and only low-grade materials should go to a cyclone that acts as a

disintegrator. A new machine has been designed by Edward Torrey and installed by the Asbestos and Asbestic Company of Danville, Quebec. It works rather by impact than pulverization, which is the principle of the old style cyclone, where the beaters revolve in opposite directions. The operation of the new machinery is also stated to be less costl . See also under Ore Dressing, paragraph on "Breaking Asbestos Bearing Rock."

ASPHALTUM

The production of asphaltum and asphaltic rock in the United States in 1910 increased about 10 per cent. over that of 1909, the figures reported by the United States Geological Survey being 228,655 short tons valued at $2,138,273 in 1909 as compared with 260,080 short tons, valued at $3,080,067 in 1910. Of the total amount produced in 1910, California contributed 143,486 short tons of oil-asphalt and bituminous rock; Utah, 35,697 tons, the majority of which was gilsonite; Oklahoma, 11,959 tons of oil-asphalt and bituminous rock; Kentucky, 11,225 tons of bituminous rock and Texas, 57,713 tons, practically all of which was oil-asphalt.

The occurrence and nature of grahamite are discussed in a paper by Clifford Richardson[1], giving many analyses of this material from various localities. Grahamite occurs in Oklahoma, but none is reported

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES. (*a*)
(Tons of 2000 lb.)

States.	1909.			1910.			1911.		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous Rock.									
California	33,788	$114,869	$3.40	37,547	$128,212	$3.45	27,507	$89,264	$3.25
Kentucky	15,165	78,041	5.13	9,938	53,703	5.40	23,821	71,955	3.02
Oklahoma (*c*)	6,423	12,846	2.00	11,959	65,244	5.46	.....	.....	.....
Arkansas	.....	.....	.....	.....	.....	.....	.....	.....	.....
Utah	.....	.....	.....	5,110	153,398	30.	.....	.....	.....
Total	55,376	$205,756	(*g*) $3.72	64,554	$400,557	(*g*) $6.20	51,328	$161,219	$3.15
Asphaltum (*b*)									
California	93,327	811,979	8.70	105,463	1,204,854	11.42	163,438	2,015,900	12.30
Oklahoma (*c*)	102	2,547	25.36	.....	.....	.....	52,650	315,900	6.00
Utah	.....	.....	.....	755	43,035	57.	.....	.....	.....
Texas	46,304	857,204	18.50	57,713	1,040,825	18.07	55,836	786,735	14.05
Total Asphaltum	139,433	$1,671,730	(*g*) $9.57	163,931	$2,288,714	(*g*) $12.47	271,914	$3,117,842	$11.45
Gilsonite (*d*)									
Utah	28,919	231,086	8.26	29,832	372,900	12.47	30,846	516,614	16.70
Oklahoma (*c*)	3,894	32,737	8.66	.....	.....	.....	5,000	15,000	3.00
Mastic									
California	.....	.....	.....	476	5,670	11.90	.....	.....	.....
Kentucky	733	6,964	9.50	1,287	12,226	9.50	.....	.....	.....

(*a*) From the Mineral Resources of the United States. (*b*) Includes hard and refined, or gum, liquid or maltha, and oil residues. (*c*) Indian Territory included before amalgamation into one State. (*d*) Includes gilsonite, elaterite, grahamite, ozokerite, and "tabbyite." (*g*) Average value per ton.

[1] "Grahamite, A Solid Native Bitumen," *Journ.*, Am. Chem. Soc., Sept., 1910; pp. 1032–1049.

WORLD'S PRODUCTION OF ASPHALT AND BITUMINOUS ROCK. (a)
(In metric tons.)

Asphalt.

Year.	France.	Germany.	Hungary.	Italy. (c)	Spain.	Trinidad. (b)	United States.	Total.
1901...	20,391	90,193	2,878	31,814	4,182	173,707	19,882	343,047
1902...	21,000	88,374	2,773	33,684	6,034	167,253	36,923	356,041
1903...	21,500	87,454	2,422	35,757	4,675	196,883	54,521	403,212
1904...	22,000	91,736	2,221	34,227	3,463	137,089	77,250	367,986
1905...	20,000	115,267	247	26,838	5,805	116,735	68,935	353,827
1906...	28,231	138,059	4,111	34,386	6,229	132,381	94,316	437,713
1907...	33,000	126,649	3,920	38,568	8,643	147,051	161,783	519,614
1908...	41,000	89,009	4,818	34,761	9,231	136,583	134,273	449,675
1909...	44,800	77,537	5,054	39,165	6,582	146,013	157,984	477,185
1910...	38,500	81,208	4,994	(d)38,000	(d)9,000	161,587	177,928	511,217

Bituminous Rock.

Year.	Austria.	France.	Hungary.	Italy.	Spain.	United States.	Total.
1901.....	541	250,000	25,161	104,111	3,956	37,393	421,162
1902.....	897	258,000	24,873	64,245	6,301	35,072	389,388
1903.....	1,273	243,000	21,552	89,690	6,277	37,334	399,126
1904.....	1,435	277,000	17,660	111,390	3,761	19,454	380,700
1905.....	4,363	191,509	19,372	106,586	5,725	32,337	359,892
1906.....	2,840	196,375	34,664	130,825	7,794	21,848	394,348
1907.....	3,858	177,000	33,096	161,126	8,219	41,301	424,600
1908.....	3,695	171,000	72,972	134,163	12,373	33,901	428,104
1909.....	2,975	169,000	(d)70,000	111,067	5,284	50,243	408,569
1910.....	1,066	170,000	(d)70,000	162,212	(d)10,000	58,570	471,848

(a) Statistics of production in Barbados, Cuba, Mexico, Russia, Switzerland, and Venezuela are not available. (b) Exports. (c) Including mastic and bitumen. (d) Estimated.

from there in 1910, the last year for which statistics are available. It occurs further, in West Virginia; in Cuba at Bahia Honda, and Campo Florida; in Mexico at Huasteca.

Asphalt occurs in two types of deposits that are workable. The first are practically pure beds, separated out bitumen, as at Trinidad and the Red Sea. The second are those occurrences bound up in limestone or sand-stone as in the Simmer pits in Hannover, Vorwohle in Brunswick, Germany, and Travers, Switzerland, also Razusa, Sicily, and on the Volga in Russia.

A third type occurs in commercial quantities in petroleum and oil shale deposits and exudations therefrom. The last as in Cuba and California grade into the pure type of Trinidad or into the Travers type depending on the amount of asphaltic base.

The principal use of these products is for roads, the improvement of which by this material has been a feature of recent years.

Imports and Exports.—The imports of asphaltum into the United States in 1910 amounted to 166,375 long tons, valued at $775,674 as compared with 134,939 tons, valued at $646,655 in 1909. In 1911 the

imports exceeded those of any previous years, being 174,234 long tons with an import value of only $778,185. This value per long ton in 1910 was $4.67; in 1911 it was $4.47. Of the total imports, the largest part is from Trinidad which in 1909 amounted to 99,250 long tons; in 1910, the import price averaged $4.40. The exports of foreign asphaltum from the United States were 3402 long tons valued at $48,552 in 1911; 5830 long tons valued at $29,942 in 1910 and in 1909, 6866 tons valued at $48,375.

The exports of domestic unmanufactured asphalt during the last half of 1911 were 16,925 long tons valued at $360,708. In addition manufactured American asphalt valued at $238,222 was exported during the same six months.

The declared exports of asphalt from Trinidad were valued at $870,649 in 1910 and $950,000 in 1911. The production, being from Trinidad Lake, the property of the Barber Asphalt Co., was 110,000 tons in 1910, an increase of 8000 tons per year. The asphalt exported in 1911 amounted to 169,127 tons. Of this, the crude lake amounted to 134,872 tons and 88,000 tons went to the United States. In 1910 the total tonnage exported was 158,983 tons. The value of the crude lake asphalt is about $5. per ton. The purified asphalt exported in 1911 amounted to 19,283 tons, valued at about $9.80 per ton.

The use of asphaltic oils for sprinkling over macadam roads to prevent dust and overcome the damage done by self-propelled vehicles which tear up the road, is more or less in its infancy. It promises, however, to solve the problem.[1]

Four kinds of asphaltic oils are now prepared for this special purpose by taking the volatile matter out of Texas oils or other asphaltic oils, until the residue contains 30 to 60 per cent. of asphalt, according to the requirements.

Chiefly because of the demand for oil for this use the production of natural asphalts as well as "oil asphalts" is rapidly increasing. The increase in the production of asphaltic material in 1910 was chiefly in the production of road oil, which rose from 128,861 tons in 1909 to 159,424 short tons in 1910. This product was worth $2,207,937, as against $1,558,463 in 1909, which shows an advance in price as well as quantity, indicating active demand.

Evidently the new sprinkling or "penetration" method of applying asphalt to roads is permanently established. There are three methods of applying it to the road surface, the mixing, penetration, and surface application. The first is more expensive than the others in first cost. The materials are mixed and heated, taken to the point on the road where

[1] Address Sec'y. Agriculture, to First Ntl. Good Roads Congress.

they are spread and rolled. There are cases where this older method has not given satisfaction, but mostly on account of large investment in machinery, etc. In the second method a coat of pitch is applied to the wearing course just before the usual flushing, watering and rolling is done as in modern waterbound work. The simplest method of all is the last which consists in cleaning from the made surface all fines and refuse, if necessary by washing, then applying the pitch, generally warm. The process of applying the pitch and chips may be repeated, covering with grit in each case and rolling. The convenience of this method is bringing it into favor. The proper use, however, of all the various grades of bitumen, and also of their method of application will only be brought out by further study on the part of engineers and road builders.[1]

In the laying of asphalt pavements, the paper by Mr. Richardson, who is said to be the chief chemist of the Barber Asphalt Company, gives a general outline. Generally three courses are used and concrete gutters when used should be attached with water-tight joints. The top of the surface course is much finer and usually consists of clean sand, and fine and graded mineral dust, coated with asphalt and rolled down. The object of this course is to prevent dust and mud, deaden sound, resist wear, and break jars. The elasticity of the material is the important factor in its use for this purpose. A number of standard tests are outlined and much fault is found with the natural rock asphalt for pavements in the United States. The Oklahoma Geological Survey states that a number of pavements made from the natural rock asphalt in various cities have, in every instance, outworn the artificial pavements in the American cities. Preliminary steps in either case for grading, etc., are the same. The artificial material is made free from all asphaltic bitumen. After sizing, the aggregates are mixed with enough asphalt, while hot, 3 to 6 per cent. being used, to coat each particle. Two forces are continually at work destroying the elasticity and viscosity of the asphalt, making it more brittle than cement and rendering the surface dusty. In the case of natural rock asphalt, the one deteriorating factor of self-consumption is removed because the rock is saturated. The other factor, loss by evaporation, is common to both the natural and artificial products. In this case the surface mixture is composed of sandstone, mixed with the finest limestone scales. This is heated and rolled down in the same manner as the artificial mixture. There is no doubt that there is no necessity for laying natural rock asphalt pavements in the same way as artificial, but in order to compare them, they must be laid in the most suitable manner in each case. The comparisons thus far made are not conclusive and methods for each of the many products have not yet been standardized.

[1] W. W. Crosby. Am. Association Highway Improvement, Richmond, Va., Nov. 21, 1911.

The following table[1] shows the characteristic developments in the use of bituminous materials during the years 1908, 1909, 1910 and 1911 under the supervision of the highway departments of Maine, New Hampshire, Massachusetts, Rhode Island, New York, New Jersey, Pennsylvania and Maryland. The quantities are square yards.

SURFACE TREATMENT OF ROADS.

Tars and tar-asphalt compounds.	Fluxed native asphalts, oil asphalts and residual asphaltic and semi-asphaltic oils.	Light oils.
1908..... 57,700	1908..... 239,500	1908.....
1909..... 95,500	1909..... 910,600	1909.....4,125,900
1910.....123,400	1910.....2,434,200	1910.....9,890,400
1911.....433,000	1911.....7,980,000	1911.....3,765,000

BITUMINOUS PAVEMENTS CONSTRUCTED BY PENETRATION METHODS.

Tars and tar-asphalt compounds.	Fluxed native asphalts, oil asphalts and residual asphaltic and semi-asphaltic oils.	Light oils.
1908..... 37,800	1908..... 25,200	1908.....
1909.....170,200	1909.....2,077,400	1909.....
1910.....339,300	1910.....4,840,200	1910..... 26,500
1911.....122,000	1911.....8,600,000	1911.....

BITUMINOUS PAVEMENTS CONSTRUCTED BY MIXING METHODS.

Tars and tar-asphalt compounds.	Fluxed native asphalts, oil asphalts and residual as phaltic and semi-asphaltic oils.	Light oils.
1908..... 52,100	1908..... 4,400	1908.....
1909.....136,000	1909..... 219,500	1909.....
1910.....158,000	1910..... 432,600	1910.....
1911..... 24,350	1911..... 508,100	1911.....

The most important developments during 1911 were the increased use of the mixing method of constructing bituminous concrete pavements due to the introduction of economical and efficient mixing plants and less fear of injunction and law suits brought by patentees; the manufacture of many types of gravity and pressure distributors for use in the construction and maintenance of bituminous surfaces and bituminous pavements built by penetration methods; the demand by many engineers for bituminous materials for bituminous surfaces which will set up in from 24 to 48 hours; increase in recording scientific data such as recommended by the special Committee on the Use of Bituminous Materials of the American Society of Civil Engineers.

[1] Use of Bituminous Materials in Construction and Maintenance of American Highways during 1911. A. H. Blanchard, American Association Advancement Science, Dec., 1911.

Ozocerite.—By Consul General Charles Denby, Vienna, Austria. Ozocerite was exported from Austria in 1910 to the amount of 25,540 centners [the metric centner, or quintal, is equivalent to 220.46 lb.], worth about $700,000. Of these shipments, 22,500 centners went to Germany, 1475 to France, and 536 to the United States. There was in the same year an import of 160 centners, valued in the customs returns at about $4000, of which 101 centners came from Asiatic Russia, the remainder being re-imports.

Ceresin was exported in 1910 to the extent of 11,637 centners, valued at about $310,000, chiefly to Germany (3192 centners), Italy (1248 centners), Spain (1061 centners), and 471 centners to the United States. The imports of ceresin into Austria in 1910 were 188 centners, of which 11 centners were from Brazil, the balance being re-imports from Germany.

Consular records show a declared value of $312,309 for the shipments of ozocerite and ceresin that were invoiced through the American consulates in Vienna, Prague, Reichenberg, Carlsbad, and Trieste during 1910.

Ozocerite is not separately recorded in the American official statistics, imports of this substance being included under the heading "mineral wax." In the calendar year 1911 the United States imported 5,280,363 lb. of mineral wax, worth $393,621, in contrast to 7,880,697 lb., valued at $606,040, in 1910.

Ozocerite, or native paraffin, a wax-like substance, was originally found in Moldavia and Austrian Galicia and more recently in Emery and Uinta Counties, Utah, where it occurs in the form of small veins. It melts between 56° and 63° C. It finds some use in the manufacture of candles as an adulterant of or substitute for beeswax and in the concoction of ointments and pomades. A residual product obtained in purifying ozocerite, having a hard waxy nature, is combined with india rubber and used as an insulating material (okonite) for electric cables. Another special use of ozocerite is its employment, in the form of bottles, as a container for hydrofluoric acid.

According to Lach (*Chem. Zeit.*, XIII, 831), the so-called ozocerite from Colorado does not yield ceresin, but is suitable for paraffin-making. It begins to distill at 360° and yields 90 per cent. paraffin and oil and about 5 per cent. of residuum. Caucasian ozocerite yields about 58 per cent. of a second-quality ceresin.

The Utah deposits in the vicinity of Colton, near Soldier Summit, and near Midway Station, Colton being 7 miles southeast and Midway 3 miles west of Soldier Summit, occur in shales, shaly sandstones, and limestone strata in the lower part of the Tertiary deposits of the "Wa-

satch" group.[1] The ozocerite had been exploited commercially at five or more localities, five mines having been equipped with more or less complete facilities for exploitation and three plants having been established for separating the product from the associated rock materials.

The manner of separating ozocerite from the associated rocks is a simple process. The plant consists of a steam boiler and engine, a crusher, and steam-heated vats. The soft rock and ozocerite mixture is crushed and run into long vats with narrow bottoms containing water kept at a boiling temperature. The ozocerite melts at a temperature of 54° to 70° C. and floats off as a liquid into cooling vats, while the rock is driven out along the narrow bottom of the vat by revolving screws. On cooling, the ozocerite is remelted in dry pans to remove the content of moisture.

The American Ozocerite company, of Portland, Maine, has filed its articles of incorporation with the county clerk at Provo, Utah. The company is capitalized at $1,000,000. Clarence E. Eaton is president. The company will develop ozocerite claims in the southeastern portion of Utah County.

Oklahoma.—The deposits of rock asphalt in Oklahoma are of importance, being the most extensive in the United States. The area of occurrence lies south of the Canadian River, extending from Arkansas to the Wichita Mountains. In Marshall County, especially, a large number of lens-shaped deposits occur in the Cretaceous Trinity sand, at no particular level, as this formation is 400 ft. thick. The deposits do not, as a rule, occur on faults, and the faults when they do occur are not the distributors of the bituminous material carried from original oil pools. There are exceptions to this in the Permian Redbeds of Stephens and Jefferson Counties, where the faults are impregnated locally.

In Jefferson County the Tar Springs Asphalt Co. hauls bituminous sand rock to Comanche, from where it is shipped. Much prospecting has been done in Garvin County, the region originally reported on by the U. S. Geological Survey in 1901. In Carter County the Downard Asphalt Co. of Ardmore operates a number of quarries, hauling the material to Overbrook. The American Mineral Wax Co. extracts the bituminous matter from the sandstone it quarrries on its 1280-acre tract near Woodford. The Brunswick district in Murray County comprises land covered by the lease of the Brunswick Asphalt Co. The Downard Asphalt Co. has been the only operator obtaining here a supply of lime asphalt. In Johnston County, near Ravia, the Ravia Asphalt Co. originally operated under a lease. The product was sand asphalt.

[1] *Bulletin* 285, U. S. G. S.

It is reported that the Barber Asphalt Company controls this land. One of the largest deposits of sand asphalt is at Ada, in Pontotoc County, where large shipments are made annually from a number of pits. The average price for the material received in 1910 was about $2.20 per ton on cars. The deposits appear to be an old oil sand, the paving material being local. In the Roff district, the Farmer Asphalt Company, though holding a lease, has not operated recently.

The Oklahoma Asphaltites, or locally known as coal asphalt, including grahamite, albertite, etc., occur in the northwestern part of the Ouachita Mountains and extend from Page, Le Flore County, to Atoka. The principal deposits are the Jackfork Valley, worked by the Ft. Smith Asphalt Co., and the Jumbo or Impson Valley and the McGee deposits. The first of these is a vein up to 25 ft. in thickness. In all these veins, while the mineral has been characterized as grahamite since it generally has a hackly fracture and dull luster, variations to other asphaltite minerals must be expected even in the same vein.

Kentucky.—The asphalt rock is found in upper subcarboniferous formation about 10 miles west of the Mammoth cave.[1] The material is a fine-grain, sharp silicious sandstone, each grain of which is coated with from 8 to 15 per cent. of bitumen; the commercial product must contain at least 8 per cent. It is found in irregular beds or pockets from 5 to 20 ft. thick, and is often quarried in open cut. The material is put through a jaw crusher and then through two sets of rolls. The ground material is spread cold on the prepared macadam surface to a depth of 2 in. and rolled until the interstices are filled. The cost of bringing a road to grade with 4 in. of broken stone, rolling once, covering the layer 1 1/2 in. thick rock asphalt, and thoroughly rolling, is given as 47.6 cents per square yard. A mastic is also made from this material consisting of 75 per cent. of the crushed rock asphalt, 20 per cent. of ground limestone and 5 per cent. of pure bitumen heated to about 250° F. The mixture is molded into blocks weighing 50 lb. They are used for water-proofing subways, etc.

Utah.—The production of gilsonite has not materially increased in 1911, though the value of the product is greater per ton than in 1910. The values as reported averaged in 1907, $26.22; 1908, $5.27; 1909, $8.26; 1910, $12.47 per ton.

Cuba.—It is stated that the bitumen deposits of Cuba are exceptionally pure and of high quality. The deposits are expected to prove superior to any others in the world.[2] In 1910 the Anglo Cuban Bitumen and Asphalt Co. and the resources of its 1400-acre tract, which also contains some oil, are being developed.

[1] M. A. Crump. *J. Roy. Society Arts*, 1911, 59, 566.
[2] Pan. Am. Union, Mar., 1911.

The asphalt exported from Cuba in 1909 was 10,796 tons, valued at 47,586 pesos. In 1910, the exports were 2,105 tons, valued at 14,497 pesos.[1]

In the fiscal year ending June 30, 1911, Cuba shipped into the United States, 14,027 long tons of asphalt and bitumen, valued at $73,426, or $5.25 per ton. The duty on this material, entered under reciprocity treaties, is about 22 per cent. advalorem.

Turkey.—It is estimated that 12,000 metric tons were produced in Turkey in 1911.

The properties and distinctions between the various minerals such as uintahite or gilsonite, wurtzilite, oil asphalt, refined bitumens, are tabulated in the Mineral Resources U. S. G. S. for 1910.

Analyses of bituminous rock from Kentucky, a sharp-gritted sandstone, show that the bitumen is of a different nature to that of Ragusa, Sicily, this rock asphalt being a soft chalk or carbonate of lime and magnesia. The bitumen of the limestone asphalts in Europe is wholly soluble in spirits of turpentine and quite different in this respect from any other.[2]

Manufacture of Asphalt.—An English patent for the manufacture of asphalt from petroleum residuum is that of C. B. Forward of Urbana, Illinois. The patentee states that he has demonstrated that asphalt is, in reality, the base of all the so-called paraffin base oils such as those of Ohio and Pennsylvania and he has made a superior grade of asphalt suitable for paving purposes, roofing and insulating, or all other uses for which the natural asphalt is adapted. The product although made from any oil in the world, cannot be determined from the natural asphalt. The first stage in the process is to try out the oil at a low temperature, below the boiling point, removing the lighter oil. When the product has reached the desired consistency, it is subjected for a number of hours to a temperature of 100° F. After this trial which tells the relative quantity of volatile oil and residuum, the process is accomplished on a larger scale. If much moisture is present, a temperature of 175° F. is used, the water being drawn off from the bottom of the retort. Small amounts of moisture are also allowed to pass out by evaporation. The temperature is then raised to 625° F. and kept there for 60 hours according to the gravity of the residuum. The lower the gravity, the less time it will take. The material is tested during the process by withdrawing a small portion. When it has reached the proper point, it is then kept at a temperature of 100° F. lower for a period of time depending again on the product desired. It is claimed that the exposure of the product at the low temperature

[1] Estadistica General.
[2] *Chemical Engineer*, Dec., 1911.

has the effect of bringing out the adhesive and lasting qualities which are lacking if the attempt be made to finish the material in the closed still.

According to C. Toone,[1] the refining of crude asphalt consists in freeing it from the more volatile oils, and at the same time separating it from the solid mineral substances of which calcium carbonate is the principal constituent. The process is as follows: (1) Melting and decanting the bitumen from the gangue; (2) boiling the crude material with water, the bitumen then running out in a molten state; (3) dissolving the calcium carbonate with hydrochloric acid, leaving the bitumen unaffected; and (4) dissolving the bitumen with turpentine, the impurities then being left as a precipitated sludge.

In the same article, Mr. Toone discusses the value of asphalt as a substitute for rubber in electrical work.

Asphalt.—Artificial asphalt, such as Byerlyte (manufactured by Byerley & Sons, Cleveland, O.) is said to contain 99.8 per cent. pure bitumen, .2 per cent. organic matter. Trinidad asphalt is 57 3/4 bitumen, 42 3/4 organic matter. It is claimed the artificial will stand low temperature without cracking and not become soft or "bleed" under the hot sun.

[1] "Asphalts and Bitumens in Electrical Work," *Cassier's Mag.*, Jan., 1911.

BARYTES

This mineral is mined in the United States by a large number of small producers, but the industry is not very large. In 1911 it appeared to be prosperous, the spurt in production that took place in 1910 being maintained. In 1907 the maximum production both in value and tonnage was reached, being approximately 65,579 tons valued at $251,308. The annual production amounts to about 55,000 tons and the total consumption probably 70,000 tons on an average in recent years. The State of Missouri is the largest producer and the average value of its product is higher than in any State except California where the mineral was first mined in 1910.

The output of crude barytes in 1911 was 41,057 short tons, valued at $152,732.

Imports.—The manufactured material pays a high import duty of $5.25 per long ton. The duty on Blanc fixe, the artificial barium sulphate, is one-half cent per pound and on barium compounds, such as oxide and chloride, 25 per cent. The imports of barytes in 1911 amounted to 18,049 short tons of crude imported at $2.03 per ton and 2,810 short tons valued at $7.87 per ton of manufactures. Barium chemicals to the extent of about $370,000 and 3800 tons were also imported.

The duty recently doubled to $1.50 per ton on the crude imported material, has not affected the domestic production and the foreign product continues to be imported for grinding, principally from Germany. There are many independent mills that buy the product from the miners, but do nothing in the way of mining themselves.

The German barytes is a strong competitor of the American product, inasmuch as it is almost free from impurities such as iron, which is found in nearly all of the barytes mined in the United States. As shown by statistics Germany is the largest producer. The duty on the German ground barytes is $5.25 per ton, and during the tariff revision a strong effort was made on the part of the American producers to increase this duty to $10.50 per ton. The German barytes does not require bleaching with sulphuric acid as does the American product. The best barytes, however, is that which is water floated, there being but two mills in the United States using the water-flotation process.

STATISTICS OF BARYTES IN THE UNITED STATES.

Year.	Production.			Imports.				Consumption.	
	Short Tons	Value.		Crude.		Manufactured.			
		Per Ton.	Total.	Sh. Tons.	Value.	Sh. Tons.	Value.	Sh. Tons.	Value.
1898....	28,247	$4.00	$112,988	1,022	$ 2,678	687	$ 8,678	29,956	$124,344
1899....	32,636	4.20	137,071	1,739	5,488	2,111	22,919	36,486	165,478
1900....	41,466	3.90	161,717	2,568	8,301	2,454	24,160	46,488	194,178
1901....	49,070	3.22	157,844	3,150	12,380	2,454	27,062	54,674	197,286
1902....	58,149	3.21	186,713	3,929	14,322	3,908	37,389	65,986	238,424
1903....	(a)50,397	3.02	152,150	6,344	22,777	5,716	48,726	62,457	223,653
1904....	(a)65,727	2.66	174,958	6,689	27,463	5,920	48,658	78,336	251,070
1905....	53,252	3.68	196,041	7,879	36,796	4,827	39,803	65,418	272,649
1906....	63,486	3.98	252,719	9,189	27,584	4,808	37,296	77,483	317,599
1907....	65,579	3.83	251,308	18,344	77,683	10,006	96,542	93,929	425,533
1908....	38,546	3.38	130,409	12,197	58,822	3,037	29,168	53,780	218,399
1909....	39,831	3.54	138,634	12,422	35,387	2,240	19,320	54,493	193,338
1910....	41,720	3.49	145,187	18,992	48,457	3,184	29,782	63,896	223,426
1911....	41,057	3.72	152,732	18,049	36,643	2,810	22,083	61,916	211,458

(a) Statistics of the U. S. Geological Survey.

PRINCIPAL COUNTRIES IN WHICH BARYTES IS PRODUCED.
(Metric tons.)

	1907	1908	1909	1910	1911
Austria-Hungary (exports)					
Barium chloride	5,220	2,974	2,585	4,344	3,741
Barytes	3,111	2,987	3,737	13	12
Belgium (production)	23,000	25,070	31,400	4,960	.....
Canada (production)	1,829	3,911	*Nil.*	*Nil.*	50
France (production)	11,150	16,277	14,111	11,632	.....
Baden (production)	9,303	8,554	15,186	13,832	15,014
Bavaria (production)	21,500	17,195	17,920	24,711	.....
Germany (exports)					
Barytes	111,209	91,111	90,555	114,264	128,452
Barium white	8,454	5,190	4,888	5,528	6,834
Barium chloride	4,189	3,389	5,340	6,403	6,180
Barytes (imports)	12,588	19,969	14,560	5,782	7,926
Italy (production)					
Barytes, ground	1,720	1,895	1,950	.....	.....
Barytes, crude	13,800	15,732	16,240	14,420	.....
Barytes (imports)	1,540	1,523	2,094	1,777	2,024
Barytes (exports)	152	724	125	150	353
Spain (production)	314	334	422	476	.....
Sweden (imports)	610	514	529	625	607
United Kingdom (production)	42,648	39,572	42,436	43,637	.....
United States (imports)	28,804	12,390	13,301	20,203	18,930
United States (production)	59,480	34,961	36,127	37,846	37,320

Markets.—The price of domestic crude barytes has been as before between $3 and $4 f. o. b. cars at the shipping points. The mines themselves are too far from manufacturing centers near sea ports, and it is therefore a question of railroad transport versus price of foreign freight and duty. The price of American ground barytes, f. o. b. New York was from $12 to $15, the average being $12.25 for 1911, a decrease from $13.50 in 1910 and $14.75 in 1909. The floated material was lower in the United States than the foreign, the latter being in demand as of superior quality.

Barytes in the United States

California.—The production is chiefly from Mariposa and San Bernadino Counties, though a new property was reported last year from Merced. The latter property was to be benefited by the Merced Valley Railroad so that products could be delivered directly in San Francisco. The value of 309 tons produced in California in 1911 was lower than before, not exceeding $3.50 for crude material. The output was not one-half of the preceding year and, owing to low prices of the manufactured product, there appears to be no inducement to increase production.

Georgia.—The Georgia production was probably less than in 1910. The important deposits are at Cartersville, Barton County, and at Elton, Murray County. As stated in Mineral Industry, Vol. XIX, an analysis of the Elton barite is 98.82 per cent. $BaSO_4$; 0.2 per cent. Al_2O_3; 0.33 per cent. Fe_2O_3; SiO_2 0.27. The latter deposit, though the analysis is better than that from Cartersville, was not worked recently.

Idaho.—The poor year in Blaine County, where a barytes prospect exists near Hailey, has prevented any work in this field. The barytes, though on a railroad, cannot be shipped to a distant market on account of freight rates.

Kentucky.—The mining is done by one company, and little was produced. The production comes from Garrard, Boyle and Fayette Counties, and in 1910 amounted to 3084 tons as stated by the Geological Survey. Some of this is ground locally, but competition was keen as the foreign products can compete with both ground and crude, although the latter can be delivered in New York for a cost of $7.50.

The Geological Survey is publishing a report by F. Julius Fohs on the Central field. This field is proven to be the greatest barytes district in the United States on account of the large available tonnage. There are nearly 150 veins, some of these being from three to five miles long, the average length being one mile. Considering the average depth of 250 ft., the district is very important. Some mining was done years ago in one vein, Henry and Owen Counties and in Franklin County, but it was solely for the purpose of obtaining lead, which was hardly profitable. When mining for barytes, lead and zinc being by-products, the whole operation, as carried on today is a paying one. Two plants of this description are in operation and there is prospect of others. These are supposed to be the pioneers of what will eventually prove to be a great barytes industry.

Missouri.—Washington County continues to be the largest producer of barytes in the United States. The production of this region was again over 75 per cent. of that of all the States. The deposits are

shallow and many are worked by open pits. There are between 30 and 40 localities tributary to Mineral Point. In some, galena occurs with the barite from which it is separated by hand cobbing. The "barite diggings" are all in a region underlain by the Potosi formation.

In the localities west of Mineral Point which are typical, the surface consists of 6 in. to 6 ft. of red clay, a residual material beneath being fragments of quartz, chert and barite all embedded in the clay. The barite is associated with drusy quartz in bunches. The masses of barite may have red iron oxide in cleavage cracks. Some galena is in the barite, which is often coated with lead carbonate. Some dolomite is associated as fragments. The barite is mined by sinking circular shafts 3 1/2 to 4 ft. in diameter into the barite horizon from which the material is removed with pick and shovel for 3 to 5 ft. around. The shafts as a rule are only 15 ft. in depth. No blasting is required. A section of what is known as Kelsey's claim is given as follows: 5 ft. soil with chert fragments at the base; 2 ft. barite and finely crystalline drusy quartz embedded in red clay; 3 ft. red clay, practically free from fragments of rock; 4 ft. barite, drusy quartz and chert embedded in red clay. The barite occurs in large masses.

The Potosi Mines Co. in Washington Co., Missouri, mines considerable "tiff" or barite from shallow lead workings. About 25,000 acres are owned and the mineral is gophered on a royalty basis by tiff diggers.

Virginia.—As in other States, most of the barite mining in Virginia is of the shallow sort. The deposits found in the Piedmont region are associated either with crystalline limestone of the Cambrian age or with Triassic red shale sandstone. The deposits of the mountain region are usually associated with the Shenandoah limestone originating generally from its residual decay. The Triassic area of Prince William County, is mined about four miles south of east of Catlett where barytes is associated with red shale and impure limestone, usually filling fractures of variable width in the red shale and in thin tabular masses in the limestone. In Campbell, Bedford, Louisa, and Pittsylvania Counties, which are all in the crystalline area, the two centers are the Evington in Campbell County and at Toshes and Sandy Level in Pittsylvania County. The latter have been worked more extensively than any other parts of the belt. Some of these bodies or pockets measure 100 to 200 ft. or more. The black clay mass in which the ore occurs, extends above and below the original limestone. There were two active producers of barytes in Virginia in 1910 and the production of the refined product was reported by one mill. The deepest shaft is at the Hewitt Mine in Campbell County. The statistics show that the production has fallen

off in recent years. The latest available figures are for 1906, 11,775 tons; 1907, 9252 tons; 1908, 3866 tons.

Manufacturers of Barium Products.—In the following table are given the names and addresses of the principal consumers and producers of barytes in the United States.

Name.	Address.	Operation.
Nulsen, Klein & Krausse Mfg. Co.	St. Louis, Mo.	Bleach, grind, float.
Finck Mining and Milling Co.	St. Louis, Mo.	Bleach, grind.
Point Milling and Manufacturing Co.	Mineral Pt., Mo.	Bleach, grind, float.
Pittsburg Baryta and Milling Corp.	Richlands, Va.	Bleach, grind.
Commercial Mining and Milling Co.	Knoxville, Tenn.	Bleach, grind.
William D. Gilman Co.	Sweetwater, Tenn.	Crush, jig, roast, salts.
Carolina Barytes Co.	Stackhouse, N. C.	Bleach, grind.
Hot Springs Mfg. Co.	Hot Springs, N. C.	Bleach, grind.
Dix River Barytes Co.	Danville, Ky.	Bleach grind.
Hammill & Gillespie	Stamford, Conn.	Bleach, grind.
Cawley, Clarke & Co.	Newark, N. J.	Lithopone.
N. Z. Graves & Co.	Philadelphia, Pa.	Lithopone.
Excelsior Mfg. Co.	Newport, Del.	Lithopone.
Grasselli Chemical Co.	Grasselli, N. J.	Lithopone.
New Jersey Zinc Co.	Palmerton, Penn.	Lithopone.
American Paint and Pigment Co.	East Alton, Ill.	Roast, salts.
Krebs Pigment and Chemical Co.	Newport, Del.	Lithopone.
Kentucky Barytes Co.	Nicholasville, Ky.	Grind and bleach.
United States Barytes Co.	Tiff, Mo.	Crude ore.
Potosi Lead, Baryta and Mer. Co.	Potosi, Mo.	Crude ore.
Leona Chemical Company	Oakland, Calif.	Crude ore.
W. R. Satterfield	Cartersville, Ga.	Crude ore.
Barium Products Co.	Scranton, Pa.	Lithopone, etc.

Foreign Countries

Canada.—Canada has not produced any barytes since 1908. Deposits occur two miles north of Five Islands, Nova Scotia. The principal outcrops are on the steep banks and bars of the East River, and at isolated points between for a distance of two miles. The barite is coarsely crystalline and remarkably pure. It occurs in various forms and as a filling for fissures, sometimes 2 ft. thick in massive rock. Near the eastern end of a fault zone north of Londonderry, barite occurs associated with much ankerite and iron ore; the barite is believed to have been formed by leaching and concentration as the adjoining ledge contains about .2 per cent. BaO.

At Scotsville, East Lake, Ainslie, Inverness County, the Barites, Limited, is operating on a 9 1/2-ft. vein of barytes. The mine is well developed and timbered and yields an excellent grade of sulphate of barium, an analysis of crude ore giving 98.25 per cent.

A plant for grinding and washing is being installed. The capacity of the plant is said to be one ton of finished product per hour.

Germany.—United States Consul General Robert P. Skinner makes the following statements: The principal deposits of barytes are found in the form of crystals near Freiberg and Marienberg, Clausthal, in the Harz

Mountains; Pribram and Svarov, in Bohemia; Felsobanya, Hungary; Kapnik, Transylvania; Courtade, Auvergne, France; and Dufton, near Bristol, England. The deposits in Germany of most importance are in the Cassel district, in Meggen on the Lenne, in Dillkreis on the Lahn, and in the Black Forest. The German deposits are mostly of the white or colorless variety but the colored variety also occurs. One of these latter deposits is at Meggen. Shipments of barytes in 1910 amounted to 114,264 tons; in 1909, 90,555 tons and in 1911, 128,452 tons. This includes some of the natural sulphate of strontium. From the reports of 1910, the largest consumers of this product were the United Kingdom, Austria-Hungary, the United States and the Netherlands. In addition to the crude barytes, manufactured lithopone exported amounted to 9576 metric tons in 1910, the largest consumers of which are the United Kingdom and France. In addition, a similar tonnage of manufactured compounds was exported, namely 11,000 tons in 1910, and 12,000 tons in 1911. Barytes compounds have been decreasing.

Lithopone is manufactured in some 12 factories, of which three are in Silesia, two in the Duchy of Brunswick, one in Thuringia, two in Nassau, three on the lower Rhine, and one in Bavaria. The capacity of these factories per annum varies between 1000 carloads at the most important and 100 at the smallest. Until 18 months ago the producers of 95 per cent. of the total product had a price agreement, but at present there is sharp competition between the manufacturers, the advantage of which is enjoyed by the consumer.

The enlargement of factories has continued steadily, and the present unfavorable condition of the industry is attributed to this, to the existence of works outside the former trust agreement, and, finally, to tariff changes in foreign countries which made it impossible for German lithopone to enter those markets.

In addition to its various uses in the paint trade, lithopone is also being employed in the manufacture of many rubber articles, linoleum, oilcloth, sealing wax, and artificial stone. The consumption of lithopone will probably increase steadily, although there is, no doubt, an overproduction at this time. Several existing factories are not being operated fully, but high expectations are entertained for the future, and nearly every factory is in position to increase its output of the standard brands by 100 per cent.

In Prussia, eleven mines produced 28,675 metric tons of barite in 1910, and 29,848 metric tons in 1909.

There were 840 tons of strotinaite produced at the Anton mine, Ottmarsbocholt, Germany, in 1910. Russia is the chief source of imported barite.

Italy.—No witherite or barium carbonate occurs in Italy; barite, however, is abundant. Albenga is an important locality for the pure white ore; large deposits also occur in the valley of the Valsassina and also in Sardinia, at both of which localities in recent years the ore is being mined. The most important mines belong to a firm in Lecco. Smaller localities are Pietiasauta where the ore is used on the spot for manufactures.

Much barium salts are imported into Italy, in spite of a duty. The cost of production is quite high thus allowing the lower priced foreign material to compete.

Barium chloride is manufactured by means of locally produced hydrochloric acid, by acting on the sulphide, but the process loses the sulphuretted hydrogen generated. The result is a heavy cost for barium nitrate, which results from a double decomposition on treatment with sodium nitrate, but this salt is mostly imported from Germany.

Barium superoxide though extensively used in the straw hat industry near Florence is also imported from England. The local industry has not yet developed. Barium hydrate is used in clarifying molasses and recently the manufacture according to a German process in electrical furnaces, has been undertaken by a large sugar factory; the barium carbonate is regenerated in this process. A calcination process formerly used was given up because the conversion of carbonate into hydrate was only 55 to 60 per cent. Hence the hydrate is also imported mostly. The duty is to be reduced on this substance.

Blanc fixe is also imported, chiefly as the local manufacture is not considered of best quality by the consumers, but also as lead can be used without restriction by Government rules. With cheap calcium chloride, possibly obtained from a phosphate industry of which there is none in Italy, cheap barium chloride would be possible, and better than the existing native product.

United Kingdom (W. W. Barnes).[1]—Sulphate of barytes is being opened up near Treby. One lode vein has been traced for some 2 1/4 miles, and is most interesting, geologically; a portion of the lode is in the Altered Ash rock, part "contact" with the altered ash and basalt, and part "contact" with the basalt and carboniferous limestone. The features geologically are entirely singular within the United Kingdom, and the deposit is one of the largest known, what is called the No. 1 vein being so extensive as to admit of castening, showing 18 per cent. of absolutely solid baryta between the cheeks or walls, and the color is exceptionally good. The geological conditions are most favorable, not only toward large deposits of the sulphate (which are assured—many thousands of

[1] *Chem. Trade Journal*, 1911, p. 129.

tons being "blocked out" and actually in sight), but also when the vein becomes "contact" between the carboniferous limestone and the basalt, carbonate of baryta will very probably be got. Under such geological conditions prolific mineral depositions are rarely found wanting; and to use the working miner's expression, "Barytes always rides a good horse," or in other words rarely fails to be associated with more valuable ores—blende, copper, wolfram, etc. The analysis of the barytes is said to be up to 99.65 per cent. $BaSO_4$, lime and magnesia "traces," silica, 0.17, iron, 0.05. The owners of the property are about erecting mills to grind the barytes; they also intend to manufacture lithophone (which is rapidly taking the place of lead pigments) and also the several salts of barium.

Strontium sulphate produced in the United Kingdom amounted to 14,042 tons valued at £10,532 in 1909, and 4761 tons valued at £3570 in 1910.

The bulk of the barium compounds comes from Durham, Northumberland, Shropshire in England and Cork, Ireland. Northumberland also supplies carbonate or witherite, as does also one firm in Cumberland. The total production for 1910 was 44,667 long tons and from mines in 1911 42,550 tons. The value in 1910 was £43,909. The witherite is worth $10 per ton and the barite $2.50 per ton in most places. In Ireland the value is given as $4 per ton for barite. One-quarter of the production given is witherite.

Uses.—Recent tests made by the Paint Manufacturers' Association of the United States show that a combination of one-third each barytes, zinc white and white lead has given good results as a pigment in paint. Barytes alone as a white pigment is not a success, though the common notion that it causes chalking in conjunction with white lead, is not correct. As much as 60 per cent. barytes and 40 per cent. zinc has been found satisfactory. The use of barytes as an adulterant can hardly deceive anyone. There is no objection whatever to the use of barytes if it is given as an ingredient and sold as such. The mixture called Venice white lead contains 50 per cent. barytes; what is known as Hamburg white contains 66 per cent. and Dutch white contains 75 per cent.

By grinding the crude ore, treating with acid to remove the iron, and finally washing, floating and drying, there is produced the commercial form of this valuable pigment. It is used in large quantity as a base upon which to precipitate colors, and also together with other white pigments in the manufacture of ready mixed paints. It renders the paint coating more resistant to abrasions, and gives to the paint certain very important brushing qualities. It is a very stable pigment, not being materially affected by either acid or alkali, and can be used with the

most delicate colors. In oil it is transparent and must be mixed with opaque pigments when used in ready-mixed paints. It is generally used with lighter pigments, such as asbestine, in order to prevent settling. Under the microscope both by polarized and transmitted light, the sharp angles of the particles appear distinctly, with no tendency to mass into compact form. Although transparent in oil, it is valuable in moderate percentage in a ready-mixed paint.

Barium Sulphate (Blanc Fixe).—Blanc fixe is the precipitated form of barium sulphate, resulting from the action of soluble barium salts on soluble sulphates. The specific gravity (4.2) of this compound is lower than that of barytes. Possessing greater opacity in oil, it is of more value as a paint pigment for some purposes. It comes in for its greatest use as a base on which to precipitate lake colors. The very fine particles show a slight tendency to agglomerate.[1]

The milling of barytes has been described in detail in Vols. VIII, X, XIII, XIV, XV, XVI and XIX of the MINERAL INDUSTRY. The impurities on account of which the milling is required, are hematite, clay, chert and galena. To some extent these can be avoided in the mining, but the product generally mined in the United States requires treatment. The process generally consists of crushing, grinding, washing and bleaching. Where clay is the principal impurity, the usual mill consists of crusher, log-washer, bleaching tanks, followed by a second log-washer, drier and fine grinding mills of the arrastra type. Tube mills are also used. Another type of mill is the one using jigs in place of log-washers, especially where galena is required to be separated.

The calcining of barium sulphate is described in the MINERAL INDUSTRY, Vol. XIX, p. 71.[2] For this process, material that is impure with iron oxide can be used, but the separation and hand-picking of impurities and also careful mining are advisable. A 30-ton plant for reducing the mineral to barium sulphide consists of a cage, disintegrator for powdering coal, two tube mills to grind the coal and barytes together, and one rotary cement kiln 7 1/2 ft. in diameter and 80 ft. long, with the usual hoppers, elevators and conveyors. The pulp from the tube mill is discharged in a wet condition of a consistency like thick cream, into the furnace. About 30 per cent. of the weight of the crude mineral is required as fuel for heating the furnace besides 30 per cent. of pulverized coal mixed with the mineral fed into the kiln. The product of the kiln averages 60 per cent. water-soluble and 20 per cent. insoluble. High silica and high iron tend to decrease the water soluble BaS and to increase the acid soluble $BaSO_3$, $BaCO_3$ and $BaSiO_3$. The insoluble

[1] From Gardner, "Paint Technology and Tests," McGraw-Hill Book Co.
[2] From S. Frazier, *Chem. Engr.*, Feb., 1911.

residue requires frequent analyses in order to insure a high efficiency. The barium sulphide solution is filtered and stored in tanks large in capacity, and is used as the base for preparing various compounds such as lithopone. The residue is treated in agitators with acid, thus obtaining the barium chloride. Details of work in one of these mills is described by A. H. Fay in Vol. XIX of the MINERAL INDUSTRY, p. 73. The causes and methods of obtaining a high efficiency are all pointed out in this article.

LITHOPONE

The lithopone tests in paints exposed for one year in Pittsburg were discouraging, but showed that the breaking down occurred in various ways, and hence in any case, the material must be prepared to suit the conditions under which it is used. As to use in paint, at the time of these tests in 1909 many European journals had recommended the use of paints containing much lithopone for exterior surfaces. The mixture proved unfavorable here excepting in those cases where zinc oxide and whiting (chalk) were used, these apparently preventing decomposition of the paints.

Lithopone may not be durable outside paint, that is, it may change its color or absorb water from the atmosphere or rain but its quality for interior flat wall paint has extended its use in recent years. It "flats" far better than zinc or lead paints, and is more durable than cold water paint and moderate in cost, 3 1/2 cents per pound. It has become indispensable with oilcloth and shade cloth makers, in rubber making and for floor paints, ship paints and cheap enamel. In this form barite has found a place in paint pigment where it fulfills requirements.

The Manufacture of Lithopone.[1]—Lithopone is, in its pure form, a product of the double decomposition of barium sulphide and zinc sulphate, consisting usually of 30.5 per cent. of zinc sulphide, 1.5 per cent. of zinc oxide and 68 per cent. of barium sulphate. It is often adulterated with china clay, gypsum or whiting (barium carbonate). Such is the Durabo White, which contains 24.5 per cent. zinc sulphide, 51 per cent. barium sulphate, 18 per cent. white clay, 5.5 per cent. infusorial earth.

A lithopone for the linoleum trade or for the roller-shade manufacturers will vary widely from one intended as an interior wall paint.

Requirements of Manufacture.—The materials used in a successful lithopone plant comprise barytes (crude), coal, spelter, oil of vitriol, common salt, sal ammoniac, sodium phosphate, chlorate of potash, calcium chloride and caustic soda. Taking the average figures for

[1] Abstract of an article in the *Oil, Paint and Drug Reporter*, Nov. 20, 1911.

several years, the percentages by weight of the consumption of these materials are as follows: Crude barytes, 37.30 per cent.; bituminous coal, 26.59; spelter or scrap zinc, 9.63; oil of vitriol, 22.05; rock salt, 3.39; sal ammoniac, 0.44; sodium phosphate, 0.14; chlorate of potash, 0.31; calcium chloride, 0.13; caustic soda, 0.2 per cent. The average yield of lithopone from this material was 41 per cent. of a normal grade, averaging 29.85 per cent. zinc sulphide, 1.63 per cent. zinc oxide and 68.52 per cent. barium sulphate.

Without going into minute details a plant with a capacity of 10 tons every 24 hours, a building at a cost of about $25,000, fitted out with apparatus costing as much more, is required. The apparatus includes engine, two reverberatory furnaces, crusher for breaking the crude barytes, at least four large burrstone mills, disintegrator, elevator, etc., steam-heated drying rooms, a series of muffle furnaces, lined tanks for the zinc-sulphate and barium-sulphide solutions, 30 vats of about 1500 gallons capacity for washing the product, filter presses, trays for handling the precipitate, aside from numerous other tools and appliances. Where perfectly clear water cannot be had, large tankage for filtering the water for levigating the product is absolutely needed.

Manufacture of the Barium Sulphide.—The process of manufacture itself begins with the preparation of barium sulphide, usually termed black ash, which consists of breaking up the crude barytes in a crusher and running it through a burrstone mill to a certain size, about like a small pea, steaming it to avoid the dust which is liable to clog the furnace, adding to every 100 lb. of this broken rock about 20 lb. of bituminous coal of small pea size, and about 10 lb. of common salt to assist fusion. This mixture is placed in a reverberatory furnace, where it is heated to dull redness without admission of air. When the reaction has taken place the mass is placed in vats, digested and filtered. When the crude rock contains, as is often the case, much clay, iron, etc., the pure barium sulphide must be leached out and the foreign residue removed. While this black ash and its solution is being prepared, spelter, zinc dross or scrap zinc or any available zinc salt is being dissolved in separate vats or tanks in sulphuric acid, forming a solution of zinc sulphate. The barium-sulphide solution and the zinc-sulphate solution should be of a concentration of 60 per cent. and at a temperature of 140 to 150° F., when they are poured together, and the zinc solution should be poured at double the rate of the barium solution, in which case a precipitate is obtained that can be filtered and dried.

Purification of the Zinc Solution.—It is evident that when zinc dross is used and in any case unless refined spelter is employed, the zinc solution must be treated with a compound that will eliminate all traces of

iron and other foreign materials. For this purpose chlorate of potash is added. The precipitate, after being filtered and dried, is placed in muffle furnaces and heated at as high a temperature as 900°, when it is suddenly plunged into cold water. In calcining in the muffle furnace, sal ammoniac to about 1 per cent. of the dry weight of the pigment is added to assist fusion. When the material is thus prepared it is carried to the burrstone mills by means of centrifugal pumps and pipes, or by other mechanical appliance and there ground in the pulp state to the standard degree of fineness, and then conveyed to the washing tanks, where it is washed over as often as it is necessary to eliminate all traces of impurities, free sulphur, iron, etc. This often requires as many as 12 to 15 washings and sodium phosphate and caustic soda are added in small portions to aid in the process. Calcium chloride is also added as a bleaching agent when the whiteness is defective. A minute quantity of ultramarine blue is also one of the ingredients to assist in eliminating too creamy a cast. When the test shows the pigment to be thoroughly washed, it is filter pressed, the cakes so formed are placed on wooden trays, the trays stacked up on suitable trucks and these put into drying rooms usually built of metal and heated by exhaust steam, which is aided by jets of live steam. When the cakes of lithopone have dried, they are fed into the hopper of a disintegrator, which pulverizes the lumps into a powder of uniform fineness, and when this powder has been put up in barrels or casks of suitable size, the product is ready for the market.

Low Carrying Charges in Factory.—The one great feature about the manufacture of lithopone, as compared with that of white lead, is in its quick manufacture, for a working batch of lithopone can be turned out in three or four days, whereas even quick-process white lead will require at least one month from the time the pig lead is unloaded, with one single exception, that of the so-called mild process of white lead. It stands to reason that the capital tied up in a lithopone plant is much smaller than in the case of white-lead manufacture. Another advantage of the process as against that of white-lead making is that the health of the workmen is not as much in danger, though the work is by no means pleasant, on account of the vapors and the heat. The cost of manufacture, exclusive of the capital invested, but including repairs to apparatus, superintendence and labor, should not exceed $10 per ton (2000 lb.), while the cost of material, packages, etc., necessarily varies with the condition of the market.

Except what is included in barite, there was no mining of strontium minerals in the United States. Strontia was imported to the extent of $23 in 1910, falling off from $270 in 1909; in 1911 the value was $44.

Metallic Strontium.—The following method of preparing strontium

was described in the *Journal of the American Chemical Society.* Pure strontium chloride was fused in a hemispherical iron cathode vessel 25 cm. in diameter, with walls 0.6 cm. thick, with a carbon anode 8 by 8 cm. This allowed a low anode current density and avoided overheating. With a current of 125 amperes and 40 volts for seven hours, 76 grm. of metal were removed in small lumps up to 3 grm. in weight. The metal is removed from the mixture by crushing on an iron plate and then sifting away the chloride. Analysis showed it to contain about 98 per cent. of metallic strontium, which is a light metal, with a silvery luster when cut, gradually becoming yellow and finally white and non-lustrous. It is softer than calcium, and can be cut with a knife. It alloys with iron. Hydrogen and nitrogen unite with the heated metal. The specific gravity is 2.55 and the specific heat 0.0742, corresponding to an atomic heat of 6.5. The iron alloy is rather hard, and decomposes water. This alloy contained 23 per cent. of strontium.

Preparation of Crystallized Strontium.[1]—A mixture of finely powdered strontium oxide and finely granulated aluminium, in proportions corresponding to the equation, $3SrO + Al_2 = 3Sr + Al_2O_3$, is placed in the lower part of a soft steel tube, one end of which is closed. The temperature of the tube is raised progressively to 1000° C., an outer porcelain tube, in which the steel tube is contained, being exhausted of gas at the same time; toward 600° C., hydrogen is evolved in abundance, being derived from the strontium hydroxide which always accompanies strontium oxide. The temperature is kept at 1000° C. for four hours, and the apparatus is allowed to cool. The cold interior tube is covered with a silvery white deposit of crystalline strontium of 99.41 per cent. purity. It is found that 30 grm. of strontia furnish about 19 grm. of the metal. The specific gravity of the metal prepared in this way is 2.63 referred to water at 0° C.

[1] A. Gunts and Galliot. *Comptes rend.*, 1910, 151, 813.

BAUXITE

The only ore of aluminum is bauxite containing up to 70 per cent. alumina. Corundum, which is 100 per cent. alumina, is too valuable for other uses and does not occur in sufficient quantity, while clay, which is abundant, contains much combined silica and no metallurgical process is yet commercially available to utilize it as an ore.

Bauxite is a source of aluminum salts. Another natural source of aluminum salts is alunite, deposits of which have recently attracted attention on account of their potash contents, but the alumina as a by-product will be important should this industry develop.

Bauxite was discovered in America in 1887, a few miles north of Rome, Ga., in Floyd County. Ore shipments were made from that point in 1889. In 1891 shipments began from Alabama and about that time large deposits were found in Arkansas.

The production of bauxite in the United States was about the same as in 1910, when the production increased rapidly from a low mark in 1908. The output of quarries in this country amounted to 155,618 long tons in 1911. The bulk of the production comes from Arkansas, and next in order from Georgia, Alabama and Tennessee.

The following table shows the production of bauxite in the United States from 1900 to 1910, inclusive.

PRODUCTION OF BAUXITE IN THE UNITED STATES.

(In tons of 2240 lb.)

State.	1901 (a)	1902	1903 (a)	1904	1905	1906	1907 (a)	1908 (a)	1909 (a)	1910	1911
Alabama....	18,038	5,577	22,374	7,087	17,094	27,131	97,776 (b)	14,464	22,227	37,229	30,170
Georgia.....		19,000		16,909				(b)37,703	(b)106,874	(b)112,450	125,448
Arkansas....	867	4,645	25,713	24,016	30,897	51,200					
Total....	18,905	29,222	48,087	48,012	47,991	78,331	97,776	52,167	129,101	149,679	155,618

(a) Statistics of the United States Geological Survey. (b) Production of Tennessee included.

The production is of the crude material, but it must be remembered that some of this is treated by washing and only a smaller bulk is shipped. In addition, stocks of crude material as mined are included in the production, but they are included in the consumption also in the table given.

Industrial Conditions and Prices.—The average price of the mineral advanced from $5.27 per ton in 1909 to $5.33 per ton in 1910, showing

that demand kept pace with the production. There were no changes worth noting in 1911. The value of crude material is as low as $4 per ton and when washed and dried as high as $8. The average value was $4.87. Imports of the mineral, which were entirely from France, increased substantially. The quantity imported in 1910 was 15,720 tons, valued at $65,743, as compared with 18,689 tons, valued at $83,956 in 1909. In 1911 43,222 tons, worth $164,300 were imported. A small quantity, about 900 tons, of refined bauxite or alumina, is included. Imported material pays a duty of $1 per ton and the ocean freight rate is approximately $2 per ton.

The accompanying table gives the production, imports, exports, and consumption of bauxite in the United States for a period of years.

The producers of bauxite in the United States are the Republic Mining and Manufacturing Company, 1111 Harrison Building, Philadelphia, Penn., the Aluminum Company of America, Pittsburg, Penn., the National Bauxite Company, Rome, Ga., John H. Hawkins, Rome, Ga., the Howard Hydraulic Cement Company, Chattanooga, Tenn., and the Cherokee Mining Company, also of Chattanooga. The first two mentioned companies are by far the largest producers, operating mines in Alabama, Arkansas, and Georgia. The National Bauxite Company operates both in Georgia and Tennessee, while the last three of the above-mentioned companies operate in Georgia only.

CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.			Imports.		Exports.		Consumption.	
	Long Tons.	Value.	Per Ton.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1897	20,590	$51,475	$2.50	2,645	$10,515	2,537	$5,074	20,708	$56,916
1898	26,791	66,978	2.50	1,201	4,238	1,000	2,000	26,992	69,216
1899	36,813	101,235	2.75	6,666	23,768	2,030	4,567	41,449	12,436
1900	23,445	85,922	3.66	8,656	32,968	1,000	3,000	31,101	115,880
1901	(a)18,905	97,914	4.25	18,313	66,107	1,000	3,000	36,218	144,021
1902	29,222	128,206	4.39	15,790	54,410	*Nil.*	.....	43,112	175,875
1903	(a)48,087	171,306	3.56	14,889	49,684	*Nil.*	.....	62,976	220,990
1904	48,012	166,121	3.46	15,475	49,577	*Nil.*	.....	63,487	215,698
1905	47,991	203,960	4.25	11,726	46,517	*Nil.*	.....	59,717	250,477
1906	78,331	352,490	(e)4.50	17,809	63,221	*Nil.*	.....	96,140	415,711
1907	(a)97,776	480,330	4.91	25,065	93,208	*Nil.*	.....	122,841	573,538
1908	(a)52,167	263,968	5.05	21,679	87,823	*Nil.*	.....	73,846	351,791
1909	(a)129,101	679,447	5.27	18,689	83,956	*Nil.*	.....	147,790	763,403
1910	149,679	778,203	5.33	15,720	65,743	*Nil.*	.....	165,399	843,946
1911	155,618	750,649	4.87	43,222	164,301	(f)	.....	198,840	914,950

(a) Statistics of the United States Geological Survey. (e) Estimated. (f) 17,000 tons concentrated.

Georgia.—The discovery of bauxite was made by Otto Veatch, State Geologist, in the latter part of 1906 and the work demonstrated that the bauxite was widely distributed throughout the central and northern parts of Wilkinson County. The bauxite is associated with white cretaceous clays and tertiary clays, in all cases near the contact of these

clays, with overlying tertiary sands. The bauxite appears to be in tr beds of considerable extent. The first shipment from the Central Geo gia field was made during the summer of 1910 by the National Bauxi Company and production was continued during 1911. Some of the beds are as thick as 10 ft. The nodular variety is generally quite ha and even flinty but some of it is more clay-like and softer. It is suppos that the similarity of the so-called chalk beds in Twiggs, Washingto Glascock, McDuffy and Richmond Counties, may lead to the discover of commercial deposits.

The following analyses was recently made by Edgar Everhart, the Geological Survey of Georgia.

ANALYSES OF WILKINSON COUNTY BAUXITE.

Al_2O_3.	Fe_2O_3.	SiO_2.	TiO_2.	H_2O.	Moisture.
52.92	7.66	10.17	2.30	26.55	0.35
57.58	0.96	9.38	2.76	29.12	0.35
41.97	18.24	17.50	2.65	16.83	2.65
57.36	1.94	0.90	3.65	32.44	0.67
42.73	14.99	15.57	1.84	20.29	2.79
61.77	1.67	0.90	2.26	31.77	0.95

While the mining has not made much progress in this region, the o development is promising. As usual in the development of new field some litigation has occurred and the owners have held their propert at very high prices. The deposits in the northwestern part of Georgi the oldest bauxite region in America, are near the margin of the Coast plain and are not similar to those of central Georgia here described.

Arkansas.—Bauxite on the Chicago, Rock Island & Pacific Railway five miles east of Benton, the county seat of Saline County, is th central shipping point in this State. The mines are located on th Bauxite & Northern Railway, which is a spur built from the St. Louis Iron Mountain & Southern Railway system at Bauxite Junction. Dur ing 1911 five mines were operated by the American Bauxite Company Other prospects are being developed. In this region the stripping ma reach a thickness of 40 ft., although it usually does not exceed 10 to 1 ft. Some of the overburden is used to obtain a small quantity of or The hillside is worked in benches and in the ore itself some blasting i necessary. Usually the ore is simply shoveled into mine cars and d livered to the crushing machinery. Wet ore is delivered to the disi tegrators and the dry ore goes through a Gates crusher. The ore then dried in rotary kiln driers. A feature of this latter treatment ha been the use of natural gas fuel from the Caddo fields in Louisian The surface of the bauxite beds is irregular, but steam shovels are us in some cases to remove the overburden. The bauxite formation re

on a decomposed syenite and is up to 30 ft. in thickness. A typical section of a bauxite deposit is as follows:[1]

1. Surface sand and gravel.
2. Clay with halloysite(?). Very difficult and expensive to remove.
3. Bauxite.
4. Transitional phase between bauxite and syenite, composed chiefly of clayey material.
5. Unaltered syenite.

Three grades of ore are mined, that containing less than 2 per cent. ferric oxide being used in the manufacture of aluminum salts. An intermediate grade is used in the manufacture of metallic aluminum. For the manufacture of some classes of alundum, a grade high in silica is used by the Norton Company of Worcester, Mass. Some of the clay-like ore having a reddish appearance, has proven to be among the best ore as far as alumina contents are concerned, a fact which was at first unsuspected.

Tennessee.—As in the previous five years, the National Bauxite Company has been working their deposits near Chattanooga, Tenn. The Perry mine was the only one worked in 1911. The mine consists of a rounded pit 200 to 300 ft. in diameter and is more than 100 ft. in depth. West of the mine, the old pits formerly worked are now full of water, but considerable stripping has been done in a large area. Bauxite has also been found at a point south of the mine on the eastern side of Missionary Ridge and is reported on the Lamb place 2 1/2 miles southeast of Tyner. The block ore is a fairly compact, light to dark olive, clay material, and is low in both iron and silica. Considerable clay is present in most of the ore. In the mine a body of lignite was found underlying the block ore and it is said to be commonly associated with bauxite deposits, having been observed in Alabama. The bauxite from the Perry mine runs between 50 and 55 per cent. alumina. The ore is mined by pick and shovel and hoisted to the washers and driers. The wet ore is dried before shipping. The pebble ore is washed and dried. In the latter part of 1911 about 80 tons per day were shipped, most of which went to Philadelphia for the manufacture of aluminum salts. The association of bauxite with other minerals, such as limonite and kaolin, may lead to the discovery of these deposits when these minerals are found on the surface. Specimens of limonite suggest that the bauxite was an alteration or replacement product from the limonite, iron having been replaced by alumina.

Extended studies of the deposits in Tennessee will be necessary to determine a theory which can be used for intelligent prospecting[2].

[1] W. C. Phalen, U. S. Geol. S.
[2] G. H. Ashley, abstract, Resources of Tennessee, December, 1911.

The deposits so far known in Tennessee occur in a great body of limestones, dolomites and cherts, known as the Knox dolomite. The same condition is found to exist in Georgia and Alabama. But the Knox dolomite comprises a very uniform series of rocks from 3000 to 4500 ft. thick that are folded in various ways so as to underlie several thousand square miles in the great valley of eastern Tennessee. If it be true that the ore is confined to that series of rocks, prospecting need not be undertaken in the thousands of square miles underlain by other series of rocks.

Bauxite Mining in Foreign Countries

Canada.—The alumina imported from the United States, was 17,653 long tons valued at $353,872 in the fiscal year 1911.

France.—The production of bauxite in France during the year 1910 was 196,056 tons valued at $379,000 being an average of $1.93 per ton. This is about 40,000 less than the 1908 production, the average price of which was $1.90 per ton. The bauxite imports during the year 1910 were 413 tons while the exports were 114,315 tons.

A large part of the output is used for metallic aluminum manufacture. The independent companies competed individually until 1907, when a combination was effected. Bauxite is named after the town Baux where the mineral was first discovered.

India.—The extensive deposits of Indian bauxite remain undeveloped. The Katui quarries continue to produce small amounts. In 1910 the output was only 66 tons valued at £25.

Ireland.—Bauxite, sometimes called alum clay in the United Kingdom, occurs in the form of seams lying between sheets of tertiary rocks, in County Antrim. The production of aluminum from bauxite dates to 1896, when the British Aluminum Company, Ltd. was started. Alumina is prepared at the works near Larne, County Antrim, and shipped to Fayers in Inverness-shire, and to Kinlochleven, Argyllshire, where abundant water-power is available for electric current.

In 1908, the greatest productive year, 11,716 long tons valued at £3025 were produced. In 1909, 1910, 1911 the production was 9500, 3792 and 6007 tons respectively valued at £2408, £882 and £ 562.

Italy.—The discovery of bauxite in Italy is comparatively recent, the first published account regarding its occurrence appearing in 1901. As yet there is only one producing bauxite mine in the country. In 1909 the output of this property amounted to 3943 metric tons, valued at $7610, as compared with 7000 metric tons valued at $12,159 during the previous year. In 1911 production was 4595 tons, valued $8,870.

The following description of the bauxite deposits at Pescosolido, in the Liri valley, in the Terra di Lavoro district of Abruzzi is abstracted from a paper by B. Lotti.[1]

Bauxite has been found at various localities in the central Apennines, especially in the district of Terra di Lavoro, and is invariably interbedded with certain cryptocrystalline or compact limestones of Cretaceous age. The mineral is reddish-brown, irregularly mottled with paler spots, and of a finely granular oölitic structure, with numerous spherules of a darker brown dispersed throughout the mass. The specific gravity ranges from 3.22 to 3.95; the percentage of alumina, from 54.60 to 58.85; of iron oxide, from 18.62 to 30.63; and of silica, from 3.65 to 7.91 per cent. The Pescosolido deposit occurs on an outlying spur of Monte la Brecciosa, at an altitude ranging between 2600 and 3000 ft. above sea-level, as a stratum interbedded with the white Cretaceous limestones, with a strike from northwest to southeast and dipping 50 degrees toward the southwest. The mineral is of uniform character, and the deposit is quite conformable to the beds above it, exhibiting a gradual passage into the limestone of the hanging wall. But, at the foot-wall, the conditions are quite different: the bauxite lies upon an irregular limestone surface, and, enveloping innumerable fragments of the same rock, appears to invade the cavities in the underlying limestone, forming a calcic-bauxite breccia or conglomerate.

The author holds that the Pescosolido bauxite is of contemporaneous origin with the marine limestones that overlie it. Deposition took place practically at the surface, and is not assignable to the percolation of acidic solutions through the limestones from below. Either the aluminous and ferruginous particles were precipitated on a shallow sea-bottom that was much disturbed by wave action; or, as the author thinks more likely, acidic solutions bearing particles of alumina and iron flowed over a sea-bottom formed of calcareous material, and the erosion of this material was accompanied by an exchange of chemical bases, with consequent precipitation of alumina and oxides of iron.

There seems to be no doubt regarding the great similarity in character, approximating identity, between the Italian bauxite deposits and those of other countries, as is the case with bedded deposits of pisolitic limonite. Moreover, there are numerous instances of the passage of bauxite into such limonite, and this again presupposes a close genetic relationship between these deposits. Certain French occurrences are cited as examples; and, barely 900 yd. away from the Pescosolido bauxite deposit, a small lens of brown iron ore occurs under similar stratigraphical conditions, and a little farther away there is a larger lens. Several

[1] *Zeit. f. prakt. Geol.*, 1908, XVI, 501–504.

outcrops in the same neighborhood are enumerated which consist partly of brown iron ore and partly of bauxite.

Rhodesia.—In some localities, the iron-bearing laterites grade into bauxite. The extent of them is not known.

Uses.—The uses of bauxite are chiefly for the production of metallic aluminum and for the manufacture of alum. It is also used for producing the abrasive or refractory called alundum, which is fused alumina. Minor uses are for bauxite bricks and calcium aluminate. For the manufacture of chemicals such as alum the material must be free from iron.

N. Lecesne has patented (Fr. pat. 424,514, Dec. 31, 1910) a mechanical method for removing most of the usual impurities of bauxite. The bauxite, after being air-dried or dried by heating, is disintegrated in any form of apparatus in which it is subjected to impact (as opposed to grinding or crushing), and the dust, which contains most of the ingredients other than alumina, is eliminated, as produced or subsequently (by means of a fan, for instance), leaving a granular concentrate rich in alumina. In order to get rid of iron more completely, the bauxite may be dried by heating in a reducing atmosphere of 350 or 400° C., and the magnetic iron oxide thus produced separated electromagnetically. As an example, it is stated that bauxite containing 3 per cent. of silica and 60 of alumina may thus be made to yield a concentrate containing less than 1 per cent. of silica and as much as 86 per cent. of alumina.

Experiments are being made with bauxite brick linings for furnaces but their merits have not yet been determined. For the properties of alundum see page 32. Calcium aluminate has hydraulic properties and according to a series of patents Nos. 903,017, 903,020, dated Nov. 3, 1908, by E. W. Lazell and Henry S. Spackman, is made by grinding and sintering a mixture of bauxite with lime and clays or slags. A typical mixture is as follows: 86 parts of hydrated lime, 10 parts of calcium aluminate, 2 parts of plaster of Paris and is ground through a 100-mesh; it sets in about 4 to 15 hours, and has a tensile strength up to 150 lb. per square inch. Some of these mixtures when used as plaster can be mixed with 5 parts of sand have the required spreading qualities.

As a possible source of raw material for alumina, it could be obtained as a by-product from alunite, which is a hydrous sulphate of aluminum and potassium having the following formula: $K_2O.3Al_2O_3.4SO_3.6H_2O$. It contains 11.4 potash and 37 per cent. alumina. According to W. T. Schaller of the United States Geological Survey, ignition will volatilize three-fourths of the sulphuric acid leaving a residue, on leaching, containing insoluble alumina. It is estimated that the deposits at Marysvale, Utah, would contribute about 100,000 tons of alumina

for every 100 ft. in depth. Other deposits are in Rosita Hills, Colo., the Rico Mountains at Silverton, at Cripple Creek, Col., at Goldfield, Nev., at Tres Cerritos, Calif., in the Clifton-Morenci district, Ariz., and in Beaver County, Utah. The European deposits occur at Tolfa, Italy, and in New South Wales at Bullah Delah. It also occurs in India. The alunite is used directly in the manufacture of alum both in Italy and Australia.[1]

Most alum that is now used is concentrated alum, that is, it contains no alkalies. The principal impurities that are troublesome in the manufacture are sulphuric acid and iron, but it is manufactured in considerable purity to-day. In order to avoid the acid, the manufacturers make it somewhat basic, that is, it contains an excess of alumina.

Alum is used in paper manufacture and as a mordant in dyeing. In paper manufacture it converts part of the rosin in an alkaline solution to aluminium resinate, a substance that permeates and binds the fiber. The small trace of free rosin is also supposed to have an effect in this process. The alum, however, is found indispensible.

[1] *Bull.* 511, U. S. G. S.

BISMUTH

No bismuth ore was mined in the United States in recent years. Bismuth is, however, obtained as a by-product by the United States Metals Refining Company at their plant in Grasselli, Ind., and also at their plant at Chrome, N. J. The first plant recovers the metal in the refining of lead bullion electrolytically by the Bett's process and the second plant uses a special electrolytic process. It is not known what the production in the United States was in 1911 but it is supposed to be in excess of that of 1910 which was about 90,000 lb. It is not known definitely whether other smelters are preparing to produce bismuth but a number of them have the opportunity, as the metal occurs as an impurity in the lead bullion. No smelter in this country pays for the bismuth in the ores purchased. In this connection, it may be considered that probably 3000 lb. of metallic bismuth are wasted in the smelter smoke from the stacks of the various smelteries in the United States. Bismuth is an undesirable constituent in lead and in most cases, must be separated. If the metal could be produced more cheaply, there is no doubt that new uses would be found. The price, at present, the world over, appears to be controlled by a syndicate and is in the neighborhood of $1.50 to $1.80 per pound. During the year ended June 30, 1911, there was imported into the United States, 178,298 lb. of metallic bismuth, valued at $321,360 or $1.80 per pound. It is, of course, duty free. No large deposits of bismuth exist in the United States but some ores are more valuable for the bismuth contents than for the precious metals for which, however, they are treated, the bismuth going to waste. The United States Geological Survey reports two mines in Colorado carrying as high as 18 per cent. bismuth. In these instances, the bismuth contents would be worth several hundred dollars, which is much more than the gold and silver value. A dispute about the Colorado bismuth ore has resulted in law suits and when these are settled, there is a possibility that the permanent industry will be established.

In Utah, Beaver County, in the Beaver Mining district, a 6 ft. vein of bismuthinite ore is found. This is an oxidized ore also associated as hydrous carbonate of bismuth called bismutite. If a demand for bismutite ores develops, these deposits would prove especially valuable because they contain no arsenic.

Germany.—One-half a ton of bismuth was produced in Prussia in 1910.

Mexico.—In Sinaloa, in the Cosala district, an important deposit exists in the Bismuth King Mine. A St. Louis, Mo., company has attempted to treat the bullion from this mine for bismuth and some progress has been made by the successors of this company in treating the ore in a special furnace. In Sonora, near Arizpe, there is said to be a vein bearing bismuth, copper, silver and gold and having a thickness of 20 ft. Assays have yielded $8,000 per ton presumably from picked samples. The Belen mine in Sonora and also the Mariposa mine in Sinaloa are responsible for the bismuth ore produced in Mexico. In 1908, the Cananea district made some shipments of bismuth. A number of other deposits are undeveloped on account of lack of transportation.

Norway.—The Kjenner bismuth mines made some test runs in 1910. The mines were operated in 1911 but it is not known what the output was.

Rhodesia.—At the Hay mine, Mazoe, the gold is being recovered from the ore which occurs in granite but the bismuth is neglected. Molyddenite and pyrrhotite are associated minerals.

Spain.—The bismuth ore production of Spain in 1910 was officially stated as 53,510 kg., but the amount of metallic bismuth is not stated.

Bolivia.—The Republic of Bolivia is probably the first producer of bismuth ores in the world. In Bolivia, bismuth is generally exploited with other metals such as tin from which its separation is easy in the crude state. Bismuth ores are found in Huayna, Potosí, in La Paz, and Chorolque, in Potosí. The principal region is that of Tazna. The largest producer and the only bismuth smeltery is at Chorolque. Owing to the control of the bismuth price by the syndicate, it is stated that the price of bismuth has been held at $1.60 per pound for a number of years. From 1904 to 1908 the exports of bismuth amounted to 1,220,824 kilos valued at 7,047,399.60 bolivianos or an annual average of 203,470 kilos valued at 1.174,566.60 bolivianos (about $468,000 or $1.05 per pound export price). The exports of Bolivia appear to have fallen off considerably as far as bismuth is concerned although actual statistics are not available for the year 1911. On February 9, 1910, an export duty was imposed so that at present one quintal of bismuth in bars pays 10 bolivianos, and 1 quintal of bismuth in small bars, 7 bolivianos, export duty. Most of the export trade goes to Europe. During the first 6 months of 1910, as is shown in tables for countries, 96,574 kilogrm. were exported as compared with 88,993 kilogrm. in a corresponding period in 1909.

Australia.—In North Queensland and New South Wales, there is a large production of bismuth ores. The Glenn mines in North Queensland have been described by W. C. Walworth Pearce.[1] The ore runs from 2 to 7 per cent. metallic bismuth and is hand picked as high as 11 per cent.

[1] Transactions Inst. of Min. and Metlgy., XXI, Third ordinary meeting.

The veins were first opened in 1905 for tungsten ores. The carbonate of bismuth was at first unnoticed but later became the chief mineral sought for, as it was easily separated. The penalizing for the presence of bismuth by some buyers practically caused the abandonment of this land until 1909, and tungsten then became a secondary product. The veins are nearly horizontal in biotite granite and are from a few inches to 2 ft. in thickness; the average is about 10 in. The principal associated mineral is topaz which occurs in innumerable small crystals. Metallic bismuth is also found. The wolframite occurs to the extent of 1 to 2 per cent., and it is difficult to increase the tungsten percentage in any product by sorting. In concentrating this ore a spring dolly is used and the ore is then jigged. The sliming of the bismuth carbonate causes considerable loss but it is not known how efficient the present processes are. With the introduction of machinery, it will probalby become more economical.

In New South Wales, at the Deep Water mine, the ore occurs in a similar way, the wolframite predominating. At Whipstick, Shafer and Robertson raised 50 tons of bismuth ore which yielded 3 1/2 tons of concentrates, valued at £479. The mines at Kingsgate, 22 miles from Glen Innes, where bismuth occurs associated with molybdenum, were idle in 1911; some fossickers won a small amount. From 1880 all told 536 tons of bismuth have been exported from N. S. W.

The bismuth production in 1910 from Queensland was 21 short tons valued at £9708. In 1911, 9.75 tons bismuth ore valued at £5525, were produced besides 129 tons of bismuth tungsten ores valued at £11,564.

Tasmania produced 11 tons, £4249 in value in 1910.

The bismuth produced in Tasmania was worth $28,020 in 1911.

PRODUCTION OF BISMUTH MINERALS IN AUSTRALIA.

(Tons of 2240 lb.)

	Queensland.		New South Wales.	
	Tons.	Value.	Tons.	Value.
Previous to 1903	345.0	£66,285	483.0	£64,535
1903	11.0	2,523	21.7	9,537
1904	20.9	3,581	40.3	12,329
1905	15.3	5,368	55.8	20,763
1906	6.5	1,882	25.9	5,700
1907	6.3	1,806	16.3	5,268
1908	22.7	10,595	8.7	2,017
1909	10.3	2,271	8.6	1,624
1910	21.0	9,708	6.4	2,004
1911	9.8	5,525	7.9	1,800

BORAX

There was no marked change in the borax industry in the United States during 1911, the competition between the producing companies resulting in a stimulated consumption. In the United States, borax was formerly found in the waters of Clear Lake in Northern California, the water when evaporated giving borax. Then it was found in the surface crust of desert marshes in California and Nevada, and later on was leached out of the clay formations. Recently, the great deposits that are worked are all calcium borate. Crystallized borate minerals in the tertiary clays of Southern California promise an ample supply for the world for many years to come. The formation extends in a semi-circular belt around the southern end of the Sierra Nevada mountains. Along Furnace Canyon on the eastern side of Death Valley, in the Amargosa desert in the low mountains north of Daggett, in the Mojave Desert, in the Cajon Canyon and the Santa Paula Valley, have been the chief sources of American supply.

The original deposits of Daggett were poor, carrying only the equivalent of 6 per cent. of boric acid, but the smaller companies have all given up and the two larger ones are now working what is known as the high-grade ore. This occurs in a formation from 40 to 50 ft. thick and carries 12 to 40 per cent. boric acid.

The colemanite is crystalline and resembles ordinary Iceland spar or calc spar, for which it has been very often mistaken. The analysis shows boric acid and other impurities, according to the localities. That from Lang, California, has the following analysis:

	Per cent.
B_2O_3	36.10
CaO	23.74
Al_2O_3	1.27
Fe_2O_3	1.32
MgO	2.36
CO_2	5.14
SiO_2	13.97
Water	16.10
	100.00

As borax contains 36.67 per cent. B_2O_3, 1 lb. of ore will make nearly 1 lb. of borax.

The Turkish boracite is found in Asia Minor near the Rhyndamus River whose outlet is in the sea of Marmora, and the area is about 20 square miles.

Pendermite occurs in an enormous bed of gypsum which is covered with several feet of clay. The nodules of ore are of various sizes, weighing as much as 1 ton. This ore is sold to European markets on the basis of 44 per cent. B_2O_3.

Turkey.—The Borax Consolidated Company works a concession near Sultan Tchair (Aidim). The boracite bed is a brown to bluish-gray gypsum, at a depth of 250 ft. The boracite nodules are dispersed in the gypsum and constitute 15 per cent. of the mass, the yearly output being about 15,000 tons.[1] It is estimated that 16,000 metric tons of boracite were produced in Turkey in 1911.

Peru and Chile.—The mineral that comes from Ascotan is a borona-trocalcite and the composition is as follows:

BORONATROCALCIT FROM ASCOTAN.

	Per cent.
B_2O_3	38.04
H_2O	19.86
Na_2O	15.91
CaO	12.34
MgO	0.37
Fe_2O_3 and Al_2O_3	0.24
SiO_2	4.95
Cl	9.64
SO_3	0.84
	100.00

Other places producing borax mineral are in the province of Ocatoma, Maricunga and Copiago. The manufacture of borax differs according to the raw material used and many methods are practicable. With ordinary colemanite, the ore is crushed to the required fineness, which depends on the kind of a mill to which it is fed afterward. The weighed ore is then boiled in the decomposers, filter presses, and the liquor crystallized in tanks. The crystallizers are fitted up with vertical iron rods on which the crystals are collected. In the case of pandermite, it is reduced in about the same way but a different type of mill is used. The mixture of ore, soda and bicarb and mother-liquor is boiled in closed vessels at a steam pressure of 60 lb. In France this mineral is decomposed with soda ash only in autoclaves. The liquor contains the biborate and monoborate of soda and to this enough boric acid is added to convert all into biborate. There is made in France, besides the regular borax, the octahedral borax which contains 5 molecules of water. The liquor for this purpose is made more concentrated and then cooled down to a low temperature. This is known as jewelers' borax. The boronatrocalcite is worked up in a similar manner. This ore is ground or disintegrated and mixed with the proper amount of bicarbonate of soda for the purpose of treatment.

[1] *Zeitschr. Prakt. Geol.*, IX, 249-262

PRODUCTION OF BORAX IN CALIFORNIA. (*a*)
(In tons of 2000 lb.)

Year.	Tons.	Value.	Year.	Tons.	Value.	Year.	Tons.	Value.
1900.....	25,837	$ 1,013,251	1904...	45,647	(*c*) 698,810	1908...	22,200	1,117,000
1901.....	7,221	982,380	1905...	46,334	1,019,158	1909...	16,628	1,163,960
1902.....	(*b*)17,202	2,234,994	1906...	58,173	1,182,410	1910(*d*)	37,000	1,950,000
1903.....	34,430	(*c*)$661,400	1907...	53,412	$1,200,913	1911(*e*)	53,330	1,569,151

(*a*) Reported by the California State Mining Bureau. (*b*) Mostly refined borax, whence the apparent discrepancy in value. Output of the other years is given as crude material. (*c*) Spot value. (*d*) Crude. From *Eng. and Min. Journ.* (*e*) U. S. Geol. Survey, crude.

SOME OF THE PRINCIPAL SUPPLIES OF BORAX PRODUCTS.
(In metric tons.)

Year.	Chile. (*a*)	Germany. (*b*)	Italy.			United States. (*c*)	Total. (*d*)
			Borax Refined.	Boric Acid.			
				Crude.	Refined.		
1898.....	7,028	230	702	2,650	166	7,529	18,305
1899.....	14,951	183	709	2,674	129	18,466	37,112
1900.....	13,177	232	858	2,491	283	23,437	40,478
1901.....	11,457	184	544	2,558	347	6,550	21,640
1902.....	14,327	196	.....	2,763	.....	15,512	32,798
1903.....	16,879	159	.....	2,583	.....	31,232	50,853
1904.....	16,733	135	569	2,624	314	41,407	61,782
1905.....	19,612	183	(*e*)1,007	2,700	(*e*)749	42,036	64,531
1906.....	28,996	161	1,062	2,561	562	52,774	86,116
1907.....	28,374	114	881	2,305	466	48,444	80,584
1908.....	35,039	128	(*e*)1,024	2,520	(*e*)429	20,140	57,827
1909.....	32,218	149	(*e*)1,110	2,431	(*e*)578	15,083	51,569
1910.....	?	167	(*e*) 912	2,502	(*e*)695	16,780	?

(*a*) Prior to 1903, figures are for borate of lime exports. (*b*) Boracite. (*c*) Crude borax. (*d*) The total falls short of the world's supply, particularly because it fails to include the important production of Turkey. (*e*) Obtained by treating a part of the crude boric acid reported for the same year.

Peru.—The Borax Consolidated (Ltd.) controls the output of the Department of Arequipa in the Province of Moquequa, which is the sole producer of borax. For 1910 the output exported amounted to 2351 metric tons.

Experiments made on the methods used in France are reported by M. G. Levi and O. Garavini.[1] The borax and sulphuric anhydride are produced in France by heating a mixture of sodium sulphate and boric acid. The author's experiments show that the velocity of the reaction is smaller with sodium sulphate than with sodium chloride. Under the conditions requisite to obtain a good borax, practically the whole of the sulphuric anhydride is dissociated, with a formation of sulphur dioxide.

The uses of borax occur in almost every industry. It is said that almost one-half of the borax is consumed in the enameling kitchen ware and is a very essential part of the enamel. It is also largely used as a flux in the manufacture of pottery and earthenware

[1] *Gaz. Chim. Ital.*, 1911, 41, I, 756–781.

glaze. Borax is also used in making certain varieties of glass and the manufacture of artificial gems and strass. In the manufacture of optical glass, it is an extremely important ingredient, and to any glass that should not be susceptible to changes in temperature such as arc lights, lamp chimneys, etc. The coefficient of expansion of borosilicate is small. It is also used for making vitrifiable pigments for staining glass and for the manufacture of encaustic tiles which are rendered fusible by its admixture.

Borax is used in the tannery and in the currying shop. It is of great benefit in the soaks where it dissolves the dirt and the blood and insures a more rapid liming. In the bates for sole and chrome stock, a borax bath has been found very commendable.

It is also used in the first stage of tanning in the liquor. It is used for cutting the oils and fats used in stuffing leather. Borax is also one of the best known agents for whitening and bleaching leather of all kinds; especially russet leather is treated with borax. It is the best known mordant for coloring leather. It really replaces any alkali used in tanning, on account of being more harmless, antiseptic and sweetening.

Borax, for its great cleaning properties, is used in the laundry in washing as well as in the starching. There are several starches on the market containing borax; its admixture helps to give the shirts and collars a very high gloss. Borax is also used as a wood preservative against dry rot.

Another very large use is as a flux for welding and brazing metals on account of its capacity of easily dissolving the metal oxides and leaving a blank surface. The jewelers use it also for the same purpose.

Varnish made by boiling 1 part of borax with 5 parts of shellac is used for stiffening hats. With casein borax forms a substitute for gum which is moisture-proof. The paper mills use it for making a kind of parchment which is fat and waterproof. It is also used in the making of certain sizes and coatings for glazed papers and playing cards.

On account of its antiseptic qualities it is used in pharmacy in cosmetics, mouth washes, tooth powders and salves. The dentists use it for cleaning the teeth. It is used as an admixture for powders for killing insects, roaches, etc. It is also used as a preservative for meats and other foodstuffs, and finally in the manufacture of other boron compounds.

Boron, though quite expensive, is coming into commercial use in assisting the making of solid copper castings of high electrical conductivi y.[1]

Abrasive and Refractory Material of Alumina, Magnesia and Boric Oxide.—In a second patent of Mr. Allen boric oxide is added to the

[1] R. Whitney. Pittsburg Meeting, Am. Brass Founders Association, 1911.

alumina-magnesia product. Boric acid is mixed with calcined magnesia, for instance, in the proportion of 35 to 65, and heated in a fuel-fired furnace to about 1300°. The product is crushed and 25 parts of it are mixed with 100 parts of calcined bauxite and 8 parts coke and this mixture treated in an electric arc furnace. The product contains 12 per cent. magnesia, 6 per cent. boric oxide, and 79 per cent. alumina, and is a dark colored crystalline material of great hardness and very brittle. Addition of boric oxide increases the brittleness and hardness. (1,001,570, Aug. 22, 1911.)

Boron.—A number of patents of Dr. E. Weintraub, of the General Electric Company, deal with an elaborate method in producing pure boron. The usual method for producing boron includes the reduction of boric anhydride with magnesium, the general process being to heat a mixture of magnesium with an excess of boric anhydride. This reaction is accompanied by the production of magnesium borate and by the formation of other compounds. The resultant product, therefore, contains a relatively high percentage of material other than boron. The usual washing with water and hydrochloric acid decreases the percentage of impurity, but does not yield a product which could be termed, in the chemical sense, "pure boron." This impure product can be greatly improved, however, by "fractional decantation," consisting in grinding the product with water or hydrochloric acid and then decanting off the top of the liquid before complete precipitation has occurred. Boron has a greater specific gravity than the impurities, such as magnesium boride. By repeating this treatment, the proportion of impurities can be reduced several per cent.

The resulting product, in the form of a brown powder, is compressed into the form of a stick or rod, say 6 cm. long and 5 mm. in diameter. The stick is then highly heated in an electric vacuum furnace to dissociate and drive off various of the impurities, like magnesium boride and boron hydride. This treatment may be continued for several hours at a temperature of say 1200° C. without any danger of contaminating the boron by the carbon of the resistance heater. As a result of this treatment, an important physical change takes place, since the boron stick which before the treatment was a non-conductor, becomes a relatively good conductor of electricity. The next step in this process consists in melting down this conductive boron by heating it as anode in a vacuum arc. For this purpose an electric vacuum furnace is used in form of a hermetically closed glass vessel, with some mercury as cathode and the boron stick as anode. Hydrogen gas is introduced into the vessel and a pressure of the hydrogen of from 2 1/2 to 15 cm. of mercury is maintained. This hydrogen gas is chemically inert, but serves to con-

centrate the arc on the lower end of the boron stick. While chemical text-books describe boron as being a non-conductive material which vaporizes at high temperatures without melting, the boron produced by the above process is a relatively good conductor of electricity, is a dense and solid body and undergoes complete fusion without apparent vaporization. Such pure conductive boron may be used for a variety of purposes and may be shaped into rods or glow bodies for use in incandescent lamps and furnaces. (997,879; 997,880; 997,881; all of July 11, 1911.)

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BROMINE AND IODINE

There is apparently an overproduction of bromine in this country, and new uses are desired as an outlet for this element.

PRODUCTION OF BROMINE IN THE UNITED STATES.
(In pounds.)

Year.	Michigan. (a)	Ohio and Penn.	West Virginia.	Total. (a)	Metric Tons.	Value.	
						Total.	Per lb.
1898	141,232	226,858	118,888	486,978	220	136,354	28c.
1899	138,272	193,518	101,213	433,003	196	125,571	29
1900	210,400	196,774	114,270	521,444	237	140,790	27
1901	217,995	227,062	106,986	552,043	250	154,572	28
1902	226,452	194,086	93,375	513,913	233	128,742	25
1903	320,000	180,000	97,000	597,000	271	170,145	28½
1904	646,249	147,807	85,256	879,312	399	215,431	24½
1905	579,434	223,000	97,000	899,434	408	139,492	15½
1906	955,000	203,000	71,000	1,229,000	553	184,350	15
1907	(b)	(b)	(b)	1,062,000	482	138,060	13
1908	(b)	(b)	(b)	1,149,000	521	103,410	9
1909	(b)	(b)	(b)	1,100,000	500	110,000	10
1910 (c)	641,300	155,000	54,000	850,300	386	110,539	13
1911	925,000	164,000	89,000	1,179,000	536	235,500	20

(a) Includes the bromine equivalent of the bromides produced directly. (b) Not reported separately. (c) From *Eng. and Min. Journ.*

The production in the United States which is connected with the salt industry, is in the hands of a few plants. The production in the table for 1911 is estimated from reliable figures. About 350,000 lbs. of the Michigan production was in the form of metallurgical salt which is a mixture of sodium and potassium, bromide and bromate, assaying 72 per cent. bromine. Another 400,000 lb. is potassium bromide, 85,000 lb. being sodium bromide and 58,000 lb. ammonium bromide. Most of the Pennsylvania production of bromine is from the John A. Beck Salt Co. and goes to the Hayden Chemical Co. of New York.

As it is necessary to purify crude bromine for chemical and pharmacy use, the price has been higher to cover the extra process and the loss of bromine incidental thereto. This dates from 1906, when a law was enacted defining the purity. The crude bromine is usually 94 per cent. Br.

Practically no bromine is imported into the United States, the Germans having withdrawn from the market in 1908. Germany exported in 1911, 228,900 kilos of which 124,800 went to Great Britain and 62,200 to France. The prices have remained constant throughout the year, probably being controlled by the Dow Chemical Company in this country.

The following new uses have been suggested for bromine or its compounds. A tanning process described by Meunice and Seyewetz[1] is based on the fact that bromine water renders water soaked sheets of gelatine or skin insoluble in boiling water. The amount of bromine fixed in exposing gelatine for 2 to 3 hours to .5 per cent. bromine solution and then washing for 48 hours, is found to be from .5 to 2 per cent. on the direct material. The skin may be tanned by any other method using the advantage that very little bromine is fixed in this process when used at first.

The use of bromates in the cyanide process for gold ores is mentioned by Charles L. Parsons.[2] Bromates are a by-product in the manufacture bromides and so far have had but little market. Bromate is, however, an excellent oxidizing agent and its use for this purpose might be greatly extended.

A patent for the manufacture of bromides from bromine was issued to the Dow Chemical Company, July 18, 1911. Lime and sulphur are boiled together to form calcium sulphydrate. This latter being boiled with bromine gives a mixture of calcium bromide and hydrobromic acid. After neutralizing with lime, the calcium is precipitated with an alkaline sulphate, leaving the alkaline bromide in solution.

Iodine

As far as known, no iodine is manufactured in the United States. There are possible sources in some of the seaweeds but such an industry could only be established as a source of potash, the iodine being a possible by-product. This is the basis of a small production in Norway and also Japan.

The chief source of iodine is the mother liquor from the Chilean niter works. The iodine exists in the mother liquor as an iodate in the proportion of 3/5 grm. per liter. A portion of the mother liquor is treated with sodium bisulphite which first liberates free iodine and then reduces that to hydriodic acid. This reduced mother liquor is then mixed with a proper proportion of the original mother liquor, the hydriodic acid and the iodic acid liberated in the untreated solution reacting with complete precipitation of the iodine, if the conditions be correct.

The Iodine Combine[3] formed in 1886 keeps the price of iodine up to an artificial level, said to be 10 times the total cost of production.

The Domeyko works in the north of Chile are the largest in the world with an annual capacity of 400,000 lb. The total production of Chile in 1910 amounted to 1,279,150 lb.

[1] Collegium, 1911, 289.
[2] *Jl. Ind. and Eng. Chem.*, IV, 3, 185.
[3] According to Consul A. A. Winslow Valparaiso this Combine that regulates the production of iodine in Chile, and thus fixes the price, has completed arrangements to continue still another 3 years.

IODINE IMPORTS INTO THE UNITED STATES.

Year.	Crude.		Resublimed.	
	Pounds.	Value.	Pounds.	Value.
1907	529,080	$1,085,586	246	$561
1908	175,929	342,777	5	15
1909	12,910	25,705	51	110
1910	771,083	1,051,036	7	24
1911	423,353	841,559	55	181

Iodine imports into Germany during 1911 amounted to 3,029,000 kilos, of which 2,356,000 came from Chile; 22,000 from Peru; 14,800 from Norway, and 7600 from Japan. The total exports from Germany were 72,700 kilos in 1911. The imports in 1910 were 362,600 kilos. There was no change in the prices of iodine or iodine products during 1911 in Germany.

Prices.—The price of iodine is reputed to be one that is highly artificial. It is certain that the output could be many times multiplied with little effort and the price greatly lowered if new uses could be found.

A French patent (429,195) claims the use of mine waters, or waters from manufacturing industries, likely to contain iodine for extraction by the usual methods. Until new uses are found, considering the artificially high price, the value of such a patent is doubtful.

CADMIUM

The production of cadmium in the United States during the year 1911 was estimated at 27,000 lb. compared with 5300 lb. in each of the years 1909 and 1910. In the United States there was also a production of about 3000 lb. of cadmium sulphide. The principal production of cadmium comes from Upper Silesia, Germany, where the production in 1911, as stated by Paul Speier of Breslau, amounted to 42,000 kilos from the Prussian Smelting and Metallurgical Works. The quotations for this material started at 600 marks and in response to the comparatively good demand, gradually rose to about 700 marks per 100 kilos, f. o. b. railroad at the works. The price of cadmium in 100-lb. lots in Cleveland, Ohio, was 70 to 85 cents per pound depending on the quality or purity. The principal source of cadmium in Germany is from the zinc smelters in Silesia as a by-product. In the United States the recovery of cadmium as a by-product, is being considered by a number of works in addition to those which have produced cadmium, the recovery of which started in 1907. In the United States, the Grasselli Chemical Company of Cleveland, Ohio, was the first to recover cadmium as a by-product although the metal had been recovered in Germany before 1880. It is estimated that the average western spelter contains .03 per cent. of cadmium which is not separated. Cadmium is deleterious in spelter especially if used for lithopone; this is on account of the yellow color of the sulphide. The flue dust from blast-furnace works carries from 1 1/4 to 1 1/2 per cent. of cadmium which is 5 1/2 per cent. of the zinc in the flue dust.

Cadmium has also been experimented with by the American Smelting and Refining Company which has undertaken the recovery of the metal from flue dust. The company began production at its Globe plant in Denver in 1910. The presence of cadmium in the zinc ores of Leadville, Colo., and Kokomo districts was noted by Bang.[1] The flue-chamber and bag-house dusts from Leadville show a small percentage of cadmium.[2] It is reported that furnaces having a capacity of 1000 lb. per month are contemplated.

The zinc ore produced in 1910 in the southwest, about 432,000 tons, contained 1,678,000 lb. of cadmium or 20 times the consumption in 1910 for the world and 85 per cent. of this material is lost in roasting or frac-

[1] Mineral Resources of United States. United States Geological Survey for 1905, p. 384.
[2] A. Guyard, Monograph, U. S. G. S., Vol. XII, 1886, p. 712.

tional distillation of ore. The Cleveland Smelting Company which recovers cadmium found that the Joplin spelter averaged .358 per cent. cadmium and began using this ore to some extent for this reason. There is not enough cadmium used, however, to consume all the Joplin ore that could be treated.

Until recently nearly all the cadmium used in the United States was imported. Cadmium is imported as metallic stick cadmium or as cadmium sulphide or cadmium yellow. Metallic cadmium is free of duty. The value of imports for consumption averages $4000 per year and reached a maximum in 1906 when 13,808 lb. valued at $10,522 were imported. The imports in 1910 were 4060 lb. valued at $2295 and in 1911, 5367 lb. valued at $3870.

Cadmium is a white crystalline metal that promises a number of uses in the arts on account of its properties. The high price would appear to keep workers from trying it in experiments which would tend to extend its uses. If the demand becomes greater it should within reason be produced as cheaply as aluminum.[1] Cadmium is used in dentistry, electroplating, medicine, amalgam and alloys. One-half per cent. of cadmium in brass impairs its value. Generally the alloys of cadmium are malleable and of a lower melting-point; in the case of gold, platinum, copper and zinc, however, the result is a brittle alloy. The melting-point of cadmium as given by Kahlbaum is 320.4° C. and it rises with heavy pressure.[2] The dental amalgam is composed of 26 per cent. cadmium and 74 per cent. mercury with tin or bismuth added. Stereotype plates or cliche metal is 50 per cent. lead, 27 1/2 per cent. tin and 22 1/2 per cent. cadmium, this being harder than the corresponding bismuth alloy. One of the chief uses is in fusible alloys in which cadmium replaces bismuth. Cadmium also makes a good yellow pigment for paints but it is expensive. A silver cadmium plating alloy came into use to a very small extent, but this process has probably been abandoned on account of its difficulty. The electro-deposition of metallic cadmium is described by Emanuel Blassett, Jr.[3] Cadmium cannot hope to replace nickel for this purpose on account of its higher price. Were it not for the expense, it has some advantages over nickel. Cadmium plate adheres tenaciously and requires no "strike" solution. The first patent for cadmium plating dates to 1849 in England and the solution then patented has been found to give excellent results. It consists of cadmium carbonate dissolved in potassium cyanide.

One of the new uses for metallic cadmium is in the manufacture of tungsten electric light filaments. An alloy of cadmium 42 per cent.,

[1] W. R. Whitney, Pittsburgh Meeting of American Brass Foundrymen's Association, 1911.
[2] *Am. Jl. of Science*, June, 1911.
[3] The Metal Industry, Dec. 16, 1911.

mercury 53 per cent. and bismuth 5 per cent. is impregnated with tungsten powder about 30 per cent. of the whole. This is accomplished by heating and grinding in a mortar. The mixture is made into wire by the extrusion method as in lead-pipe manufacture and then passed through a die. It is finally heated to drive off the alloy and finished by heating in a vacuum[1] to render the tungsten solid.

As the boiling-point of zinc is 930° C. and that of cadmium 778°, this property is used to bring about a concentration of the cadmium in the powder deposited during the first distillation. If the oxidized metals are recovered by distilling them with reducing agents, the comparative ease of reduction of the cadmium oxide at a lower temperature than the zinc is made use of in a similar manner. The process is troublesome and was formerly the only one used. Many distillations were necessary to make pure cadmium. The purification of zinc solutions for lithopone is done by adding zinc powder which precipitates the cadmium from the boiling solution, or as at Marienhutte, by electrolysis. Here a zinc anode deposits the cadmium from the solution on a lead-lined tank which orms the cathode.

M. Wiedeman of the Hohenlohehütte has patented a process of separating alloys of zinc-lead-cadmium by acids and the process is said to recover 20 per cent. of the cadmium and will allow the use of waste from sulphuric acid plants.

Cadmium has been found in cyanide bullion sent to the United States assay office. It is derived from the zinc used in precipitating the gold. A sample from the Goldfield Consolidated Company, supposed to be a cyanide bar, did not show the cadmium but it also did not show zinc.[2]

[1] *Brass World*, November, 1911.
[2] F. P. Dewey, *Engineering and Mining Journal*, April 13, 1912.

CALCIUM CARBIDE

In the manufacture of carbide the raw materials used are lime or limestone, coke or anthracite, and in addition to this cheap power must be available. It has been estimated in a general way that 1 electric horsepower a year a ton of carbide is required but in large works such as at Odda in Norway, this figure is decreased, for example: this factory makes more than 30,000 tons of carbide a year and the power plant consists of six turbo alternators, each of 4000 h. p., one being a standby. The plant contains 12 furnaces of 1500 h. p. each and one large one of 2000 h. p. Yearly 10,000 tons, largely fines, go to the cyanamide works. The plant is to be doubled in order to supply these works. The purity of the raw materials is required to be such that the coal contains only a small quantity of ash, and the limestone, no magnesia, nor phosphorus. The proportion of ingredients is about 2000 lb. of limestone to 1200 lb. of anthracite for 1 ton of carbide, the lime being somewhat in excess of the theoretical.

Carbide is largely employed for the production of acetylene and in the manufacture of fertilizers such as nitrolim made indirectly from carbide and from nitrogen extracted from the air. At Odda, nitrolim works have been established close to the carbide.

In recent years the selling price of carbide has been very much cut, but there is no doubt that the demand will increase.

The price of carbide in England has been under £12 per ton. Competition is keen and there are no combines.

In the United States two companies control the output, but no statistics are available for the total output. The world's production of calcium carbide in 1909 amounted to about 250,000 tons, one-fifth of which was produced in the United States and Canada. The production in other countries was as follows (in tons): Italy, 43,000; France, 28,000; Norway, 30,000; Switzerland and Austria-Hungary, 20,000 tons each; Spain and Portugal, 18,000; Sweden, 15,000; Germany, 9000; England, 2000. In 1910 Italy produced 36,115 tons.

The exports in 1910 were 25,933,670 lb. valued at $732,574; in 1911 they were 28,836,987 lb. valued at $827,857; the distribution was principally in North and South America, although some went to Africa and Australia. It was about as follows in 1911: To Cuba, 12,954,939 lb.; Mexico, 1,982,665 lb.; Argentine, 5,590,800 lb.; Brazil, 733,999 lb.; Uruguay, 428,500 lb.; Venezuela, 304,297 lb.;

New Zealand, 369,860 lb.; South Africa, 534,271 lb. exported from the United States. The United Kingdom imported in 1909 26,480,384 lb. valued at $648,574; in 1910, 28,997,808 lb. valued at $648,617 and about the same in 1911. The exports were 820,400 lb. valued at $22,493 in 1909; and 85,568 lb. valued at $1878 in 1910.

In 1909 Norway exported 106,597,133 lb. worth $1,554,990. Sweden exports 3,429,294 lb. valued at $84,286; imports 156,346 lb.. Germany exports 2,133,391 lb. valued at $39,270; imports 59,427,859 lb. valued at $1,026,494. France imports 15,985,775 lb. valued at $58,950; exports 7,566,628 lb. valued at $145,731. Spain produced 7282 metric tons in 1910, valued at 2,093,000 pesetas.

Australia.—The manufacture of carbide of calcium has up till now not been attempted in Australia, although the importation of the carbide has reached large proportions. The Carbide Manufacturing Syndicate Ltd., of Sydney, is now erecting works at Waterloo. Most of the imported carbide is from Sweden and the United States.

Oxyacetylene.—The Linde Air Products Company has acquired sites for the manufacture of liquid air to be used in oxyacetylene work, at North Trafford, Pa., and South Elizabeth, N. J. It is proposed to lay pipe-lines to the Pittsburg district where the consumers are the steel works, foundries, etc. A plant for oxygen and hydrogen manufacture is being erected near Liverpool, England. The capacity is to be 200,000,000 cu. ft. of gas per year; the price for oxygen is expected to be as low as $1.21 per 1000 cu. ft. in pipe-lines from holders or $4.86 in cylinders.

Oxyacetylene is an improvement on oxyhydrogen for the blow pipe. Acetylene is now furnished in compressed form in tanks at a pressure of about 120 lb. to the square inch. The use of acetone which dissolves 25 volumes of the gas per atmosphere at 60° F., is universal. About 40 per cent. of the tank volume is filled with the liquid and an absorbing material such as asbestos. The Commercial Acetylene Company at 80 Broadway, New York City, is a large producer with a plant at Joliet, Ill. Acetylene is generally used now for cutting off steel work and also for welding. A full description of the Fouché blow pipe and other accessories, as well as experiments on the strength of welding, is found in a paper by Herbert L. Wittemore.[1]

The industrial uses of acetylene have been established and in a similar manner the incidental uses of oxygen have made it an article of commerce. The gas *hydrogen* which has been a by-product, is also used extensively, lately, in various industries such as the reduction of oils to waxes, in aeronautics, etc., and it promises to have a wide use on account of its convenience, as in the cottonseed oil industry.

[1] University of Illinois, *Bul.*, Sept., 26, 1910.

The use of acetylene in lighting is well known. Formerly underground lamps using carbide were not considered reliable, but in the last few years such lamps are finding favor, many of the difficulties being overcome. Among manufacturers now furnishing lamps of this class are the John Simmons Co., 100 Center St., New York, making the Baldwin Lamp; the Scranton Acetylene Lamp Co., 144 Belmont Terrace, Scranton, Pa., and Hughes Bros., Scranton, Pa. Most of these lamps are rather heavy to wear on the hat and require recharging probably once a day. A lamp is also manufactured supplied from a small generator by means of a rubber tubing. The generator is carried in a pocket and only the burner and reflector are carried on the hat.

CEMENT

By Robert W. Lesley

The cement industry for the year 1911 compares unfavorably with previous years in this industry. So far as the production of natural cements, puzzolan and cements other than Portlands are concerned, there was an actual decrease. However, the output of Portland cement brings the production of all character of cements in this country in 1911 to 79,547,958 barrels as compared with 77,785,141 barrels in 1910. The value of the cement produced in 1911 was $66,705,136 as against a valuation in 1910 of $68,752,092. The figures therefore show an increase in barrelage in 1911 of 1,762,817 barrels, or 2.27 per cent., but a decrease in value of $2,046,956 or 2.98 per cent. In the case of Portland cement alone the figures are 78,528,637 barrels in 1911 valued at $66,248,817 as compared with 76,549,951 barrels in 1910 valued at $68,205,800 or an increase in output of 2.58 per cent. with a decrease in value of 2.87 per cent. From these figures it will be seen that an actual loss was sustained in the amount received for the cement sold.

PRODUCTION OF CEMENT IN THE UNITED STATES. (a)
(In barrels.)

Year.	Portland.			Natural Hydraulic.			Puzzolan Cement.			Total.	
	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.
1899	5,805,620	$10,441,431	$1.80	9,686,447	$ 5,058,500	$0.52	244,757	$ 360,800	$1.47	15,736,824	$15,786,789
1900	8,482,020	9,280,525	1.09	8,383,519	3,728,848	0.45	446,609	567,193	1.27	17,312,148	13,576,566
1901	12,711,225	12,532,360	0.98	7,084,823	3,056,278	0.43	272,689	198,151	0.73	20,068,737	15,860,731
1902	17,230,644	20,864,078	1.21	8,044,305	4,076,630	0.50	478,555	425,672	0.81	25,753,504	25,366,380
1903	22,342,973	27,713,319	1.19	7,030,271	3,675,520	0.50	525,896	542,502	1.03	29,899,140	31,931,341
1904	26,505,881	23,355,119	0.90	4,866,331	2,450,150	0.50	303,045	226,651	0.75	31,675,257	26,031,920
1905	35,246,812	33,245,867	0.94	4,473,049	2,413,052	0.54	382,447	272,614	0.71	40,102,308	35,931,533
1906	46,610,822	51,240,652	1.10	3,935,151	2,362,140	0.60	481,224	412,921	0.86	51,027,321	54,015,713
1907	48,785,390	53,992,551	1.10	2,887,700	1,467,302	0.51	557,252	443,998	0.79	52,230,342	55,903,851
1908	51,072,612	43,547,679	0.85	1,686,682	834,509	0.49	151,451	95,468	0.63	52,910,925	44,477,653
1909	62,508,461	50,510,385	0.80	1,527,279	623,141	0.41	160,646	99,453	0.62	64,196,386	51,232,979
1910	76,549,951	68,205,800	0.89	1,500,000	600,000	0.40	160,000	96,000	0.60	77,785,141	68,752,092
1911	78,528,637	66,248,817	0.84	926,041	378,533	0.41	93,230	77,786	0.33	79,547,958	66,705,136

(a) Statistics of production for 1900 and subsequent years are as reported by the U. S. Geological Survey. The barrel of portland cement contains 380 lb. of the material; of natural cement, 265 lb.; of slag cement, 330 lb.

STATISTICS OF CEMENT IN THE UNITED STATES.

Year.	Production.		Imports.		Exports.		Consumption. (*b*)	
	Barrels.	Value.	Barrels (*a*)	Value.	Barrels.	Value.	Barrels.	Value.
1899...	15,736,824	$15,860,731	2,219,246	$2,858,286	116,079	$213,457	17,839,991	$18,505,560
1900...	17,312,148	13,576,566	2,512,300	3,330,445	147,305	289,186	19,677,143	16,617,825
1901...	20,068,737	15,786,789	994,624	1,305,692	303,380	752,057	20,759,981	16,340,424
1902...	25,735,504	25,366,380	2,100,513	2,582,281	367,521	575,268	27,468,496	27,373,393
1903...	29,899,140	31,931,341	2,439,948	3,027,111	312,163	466,140	32,026,925	34,492,312
1904...	31,675,257	26,031,020	1,101,361	1,383,044	816,640	1,158,572	31,959,978	26,256,392
1905...	40,102,308	35,931,533	891,134	1,102,041	1,060,054	1,428,489	39,933,308	35,605,085
1906...	51,027,321	54,015,713	2,321,803	2,950,268	600,386	964,373	52,748,738	56,001,608
1907...	52,230,342	55,903,851	2,006,228	2,637,424	900,550	1,450,841	53,336,020	57,090,434
1908...	52,910,925	44,477,653	839,247	1,189,560	846,528	1,249,229	52,903,425	44,417,984
1909...	66,679,356	53,580,948	431,785	642,397	1,056,922	1,417,534	66,054,219	52,805,811
1910...	77,785,141	68,752,092	292,314	396,428	2,475,957	3,477,981	75,601,498	65,670,539
1911...	79,547,958	66,705,136	163,802	254,258	3,135,409	4,632,215	76,576,351	62,327,179

(*a*) Barrels of 400 lb. (*b*) Includes all kinds.

STATISTICS OF CANADIAN CEMENT. (*a*)

Year.	Natural Cement.			Portland Cement.			Imports.
	Bbl.	Value.	Per Bbl.	Bbl.	Value.	Per Bbl.	Bbl. (*b*)
1898.....	87,125	73,412	.842	163,084	324,168	1.987	.....
1899.....	141,387	119,308	.843	255,366	513,983	2.012	.....
1900.....	125,428	99,994	.797	292,124	562,916	1.927	.....
1901.....	133,328	94,415	.708	317,066	565,615	1.783	555,900
1902.....	127,931	98,932	.773	594,594	1,028,618	1.729	544,954
1903.....	92,252	74,655	.809	627,741	1,150,592	1.834	773,678
1904.....	56,814	50,247	.884	910,358	1,287,992	1.414	784,630
1905.....	14,184	10,274	.724	1,346,548	1,913,740	1.421	918,701
1906.....	8,610	6,052	.703	2,119,764	3,164,807	1.493	665,845
1907.....	5,775	4,043	.704	2,463,093	3,777,328	1.555	672,630
1908.....	1,044	815	.781	2,665,289	3,709,139	1.390	469,049
1909.....	.....	.....	.....	4,067,709	5,345,802	1.319	142,194
1910.....	*Nil.*	.....	.....	4,753,975	6,412,215	1.340	349,310
1911.....	*Nil.*	.....	.....	5,635,950	7,571,299	1.342	661,916

(*a*) Canada Department of Mines, Mines Branch. (*b*) Bbl. =350 lb.

When the very small percentage of increase of production shown by the 1911 figures is contrasted with the enormous increases in the industry in previous years, both in number of barrels and in value, it can be readily understood why the trade has been in a depressed condition. Looking back over the figures for many years it will be seen that the present figures show the lowest percentage of increase in output since 1890, the first year in which the official figures were based on complete returns from all producers. The nearest approach to it was in 1892 and 1893 when the production was 547,440 barrels in 1892 which increased to 590,652 barrels the following year.

Hitherto in considering the Portland cement industry, figures by states have been given and figures by districts have been presented, the districts being broadly divided into East, West, South and Pacific. This year, however, for the first time, the industry is being considered from a new standpoint of distribution, namely, that of commercial geographical handling. In other words, Portland cement, which is a commodity weighing 380 lb. to the barrel, can no longer be shipped great distances from the manufacturing point as was the case in the early days of the industry. When it was first imported from Europe, Portland cement was shipped by lake and rail or by canal and lake as far west as Kansas City, and the early American mills owing to the demand for this new material of construction were enabled to extend their markets practically all over the United States. By degrees, mills east, west and south have been constructed and the American Portland cement industry as a manufacturing proposition is becoming more and more one of proximity to the ultimate consumer. It would seem that with a commodity of the weight of Portland cement, averaging from 40 to 50 cents per barrel for transportation charges from the mills to consumer, that the question of cross-shipments is one that must be in time regulated, as this fact largely accounts for the very low return cement manufacturers have been receiving of late years for their product. In Germany by Cartel agreements restricting cross-shipments from one territorial group into the territory of another territorial group, prices have been maintained at reasonable figures and mills have realized some return on their capital. In this country the statistics show that the manufacturer in many sections has received less for his cement than in any previous year in the history of the industry. It is to be hoped that some method will be arrived at to avoid the conditions that have produced the above state of facts.

In the distribution of the country into commercially localized districts a much more intelligent idea of the trade of the country can be had, as well as of conditions prevailing in the various districts. The figures on page 121 for the years 1910 and 1911 are of interest:

By this grouping the Lehigh District still retains its old status embracing the plants in New Jersey and eastern Pennsylvania. The mills in western Pennsylvania around Pittsburg logically combine with those along the Ohio River, while the mills in New York State, located as they are along the Hudson River and the waterway system of that Commonwealth, are grouped together as they command, practically, a single geographical territory. The Chicago district properly embraces the mills in Illinois and northeastern Indiana which have freight rates enabling them to command the market in question. The southern grouping embraces eleven mills commanding the territory of the south,

PRODUCTION OF PORTLAND CEMENT IN 1910 AND 1911 BY COMMERCIAL DISTRICTS.

District.	Active Plants 1910.	Active Plants 1911.	Production 1910 Barrels.	Production 1911 Barrels.	Change 1911 P. ct.	Average Factory Price per Barrel 1910.	Average Factory Price per Barrel 1911.
New Jersey and eastern Pennsylvania (Lehigh district).	24	24	26,315,359	25,972,108	1.30	$0.729	$0.715
New York	8	7	3,296,350	3,314,217	.54	.882	.805
Ohio and western Pennsylvania.	9	9	6,072,987	6,756,313	11.25	.776	.766
Michigan and northeastern Indiana.	14	13	4,524,591	4,519,726	.11	.921	.827
Kentucky and southern Indiana.	3	3	2,824,832	2,818,820	.21	.799	.793
Illinois and northwestern Indiana.	6	6	8,376,450	8,617,341	2.88	.940	.791
Southeastern States (Maryland, Virginia, West Virginia, Tennessee, Georgia and Alabama).	8	11	3,071,009	4,049,063	31.85	.794	.793
Iowa and Missouri	6	7	5,722,971	6,067,449	6.02	.916	.862
Great Plains States (Kansas, Oklahoma and Central Texas.)	16	17	7,723,253	7,010,396	9.23	.996	.834
Rocky Mountain States (Colorado, Utah, Montana, Arizona and western Texas).	8	7	2,236,561	2,124,930	4.99	1.288	1.186
Pacific Coast States (California and Washington.)	9	11	6,385,588	7,278,274	13.98	1.385	1.406
Total	111	115	76,549,951	78,528,637	2.58	.891	.844

while the Pacific coast territory in turn embraces the mills of California and Washington. By an examination of the above table the logic of the distribution will be appreciated.

It will also be noted from the foregoing figures that the Lehigh District, which was formerly the principal manufacturing point in the United States, is gradually falling behind so far as its percentage of the total output of the country is concerned. The figures also show that it has fallen behind in prices as compared with other districts, the figures for 1911 showing slightly above 70 cents as compared with $1.40 on the Pacific Coast. These figures, however, are average ones only and do not represent the actual price conditions prevalent in the latter months of 1911 in the eastern part of the country, and more especially in the Lehigh district. As many large contracts made in previous years at prices running from 75 to 85 cents expired, new engagements were made by the mills as low as 60 and 65 cents.

Conditions in the cement industry during the past year appear to have been due to excessive promotion of new plants in the years 1908 and 1909, and to the substitution in many plants of the long kilns for the shorter kilns heretofore in use, the effect of this latter change being to largely increase outputs of existing mills. While it is true that there is an apparent excess of capacity in this country over the demand, as shown by the 1911 figures, it must not be forgotten that this excessive

capacity is a largely potential one and represents, in many cases, mills that have no large local territory in which to put their material and which can only exist in times of prosperity; it also represents many old plants owned by well-established companies which plants are not in operation and which could only be run under the most favorable market conditions. It must also be considered that the usual winter shut-down is becoming more and more a recognized factor in the cement industry by reason of the greater economies of continuous summer running and the greater efficiency of the machinery when the raw material is in its most favorable condition. Consequently, with renewed prosperity in the country the cement industry is bound to get its share and prices are likely to advance. This will be largely due to the enormously increased purposes for which cement is used. The Association of American Portland Cement Manufacturers has built up a large outlet for Portland cement among the farmers of the country, and all through the farming regions of the West and Middle West the farmers are among the largest consumers of Portland cement, and much of the work hitherto done with lumber is being done with concrete in a permanent and efficient way. Another great field is being opened up in the construction of roads. In the State of Michigan alone there are over 100 miles of concrete roads in successful use and in many parts of Indiana and Illinois similar roads are being constructed. In the demand for better highways, coupled with a demand for the best highways, cement will find an enormous new outlet and an outlet which will be practically local to all the mills in the country as it will be in their own immediate vicinities. Concrete lumber, to be sold in lumber yards precisely as the ordinary lumber of commerce is sold, is another new field and one which is being rapidly developed in Germany. Plants are already being established in this country for the purpose of producing this material, and when it becomes possible for the average man who desires to build to buy either concrete lumber, which is waterproof and imperishable, or the ordinary lumber at or about the same price, it is needless to say that a new and vastly developing market for cement will be produced. What the development of the railroads in permanent road beds may require in the way of cement is still in the future, but to-day concrete for bridges, culverts, embankment walls and practically all the ordinary masonry of a railroad is the recognized material of construction.

New construction of cement plants in 1911 was limited, promoters finding little encouragement for their schemes. Nine new plants reported their first commercial output during the year as follows, this promotion dating to prior periods:

Golden State Portland Cement Co., Oro Grande, Cal.; Piedmont

Portland Cement Company, Portland, Ga.; Lehigh Portland Cement Company, Mason City, Iowa; Chanute Cement Company, Chanute, Kansas; Tidewater Portland Cement Company, Union Bridge, Md.; Michigan Portland Cement Company, Chelsea, Michigan; Knickerbocker Portland Cement Company, Hudson, N. Y.; Clinchfield Portland Cement Company, Kingport, Tenn.; and the Inland Portland Cement Company, Metaline Falls, Wash.

It is of some encouragement to the American manufacturer to realize that Germany with a much smaller area than that of our country and with what might be called established permanent construction of stone and brick as distinguished from lumber, produced over 30,000,000 barrels of cement last year. Of this, about 4,500,000 barrels were exported thus pointing the finger to a remedy adopted in that country and in England for overproduction and consequent low prices.

An interesting side-light on the industry from a world-wide standpoint is contained in the annual report of the Green Island Cement Company of Hong Kong, China, where reference is made to the fact that "the competition continues unabated but has somewhat changed in form. European shipments to the Far East have decreased and in the main can be ignored. The establishment of new factories in near-by territory is what interferes with us most." This condition is one that seems to have been affecting the industry throughout the world during the year 1911.

During 1911 considerable progress was made by the Joint Committee on Concrete and Reinforced Concrete in the preparation of their report governing methods of construction, specifications, etc. This Committee, composed of representatives of the American Society of Civil Engineers, the American Society for Testing Materials, the Association of Amercian Portland Cement Manufacturers, and the American Railway Engineering & Maintenance of Way Association, held many meetings and the result of their deliberations, as shown in the reports, will tend greatly to standardize concrete construction.

Slight changes were made in the Standard Specification for Portland Cement of the American Society for Testing Materials, and the Government, through a board of army engineers and a department board representing all the departments of the Government, issued during the fall a standard specification regulating all Government work. This specification has been approved by the President and has been recently sent out. It differs in a few immaterial details from the Standard Specification for Portland Cement of the American Society for Testing Materials and by its adoption many conflicting methods of cement testing have been reconciled.

CHROMIUM

In the United States chromite, the mineral mined as ore, was produced only in California. There are deposits in Pennsylvania, Maryland, and North Carolina; from Wyoming shipments were made in 1908 and 1909. The deposits in this country are either located where there is no cheap transportation or are of such a disseminated nature that the necessary mechanical concentration is too expensive per ton of product to yield a profit when competing with foreign ore at $10 to $20 per ton. Most of this ore is used in the manufacture of chemicals, but no ferrochrome is produced in the U. S. As a lower grade ore in the form of lumps and fines can be used for furnace bottoms, most of the domestic ore goes for that purpose. As this material lasts nearly indefinitely in the furnace, the domestic production suffices to supply this demand.

"The Distribution and Utilization of Chromium Ore" is the subject of a paper which appeared in the *Bulletin* of the Imperial Institute, Vol. VIII, Nos. 3 and 4, 1910. This paper describes the distribution of chrome ore in Europe, Asia, America, and Australasia. It also contains some notes on the preparation of chrome ores, the utilization of chromium and the production of chrome alloys. A complete discussion of the sources, uses and occurrence of chromite was published by the U. S. Geological Survey.[1]

The production for 1910 was 450 and for 1911, 120 long tons. The values are given as about $13.40 and $13.55 at the mines, though locally at furnaces small quantities are often sold at $25 per ton. The imports into the U. S. are mostly from New Caledonia, Greece, Turkey and Japan. The ore is duty free. Other chromium compounds imported are potassium bichromate and chromic acid, duty 2 1/4 cents per lb.

In 1909, 406,790 lb. of potassium bichromate and chromate were imported, valued at $19,569; also 9850 lb. of chromic acid valued at $1635. The import price of foreign ores is based on the grade which is mostly 50 per cent. chromic oxide, or less. Fifty per cent. ore brought about $15 in 1909, and $16 in 1910 in large lots. The 50 per cent. ore from New Caledonia, brought from $13.60 to $15.75 per long ton ex ship.

[1] E. C. Harder, Chromic Iron Ore, Mineral Resources for 1908, Part 2.

STATISTICS OF CHROME ORE IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production. (a)			Imports.			Consumption.	
	Long Tons.	Value.	Value per ton.	Long Tons.	Value.	Value. per Ton.	Long Tons.	Value.
1898	*Nil.*	.....	.....	16,304	$272,234	$16.70	16,304	$272,234
1899	*Nil.*	.....	.....	15,793	284,825	18.03	15,793	284,825
1900	140	$1,400	$10.00	17,542	305,001	17.39	17,682	306,401
1901	130	1,950	15.00	20,112	363,108	18.05	20,242	365,058
1902	315	4,725	15.00	39,570	582,597	14.73	39,885	587,322
1903	150	2,250	15.00	22,931	302,025	13.13	23,081	304,275
1904	123	1,845	15.00	24,227	348,527	14.38	24,350	350,372
1905	40	600	15.00	54,434	725,301	13.32	54,874	725,901
1906	317	2,859	9.00	43,441	557,594	12.84	43,758	560,453
1907	335	5,620	20.00	41,999	491,925	11.71	42,333	498,605
1908	280	5,600	20.00	27,876	345,960	12.40	28,156	351,560
1909	(b)598	8,300	13.88	39,820	460,758	11.57	40,418	469,058
1910	450	6,075	13.50	38,579	415,768	10.77	39,029	421,843
1911	(b)120	1,629	13.55	37,529	407,958	10.85	37,649	409,587

(a) As reported by the California State Mining Bureau except for 1907, 1908 and 1910, for which years the statistics are compiled from our own reports. (b) U. S. Geological Survey.

THE PRINCIPAL SUPPLIES OF CHROME ORE. (a)
(In metric tons.)

	1901	1902	1903	1904	1905	1906	1907	1908	1909	1910	1911
Bosnia	505	270	147	279	186	320	310	500	310	320	250
Canada	1,156	817	3,184	5,512	7,781	7,936	6,528	6,554	2,470	279	200
Greece	4,580	11,680	8,478	15,430	8,900	11,530	11,730	4,350	9,600	7,000	(c)
India	....	....	260	3,654	2,751	4,445	18,597	4,821	9,398	1,765	(c)
New Caledonia (b)	17,451	10,281	21,437	42,197	51,374	57,367	25,371	46,890	40,000	40,000	82,806
New South Wales	2,523	454	1,982	403	53	15	30	*Nil.*	*Nil.*	*Nil.*	150
Rhodesia	....	....	....	....	....	3,308	7,273	12,118	37,024	40,000	47,500
Russia	22,169	19,655	16,421	26,575	27,051	16,969	25,528	10,950	(c)	(c)	(c)
United States	132	320	152	125	40	322	339	284	606	456	121

(a) From the official reports of the respective countries. No complete statistics are available for Turkey or Africa. (b) Exports, except 1909 and 1910. (c) Statistics not yet available.

COMPOSITION OF CHROMIUM ORES.[1]

	Chromium sesqui-oxide Cr_2O_3	Ferrous Oxide FeO	Alumina Al_2O_3	Silica SiO_2	Magnesia MgO	Lime CaO.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per ct.
United Kingdom, Shetland Islands	42.20	24.93	7.47	6.10	17.30	....
Hungary, Orsova	38.95	16.13	17.50	8.00	17.20	2.20
Norway, Drontheim	42.00	19.72	12.00	5.00	21.38	....
Russia, Urals	55.75	21.56	3.37	5.37	13.85	0.60
Ekaterinburg	49.49	23.27	6.77	7.07	13.40	....
Orenburg	53.00	24.92	8.05	3.05	10.98	....
Asia Minor, Broussa	56.80	12.06	14.00	1.45	15.00	0.70
	60.14	15.70	6.34	1.08	16.40	....
Ceylon, Bambarabotuwa Dist.... Central Provinces	46.58	19.90	2.86	trace	5.63	....
British North Borneo, Malliwalli Island	51.21	18.90	9.58	1.04	16.46	....
United States, Shasta Co., California	42.45	15.00	16.75	7.50	16.50	0.90
San Luis Obispo Co., California (crude ore)	43.70	14.00	15.96	7.96	16.49	0.66
(concentrate)	52.86	15.45	11.59	3.00	16.26	0.76
North Carolina, Corundum Hill	57.20	25.68	7.82	2.80	5.22	....
Pennsylvania, Chester	41.55	62.02	.....	1.25	.....	....
Canada, Coleraine (crude ore)	45.95	22.50	8.90	7.68	4.90	0.12
Black Lake (concentrate)	50.35	17.77	9.90	3.10	16.02	1.80
Bolton	45.90	35.68	3.20	.....	15.03	....
Newfoundland, Port-au-Port Bay	49.23	17.21	7.50	6.51	18.66	....
Africa, Transvaal	37.03	23.95	17.23	7.63	9.94	2.00
Rhodesia	46.36	18.66	13.18	4.58	13.64	....
New Caledonia (cargo sample)	54.50	17.70	11.00	3.10	8.00	1.50

[1] Imperial Institute, Vol. VIII, No. 4.

Pure chromite contains 68 per cent. chromic oxide, the balance being ferrous oxide, but as most ores contain other minerals, the mined product seldom runs much more than 50 per cent. In some ores the iron and alumina in other minerals prevents concentration to obtain a high-grade product. Low-grade material can however be utilized for chemical purposes and some new processes are said to be commercially profitable on lower-grade ores in Sweden. The table gives some typical analyses.

One recent development in this age of trusts is the formation of the Chrome Co., Ltd., with a capital of £190,000. It has acquired the Tiebaghi mines in New Caledonia, formerly the property of the Societé Anonyme de Chrome, and also has arrangements with the Rhodesia Chrome Mines, Ltd., for marketing its output. Two of the most important producers are therefore acting together.

Africa.—Chromite is found on the De Kroon, 20 miles west of Pretoria. This is the ore which has been stated as containing traces of platinum and gold, besides about 35 per cent. chromium oxide. It is also one of those that is difficult to concentrate. In the eastern portion of the Transvaal in the Lydenburg district, outcrops of basic rocks are associated with chromite. Most of this material is of lower grade than the 50 per cent., which would appear to be the grade required in the bichromate industry. The belt of the country rock is about 1 1/2 miles wide and contains in places much hypersthene. Chromite has been located on a number of farms northwestward in the Rustenburg district, where a well-defined zone of deposits occur along a line some 28 miles in length and where again the chromite is found on the edges of basic igneous rocks. The ore consists in the Transvaal of black, lustrous, fine grained, granular chromite with a gradation into basic rock with only scattered grains. An additional analysis to the ones given in the table is as follows:

	Per cent.
Cr_2O_3	35.30
FeO	19.6
Fe_2O_3	2.10
MgO	13.45
Al_2O_3	13.70
SiO_2	12.70
CaO	2.05
MnO_2	.80
Phosphoric acid	.15
Combined water	.08
	99.93
Moisture in ore as received	.11

In Natal, specimens have come from a farm at Tugela Rand near Krantz Kop.

During recent years Rhodesia has steadily increased its production.

The Rhodesian chromite deposits were described in a *Bulletin* of the Imperial Institute, Vol. V, 1907, page 136.

An important deposit of chromite occurs at Selukwe and is being actively worked. The output has increased greatly within late years. During 1910, shipments amounting to 44,002 tons, valued at £98,132, were made. The total production to date is 94,044 tons, valued at £226,420. The output is rapidly increasing in 1911, and in the month of April alone, the value of the output was £17,625. The deposit at Selukwe is a huge segregated mass associated with serpentine, and being easily quarried, and in close proximity to the railroad, the cost of mining is low, and the supply is exceedingly large.

Chromite is known in many other parts of southern Rhodesia, at Victoria, Makwiro, Lomagundi, etc., and it is not unlikely that large bodies exist there. The chromic-oxide content of these ores varies, and is sometimes low, as, for example, at Makwiro. The deposits appear to be connected with the great intrusion of picrite which runs north and south through southern Rhodesia from the Zambesi River almost to the Limpopo, and is marginally altered to serpentine in several places. The chrome ore produced in 1911 amounted to 52,363 tons (2000 lb.). The ore is largely shipped via Beira.

Australia.—No chromite appears to have been produced besides that from New Caledonia, principally because transportation facilities have not developed any prospects. In New Zealand no chromite has been produced since 1902, when 175 tons were shipped from the Onatea deposits which have since proven of limited extent. In Tasmania, chromite is known to occur in the tin placer grounds and there are large deposits of a low-grade brown hematite containing about 3 per cent. chromium sesquioxide and very little sulphur or phosphorous. The chromiferous iron ore in western Australia has not been mined. It occurs at Comet Vale, North Coolgardie, and has a composition of ferric oxide 79 per cent., chromium sesquioxide 5.30 per cent., silica 3.14 per cent., phosphorous .08 per cent. and sulphur .12 per cent. In Queensland, material from the Elgalla mine has been used in the manufacture of silica bricks. In New South Wales, in 1889, the largest production was 5240 tons. During recent years, the chromite from the Gundagai district was found too low in grade. In 1911 150 tons of ore were mined nine miles from Barraba; the deposit contains 5000 tons.

New Caledonia.—The large proportion of the world's production of chromite comes from New Caledonia in the south of the island. The most friable ore occurs in ferruginous clay and is the ore more generally mined. The Tiebaghi mine yields about 4000 tons a month and up to 67 per cent. chromium. The only underground mine is the Lucky Hit

which furnished mineral containing between 30 and 40 per cent. sesquioxide. This is therefore concentrated. The ores from the Plum district on the Bay of Prony average about 30 per cent.

Canada.—The output of chromite which has been about 6000 to 8000 tons per year, was only 197 tons in 1911. Practically all of the production comes from eastern Quebec. The largest mine is in the Black Lake district between Collrain and Black Lake. At a depth of 80 ft. the ore body takes a sudden pitch and is largely increased in size. The ore in Ireland township occurs in a band disseminated through a width of 8 ft. and assays 23 per cent. sesquioxide.

Greece.—Chromite produced in Greece in 1910 was 7000 metric tons, containing 39 per cent. chromic acid. It comes from Pharsala. The value of 9450 metric tons sold was 400,660 francs. The stocks on hand at the beginning of 1911 were 2550 tons. In 1909, 9600 metric tons were sold for 430,655 francs (44.85 average) and 7400 metric tons were produced. The chromiferous ores of northern Greece and the islands of Eubea and Spyros have a proximate analysis as follows: iron 47 to 50 per cent. Cr_2O_3 2.5 to 3; alumina 9 to 10 per cent; silica 7 to 11 per cent.

India.—There was a fall in the production of chromite from 9250 tons in 1909 to 1737 tons in 1910. None was produced in Mysore and the production in Baluchistan fell from 4325 tons to the amount mentioned. Low prices, such as $12.75 c. i. f. London, did not permit any substantial profit. The rush for mining leases in Baluchistan during recent years, although it resulted in the grant of 76 leases in 1909, did not lead to any development. The localities in Baluchistan are chiefly at Quetta-Pishin and Zhob districts, where the ore occurs in masses in serpentine. In Mysore, the chief locality is Kaadkola where 30 to 50 per cent. ore occurs in narrow veins or lenses.

In the Bombay Presidency, 60 miles from Ratnagiri, large outcrops are found, associated with serpentine. Some of the chrome outcrops are stated to be 1000 ft. long and 300 ft. wide. Specimens run 34 per cent. Cr_2O_3.

Newfoundland.—The only deposits of chromite in Newfoundland are those at Buff Head on the north side of Port-au-Port Bay on the western coast. These deposits are in diorite which is traversed by thick beds of serpentine. While some ore has been shipped as mined, a large body of low-grade concentrating material is known to exist and has been prospected. A mill is erected on the ground. It has been demonstrated that 55 per cent. concentrates can be readily produced. Transportation facilities are good, and a large tonnage can be placed in sight with little expense. The property has not been worked for several years.

Turkey.—The chrome ore mines east of Anamur near the port of

Mersina, will probably report 2000 tons for the year 1911. Most of it is shipped, after a rough sorting, to Germany or France. It runs 51 or 53 per cent. chromic oxide and is valued $19.50 per ton on board steamers. Another chrome mine at Adranos is operated by a Smyrna firm and its production will no doubt exceed 15,000 tons. This ore is hauled by wagons a distance of 90 kilometers to the seaport of Gheumlek. The selling price of this ore which runs 55 per cent. usually, is on a 50 per cent. basis, plus a payment of 62 cents per unit above. The mining costs are $1 per ton and the taxes $2.30 per ton, but a profit of $6 is estimated. Another chrome mine at Karlyer with a production of 2000 tons, is similar to this one. A number of developments of chrome ores were carried on during the year at Aleppo and near Anitab.

The most important of the chrome deposits are those at Daghardi near Broussa, the production being estimated at 15,000 tons. Irregular pockets of serpentine occur at this place, assaying up to 56 per cent. Cr_2O_3. The ore is hauled 70 kilometers by camel to the railroad, which transfers it to steamers. The cost per ton is stated as $7 to $9 on steamers.[1]

In the European part of Turkey, the deposits of chrome ore at Oracha, are estimated to produce 1000 tons, which are shipped to Austria.

Chrome ore exports from Turkey to the United States were valued $35,014 and $46,993 in 1910 and 1911.

In the United States furnace linings and chrome brick are the principal uses of the ore. The price of chrome brick f. o. b. Pittsburgh was $175 per thousand. Chrome ore is used in the manufacture of chromate and chromic acid. The American bichromate production sold at 7 3/4 cents per pound while the foreign product brought 10 1/2 cents per pound in 1910. Potassium bichromate is employed for the production of chrome yellow and chrome orange, both used in calico printing and in ceramic pigments. Chromium salts, such as sulphate or chrome-alum, are employed to some extent for tanning leather. A method for colorimetric determinations in tan liquors is given by J. T. Wood and D. J. Law.[2] Chrome steel is an important product made by the addition of ferro-chromium. It is extremely hard, tough and strong. It is used for crusher jaws, stamp shoes, and machinery parts subject to abrasion. Another steel containing about 2 per cent. chromium and a small amount of nickel, finds extensive application.

The form in which chromium is produced is generally in iron as ferro-chromium containing either 60 per cent. chromium and 6 to 8 per cent. carbon or only 1 to 2 per cent. carbon. Metallic chromium is also

[1] Leon Dominian, *Min. and Sci. Press*, Jan. 6, 1912.
[2] *Journal of Soc. of Chem. Ind.*, Apr. 15, 1911.

produced, generally by the Goldschmidt process, which consists of mixing pure chromium oxide with aluminum powder in the proper proportions and igniting the mixture in a magnesia crucible, a by-product being fused alumina which rises to the top.

Bradley Stoughton states that by the smelting of chromiferous ores of Cuba in the blast-furnace, followed by refining in the basic open hearth, a steel is made which contains about 10 per cent. of the original chromium present in the ores, together with the greater part of the nickel.

Hence, about 90 per cent. of the chromium is oxidized and slagged off in the basic open-hearth furnace. The steel companies making this material desire to conserve all of that element possible because of the high qualities which it contributes to the steel.

They are able to turn out a very superior product, by virtue of the nickel and chromium which these ores retain, and find a ready market for it in many lines where steel having a combination of high strength and good ductility is demanded; thus, automobile steels and important parts of locomotives and railroad rolling stock are made of this material, which will have properties somewhat as follows:

Tensile strength	83,400 lb. per sq. in.
Elastic limit	62,500 lb. per sq. in.
Elongation in 2 in	8.67 per cent.
Reduction of area	49.53 per cent.

After different heat treatment, the steel may have properties somewhat as follows:

Tensile strength	162,500 lb. per sq. in.
Elastic limit	152,000 lb. per sq. in.
Elongation in 2 in	16.0 per cent.
Reduction of area	56.0 per cent.

or else:

Tensile strength	226,000 lb. per sq. in.
Elastic limit	190,000 lb. per sq. in.
Elongation in 2 in	8 per cent.
Reduction of area	32 per cent.

This steel is used for many different purposes, such as crankshafts, axles, hatchets, axes and other tools, as well as springs. It is now being largely advocated for screw rods and it is said to be an excellent material for wood screws and other screws, and it can be worked with great ease. The composition of some ferro-chromium alloys of commerce is shown in the following table:

Chromium	67.000	71.980	69.600	63.586
Iron	24.380	22.610	28.452	35.596
Carbon	8.050	4.789	1.560	0.650
Silicon	0.490	0.550	0.350	0.140
Sulphur	0.007	0.061	0.038	0.028
Phosphorus	0.005	0.008	trace	trace

Ferro-chromium is a tin-white, hard, brittle alloy with a crystalline tructure, and is now produced on a commercial scale in the electric urnace. The energy consumed amounts to about 10.6 h. p.-hours per ilogram of alloy obtained. The electric furnace method has the ad-antage of producing an alloy containing a higher percentage of hromium and a lower percentage of carbon than can be produced by he blast or crucible furnace methods formerly in use.

Metallurgy of Chromium

By Walter S. Landis

During the past year but little new information of interest to the hromium industry has found its way into print. A new chromate plant as been put into operation, the Yorktown Chemical Works, at Yorktown, Virginia. The company has not seen fit to give out information con-erning its products or methods, but it is safe to say that they are con-erting an imported chromite into chromate by the usual fusion process.

The preparation of the pure oxide, from chromite, $FeCr_2O_4$, is carried ut as follows: Pure chromite is crushed fine, mixed with sodium car-onate and heated to just below the fusing point in a reverberatory fur-ace similar to the LeBlanc soda furnace. It is necessary to have a ighly oxidizing atmosphere in the furnace. A sodium chromate is produced according to the reaction.

$$2FeCr_2O_4 + 7O + 4Na_2CO_3 = Fe_2O_3 + 4Na_2CrO_4 + 4CO_2.$$

The sodium chromate is leached out by water, filtered, converted into dichromate by the addition of sulphuric acid, and the resulting sodium sulphate removed by crystallization. The sodium dichromate is then reduced to chromic oxide by various reducing agents, sulphur being a common one. The reaction in this case is

$$Na_2Cr_2O_7 + S = Na_2SO_4 + Cr_2O_3.$$

This reaction takes place at low temperatures in iron pans. After leach-ing out the sodium sulphate the chromic oxide is dried and ready for use.

The resistance to corrosion which metallic chromium offers, together with its hardness, has always made this metal an attractive study for the electro-plater. Chemical plants of many kinds have searched for a hard, chemically resistant plating for covering various parts of apparatus to protect them from corrosion, and if a suitable plating process could be devised for depositing a firm adherent plating of chromium it would go a long way toward solving this difficulty. Many plating methods are recommended in "The Production of Chromium and Its Compounds by

the Aid of the Electric Current," by LeBlanc (Translated by J. W Richards), but many of them do not give the results claimed by th discoverers. J. Voisin in *Revue de Metallurgie*, VII, 1137 (1910) claim to have plated 70 grm. of chromium out of a bath of chromium acetat made by the Moissan method, except that bichromate was used in plac of chromic acid in its preparation. A current of 7 amperes and 10 volt was used, and the coating was stated to be particularly adherent.

The resistance of chromium-iron alloys to corrosion by nitric aci was made the subject of an exhaustive study by Monartz (*Metallurgi* VIII, 162, *et seq.* (1911)). He prepared 21 iron-chromium alloys by th alumino-thermic method, using 10 per cent. less aluminium than theoret ically required to combine with the oxygen present. In brief his result were: All iron-chromium alloys containing less than 70 per cent. chrom ium and free from carbon could be drilled and turned. The alloys con taining between 1 and 4 per cent. chromium showed a decreasing resist ance to attack by dilute nitric acid and an increasing resistanc toward concentrated nitric acid with increase of chromium content In the 4 to 14 per cent. alloys the resistance to attack of dilute aci increased rapidly with the chromium content. In the 14 to 20 per cent chromium alloys the resistance increased almost proportional to th chromium content. The 20 to 40 per cent. series showed but littl difference in attack from pure chromium; they dissolve readily in dilut nitric acid containing 5 per cent. sodium chloride. The 40 to 100 pe cent. chromium alloys showed practically no attack by concentrate nitric acid, or by boiling dilute acid containing sodium chloride. Th 65 per cent. alloy showed the least attack of any of the series. With th good mechanical properties of many of these alloys and their gres resistance to chemical action they seem to open a field of usefulness i the chemical industry that will be supplied by no other metal or alloy

A new bearing metal called "Chromax" is said by the patentee t possess some remarkable properties. It is a copper-nickel-chromiu alloy said to be acid proof, non-corroded by sea water, and of high mel ing-point. With a tensile strength of 79,000 lb. per square inch and 3. per cent. elongation it is remarkably strong and hard, and is said to ru well in bearings at a pressure of 20,000 lb. per square inch.

The great strength and toughness of the chrome bronzes has gaine for them a reputation second only to the manganese bronzes. The gres difficulty has been to produce them in the ordinary brass foundry. Suc a bronze containing 58 copper, 41 zinc and 1 chromium has shown a ultimate strength of 82,000 lb. per square inch, an elastic limit of 41,00 lb. per square inch and an elongation of 18–20 per cent. A U. S. Pater (No. 964,122) granted to Walter Rubel for a method of producing suc

a bronze calls for the heating of chromous chloride with zinc, or better still, with zinc and a little aluminium; the result is a zinc-chromium alloy which may be added to the copper, and zinc chloride.

The use of chromium salts in the ceramic industry as an under-glaze color for producing purplish pinks has been exhaustively studied by A. S. Watts and his results published in the "Transactions of the American Ceramic Society," XIII, 301 (1911). The best color was obtained from a mixture of 83.62 Al_2O_3, 9.34 Cr_2O_3, 7.04 B_2O_3 when fired at cones 18–22.

A rather extraordinary property of chromium salts was discovered by P. Koenig. In "Chemiker Centralblatt" for 1911, page 498, he states that he found that the application of small quantities of chromium compounds, especially chromous acetate and chromite stimulated germination and growth of plants and favorably influenced the yield. In larger quantities a toxic effect was noted, particularly in the case of chromates.

COAL AND COKE

By Floyd W. Parsons

The industry during the year 1911 was remarkable rather for what did not occur, than because of any new features developed. No important departures in the science of mining or in mining appliances have been made, nor has the country been visited by such an extended series of holocausts as caused a veritable reign of terror in several previous years. That it has met with many troubles and disappointments cannot be denied, but it has acquired confidence in meeting and overcoming these and has gathered new strength.

The year witnessed further consolidations of corporate interests in which the coal industry figured prominently. The economies possible to effect by such consolidation are fully appreciated by capital and the ensuing years will no doubt bring forth larger and stronger combinations along these lines. An entirely new development, in this direction, has been the organization of a National Coal Operator's Association, only yet in its infancy.

In production, the year was a good one, since the output nearly equaled that of 1910, but this, unfortunately, does not bespeak an entirely satisfactory business, because of the low coal prices that prevailed. It is true also that general industrial conditions were all below normal, and the heavy tonnages, thrown on an already weakened market, have produced a condition bordering on demoralization. With the productive capacity of our mines far in excess of the average normal consumption, such conditions will continue to prevail until some system of regulation is inaugurated, and such problems as these have made necessary the operators' organization.

The export shipments for the year show an unprecedented increase which is, however, hardly normal because largely due to the labor troubles that occurred in British Columbia. The United States is forging to the front as a fuel exporter and this may eventually become an outlet for our overproduction.

Labor troubles during 1911 amounted to practically nothing, all attention being directed to the renewal of present agreements which expire April 1, 1912.

A number of states have enacted new mine laws, the recent Pennsylvania bituminous legislation having attracted the greatest amount of

attention. This latter law was approved June 9 and has occasioned much unfavorable comment. The code fills 136 pages, and is one of the most complete mining enactments on record. Its principal fault is its obscurity. The electrical provisions of the new law have particularly met with censure.

Operators in practically all fields having had a rather hard struggle for some years past; there has not been such great desire on the part of smaller individuals to open new mines in the different states. The curse of the industry for some time has been overproduction, and until the majority of the mines in this country are so controlled by large and powerful companies, that production can be regulated better to meet the demands, but little improvement can occur.

In the Pittsburg district the demand was lighter than in the previous year, but with ample car supply and increased capacity, the production was only a few per cent. less than in 1910, with the result that prices were lower.

The coke production throughout the United States showed a general decrease of about 13 per cent. This was due to decreased consumptive demand, and was partly caused by the lessened output of pig iron.

There is no doubt but that one of the chief questions in the minds of coal operators to-day is the problem of caring for and dealing with employees. One of the largest operators in this country expresses his own opinion, and that of many of his colleagues in the following words: "Capital and labor must recognize that their interests are mutual. When treating with union labor, capital should insist that the union officials be honorable and intelligent men, in order to deal honorably and intelligently with them. Where labor is not organized, capital should get closer to the men and deal with them, as is already done by a large number of companies, in the way it would like to be treated. Capital should do everything possible to disarm labor, whether union or non-union, of the belief that it is its desire to make every dollar possible from its toil, regardless of its welfare."

Another problem that looms large before all of those who are interested in coal mining is the question of securing a fair and sufficient compensation law. It is to be hoped that Congress will recommend to the several states a law which will provide for a man and educate his family in case of accident or death. Any legislation which means individual liability for the different companies, to be recovered through the courts of our country, will not only be injurious to both operator and miner, but is certain to prove a boon for "shyster" lawyers, commonly and properly termed "ambulance chasers." A law of this sort only pays compensation to the workman where it can be proved beyond

doubt that it was not through his own negligence that the accident happened.

In case of a coal-mine explosion, it often occurs that a hundred or more men are killed, and very few companies in this country would be able to pay the required indemnity that would occur under the kind of liability acts that have been advocated in some states. The only sort of compensation laws that can operate satisfactorily in their application to coal mining are such acts as will require a company to pay a fixed amount to afflicted families as long as they are dependent upon the income of the injured or deceased employee.

PRODUCTION OF COAL IN THE UNITED STATES.

(In tons of 2000 lb.)

Bituminous.	1910.			1911.		
	Short Tons.	Value at Mines.		Short Tons.	Value at Mines.	
		Total.	Per Ton.		Total.	Per Ton.
Alabama	16,139,228	$19,528,466	1.21	15,040,267	$18,499,528	$1.23
Arkansas	1,200,000	1,896,000	1.58	1,660,000	2,623,000	1.58
California and Idaho	20,000	70,000	3.50	5,000	17,500	3.50
Colorado	12,104,887	16,946,842	1.40	10,075,861	14,207,000	1.41
Georgia	280,000	369,600	1.32	210,000	294,000	1.40
Illinois	48,717,853	52,615,280	1.08	51,900,000	57,090,000	1.10
Indiana	18,125,244	19,575,264	1.08	15,710,000	17,124,000	1.09
Iowa	7,660,000	12,945,400	1.69	(c) 7,574,010	12,875,817	1.70
Kansas	5,750,000	8,625,000	1.50	6,400,000	9,600,000	1.50
Kentucky	14,720,011	14,720,011	1.00	13,170,227	13,038,525	.99
Maryland	4,972,027	5,767,551	1.16	4,615,000	5,307,250	1.15
Michigan	1,620,000	3,916,000	1.80	1,560,042	28,080,756	1.80
Missouri	3,000,000	5,250,000	1.75	3,910,000	6,843,000	1.75
Montana	3,050,000	6,100,000	2.00	2,913,397	5,835,062	2.02
New Mexico	3,616,665	5,424,997	1.50	3,375,000	5,163,750	1.53
North Dakota	385,882	655,999	1.70	(a) 502,628	829,362	1.65
Ohio	34,424,951	36,146,200	1.05	31,500,000	31,657,500	1.05
Oklahoma	3,400,000	7,310,000	2.15	2,800,000	5,880,000	2.10
Oregon	80,000	220,000	2.75	40,000	112,000	2.80
Pennsylvania	144,365,816	155,915,081	1.08	142,237,605	153,616,613	1.08
Tennessee	6,750,000	7,020,000	1.04	6,450,000	6,770,000	1.05
Utah	2,526,093	4,092,271	1.62	2,501,472	4,102,414	1.64
Texas	(d) 1,750,000	2,905,000	1.66	1,870,000	3,141,600	1.68
Virginia	5,700,000	5,301,000	0.93	6,500,000	6,045,000	.93
Washington	3,510,207	8,775,518	2.50	3,550,000	9,052,500	2.55
West Virginia	59,690,300	54,318,173	0.91	60,000,000	54,000,000	0.90
Wyoming	7,469,452	14,938,904	2.01	6,750,000	13,770,000	2.04
Alaska and Nevada	10,000	40,000	4.00	10,000	40,000	4.00
Total Bituminous	411,038,616	$471,388,554	$1.14	411,830,509	$485,666,177	$1.18
Anthracite.						
Colorado	70,586	197,641	2.80	65,500	$14,340	$2.80
New Mexico	8,000	24,000	3.00	(b) 5,000	15,000	3.00
Pennsylvania	80,310,720	160,621,440	2.00	90,419,856	174,852,843	1.93
Total Anthracite	80,389,306	$160,843,081	2.00	90,490,356	$174,882,183	$1.93
Total coal (Short tons)	491,427,922	632,231,638	1.28	502,320,865	660,548,360	1.32
Total coal (Metric tons)	445,816,040	.....	1.40	455,720,550	.....	.....

(a) As reported by the U. S. Geological Survey. (b) Estimated. (c) For fiscal year ending June 30. (d) Does not include 979, 232 tons lignite, valued at $941,700.

PRODUCTION OF COKE IN THE UNITED STATES.
(In tons of 2000 lb.)

	1910.			1911.		
	Short Tons.	Value.		Short Tons.	Value.	
		Total.	Per Ton.		Total.	Per Ton.
Alabama	3,449,023	$9,762,245	$2.82	3,172,102	$8,945,328	$2.82
Colorado	1,190,901	3,620,339	3.04	946,284	2,876,703	3.04
Georgia and North Carolina	62,000	217,000	3.50	60,000	210,000	3.50
Illinois	390,000	1,755,000	4.50	675,000	3,037,500	4.50
Kansas	10,000	36,000	3.60	10,000	36,000	3.60
Kentucky	58,700	129,140	2.20	75,000	165,000	2.20
Missouri	5,000	18,000	3.12	10,000	31,200	3.12
Montana	58,200	325,920	5.60	60,000	336,000	5.60
New Mexico	510,000	1,540,200	3.02	450,000	1,359,000	3.02
Ohio	260,000	767,000	2.95	275,000	811,250	2.95
Oklahoma	30,000	117,600	3.92	45,000	176,400	3.92
Pennsylvania	22,875,000	45,978,750	2.01	20,130,000	40,461,300	2.01
Tennessee	240,000	669,600	2.79	500,000	1,395,000	2.79
Utah	146,064	449,877	3.08	212,368	654,093	3.08
Virginia	1,310,000	2,882,000	2.20	1,099,000	2,417,800	2.20
Washington	40,000	220,000	5.50	65,000	357,500	5.50
West Virginia	3,803,881	7,265,413	1.91	3,105,000	5,930,550	1.91
Other States (b)	1,890,000	7,560,000	4.00	2,460,000	9,840,000	4.00
Total	36,094,769	$83,314,084	$2.30	33,349,754	$79,040,624	$2.37

(a) Estimated. (b) Includes output of by-product coke for Massachusetts, Maryland, Minnesota, New York, Michigan, Wisconsin. (c) Fiscal year ending June 30.

During the year 1910, 2834 persons were killed in the coal mines of this country. This averages 3.91 persons per thousand men employed, and 5.65 persons per million tons of coal mined. During the year 1911, 2517 people were killed in our mines, which figures out 3.74 persons for every 1000 employees. In 1910 about 725,000 men worked in our coal mines; in 1911 only 670,000 men were so employed.

Of all the men killed in 1911, 1073, or 42.63 per cent. of the fatalities was caused by falls of roof. Fourteen per cent. of the deaths was due to mine cars and haulage motors. Seven per cent. of the fatalities was caused by gas explosions, and another 7 per cent. was due to coal-dust explosions.

Review of Coal Mining by States

Alabama.—The production for the year 1910 was 16,139,228 tons, but it did not go so high in the present year, the chief mine inspector estimating the total at 15,040,267 tons.

The labor supply was fair all year, with the exception of a temporary shortage toward the latter part of the summer when mines which had been closed down were started up and other mines put in more regular operation. The year was free from strikes or other disturbances of like nature.

The production of coal was low during the spring and early summer as the market conditions were not as good as during that period of 1910, for during that year, strikes in the coal fields of Illinois and Oklahoma helped conditions greatly. Also furnace requirements were not as large this year as in 1910.

In regard to the labor supply, it should be said that as far as actual numbers on the rolls are concerned, there was little or no difficulty. A sufficient number of men were enrolled to operate the plants, but many of these are inefficient workmen. Mining is rendered hazardous by the employment of a shifting force of men, and profits are curtailed when the daily output is liable to be reduced by the unforeseen absence of some of the men.

The quantity of coal made into coke during 1911 showed a decrease of about 8 per cent. from the coke production of the preceding year.

Colorado (By James Dalrymple).[1]—The coal business in Colorado for the year 1911 was not so lucrative as in 1910, although the output has been normal and satisfactory, all things considered.

The larger production of 1910 was due to the demand from markets which were affected by the miners on strike in Illinois, Kansas, Arkansas and Oklahoma; the needed supply being drawn from the Colorado fields. Then, too, the adverse financial condition and business lethargy of the entire country had considerable bearing on the coal industry. The disturbed industrial condition in Boulder and Weld Counties where lignite is mined continued and was only settled in the first part of 1912. Otherwise coal production was steady and found a brisk market. Prices have advanced from 25 to 50 cents per ton on domestic coals.

Operators had to cope with the usual shortage of cars in the last four months of the year, which retards the meeting of the market's demands. The working force for 1911, which was 13,813, showed a reduction of 955 men, as compared with 1910, when 14,768 men were employed. There was a decrease in the tonnage of coal produced during 1911 of 2,029,026 tons.

The death rate was 6 per 1000, which compares favorably with other years in Colorado. The non-fatal accidents numbered 248, as compared with 146 in 1910. Surface fatal accidents numbered six, and underground fatal accidents totaled 82. A dust explosion, augmented by powder, occurred at the Cokedale mine, owned by the Carbon Coal & Coke Co., of Las Animas County, on February 9, 1911, in which seventeen lives were lost. This was the only catastrophe during the year.

The mining laws of Colorado are faulty, but after the Governor appointed a commission to amend and improve the present laws, which

[1] Inspector of coal mines, Denver, Colo.

commission worked faithfully for five months, the Governor felt it incumbent upon him to veto the bill, as it was mutilated beyond all recognition.

SUMMARY OF THE COAL PRODUCTION OF COLORADO IN 1911

Number of mines in operation	158
Tons of lignite coal produced	1,676,975
Tons of semibituminous coal produced	761,526
Tons of bituminous coal produced	7,502,981
Tons of anthracite coal produced	64,379
Tons of unclassified coal produced, estimated	70,000
Total number of tons produced	10,075,861
Decrease in 1911	2,029,026
Total number of tons of coke produced	946,284
Total number of coke ovens in operation	2,764
Total number of employees in and about the mines	13,813.3
Total number of days worked	188.7

Idaho.—Fremont County, Horseshoe Basin, 30 miles east of St. Anthony and 12 miles from the terminus Driggs, of the Short Line railroad under construction, contains Laramie cretaceous shales and sandstones, identical with those at Rock Springs, Wyoming; the seams dip steeply, and are disturbed by faulting, but could be mined by tunnels.

On the Brown Bear property, several thousand feet of work have been done. An analysis of the coal shows 54 per cent. fixed carbon, 40 per cent. volatile matter and 6 per cent. ash and moisture. The claims are among withdrawals by the government for the purpose of classifying, and also lie in a forest reserve, which conditions may complicate the perfection of title, unless specifically provided for in the legislation that will properly dispose of them.

Illinois.—The fiscal year which ended June 30, 1911, showed only 53 counties producting coal in Illinois, as against 55 in the previous year. The number of mines and openings have also been reduced from 881 to 845. In the fiscal year, 184 mines were newly opened or reworked, but in the small time 217 were closed or abandoned. The total tonnage rose from 48,717,853 tons in 1910 to 50,165,099 tons in 1911. The price per ton rose from an average of $1.016 to $1.101, or $0.085. The gross selling value at the tipple totaled $56,064,494, or an increase of 11.7 per cent. There are now 316 motors used in Illinois, whereas in 1910 there were but 229, a significant increase of 38 per cent. Mining machines were used in 126 mines, whereas in 1910 there were only 114 mines in which they were to be found. The mining machines accordingly increased in number from 1289 to 1430. The number of tons undercut by machines were 19,998,259. In 1911 the number of tons hand-mined declined from 30,541,599 tons to 30,166,840 tons. In 1911 the miners employed numbered 39,912. During this same year more than 30,000 persons other than miners worked in the coal mines of the state. The average rate for hand-picked mining in the shipping mines per gross ton

was $0.627, an increase over the year before of 3 cents. Machine mining also cost 3.2 cents more, the price so advanced averaging $0.494 per gross ton. The number of kegs of powder used for coal blasting was 1,240,293.

While the production figures for the 12 months ending December, 1911, will show a considerable increase in output, the profits to operators have not been satisfactory.

Indiana (By Frank I. Pearce).—During the first month of the year 1911, the demand for coal became less brisk, and many of the mines that had been operating nearly full time previous to and for a considerable time after the settlement of the strike in Illinois, were unable to work more than half time during January, February and March.

With the approach of milder weather, market demands grew weaker, competition stronger and selling prices took a drop. As a result of this condition many mines that had been working about half time were unable to operate more than one or two days a week and a number were closed down indefinitely, or to make repairs.

This condition continued through April, May, June, July, August and the first half of September resulting in considerable suffering among hundreds of idle employees depending upon the mines for their livelihood, and in many instances a considerable outlay of money to the companies to keep their mines in repair. Even where this was done property in many instances suffered a certain depreciation in value, that is, a fixed loss.

The latter part of September the demand for coal began to improve somewhat and from this period until Dec. 15 the mines averaged about half time, and a number that were closed down earlier in the year resumed operations. Market conditions grew a little stronger and selling prices advanced.

The coal production of any state is governed by the demand, number of days worked and number of persons employed. The demand for coal has been poor for the year 1911 as compared with 1910 resulting in a greater loss of time, a larger number of employees being idle, and the production proportionately reduced.

The condition of the coal trade being somewhat similar to that of the year 1909 the production was about the same. There were a number of strikes during the year but most of these occurred at a time when the demand for coal was such as to not seriously affect the state's output.

Iowa (By L. E. Stamm).—During 1911 the conditions in the coal industry have been good, and the tonnage has reached the highest mark in production since coal mining first began in this state during 1840. In recent years the annual coal production of Iowa has increased steadily,

except in the years 1908 and 1910, when a slight falling-off in the production was noted, caused possibly by a suspension of the mines for a period of 30 days during each of those years, pending an agreement between the miners and operators relative to a wage agreement.

Under the laws of the State of Iowa, statistics relating to the production of coal in the State are gathered for the fiscal year ending June 30. These statistics gathered for the last fiscal year show a production of coal amounting to 7,574,010 tons. This is the largest tonnage ever produced in this state, and while these are figures for the fiscal year, it would be safe to say that they will approximate those for the calendar year. In the production of the tonnage above noted, 16,571 persons, both miners and others, were employed in and around the mines, and the work of mining was fairly good during most of the year.

Kansas (By Leon Besson).—The coal production for the year 1911 was approximately 6,000,000 tons. It was greater for the year 1911 than it was for the previous year, owing to the fact that in the year 1910 there was a long suspension of work, extending from April 1 to Sept. 22. On the other hand there have been few local labor troubles in the present year, and these not of any importance. An average of about 11,500 men were employed in the mines of the State. A few mine explosions occurred this year, causing the death of five shot-firers and two rescuers. I am of the opinion that the coal production will be steadily on the increase, for there is a tendency to go back to coal, as natural gas seems to be approaching exhaustion.

Kentucky.—The chief inspector of mines in Kentucky estimates that the production of coal during 1911 totaled 13,170,227 tons. The production of the respective districts was about as follows: western, 6,800,276; southeastern, 4,019,545; northeastern, 2,350,406. The total for the year 1910 was 14,720,011 tons. The returns for 1911, therefore, indicate a loss of about 1,500,000 in the western district, and a decrease of about 235,000 tons in the southeastern district, but a gain of about 312,000 in the northeastern region. Although the output was much less than for 1910, it exceeded the production for 1909 by more than 2,000,000 tons.

Notwithstanding the fact that 1911 proved rather a bad business year for the coal industry, the interest of investors in Kentucky has not flagged. The development of mines in the eastern coal field is increasing rapidly. Harlan County began shipping commercial coal in October, and is expected to become an important producer in 1912. Letcher County will become a commercial producer in 1912, and a large output is expected from it.

Maryland.—The coal output of Maryland for 1911 showed an appreciable falling-off. The total production for the year amounted to

4,615,000 tons. Maryland was particularly fortunate during the past year with respect to labor troubles, having experienced no inconvenience from this source whatever. It is estimated that the average days worked will be about 240, and the number of fatal accidents during the year will not total more than 15.

Michigan.—The tonnage produced in 1910 was below that of the preceding year. In 1911 it is estimated that the output totaled 1,560,042 tons. The coal area which lies in the drainage area of the Great Lakes covers at least 6500 square miles, but the seams average only 26 in.; none is mined under 24 in. Eight billion tons are estimated as still unmined. The coal is mostly absorbed in the local markets. Wood is largely used as fuel.

Missouri.—The year 1911 marked the beginning of the downward trend of coal operations in Missouri. There have been practically no new mines sunk in the state and several of the small older mines have been abandoned. As a whole there were few mines in the state that made expenses, as the demand for Missouri coal is limited, as well as its market. This is due principally to the fact that higher grade coals are produced in the other nearby states. The Missouri coal market has been lost chiefly to coal from the Carterville and Springfield districts in Illinois, and there seems to be no hope of the Missouri operators recovering this lost ground.

There is a feeling against the development of any more coal lands in Missouri on account of the low vein, and the high mining scale, and it is expected that from now on, each year will show a lower figure in the coal production for the state. No coal produced in the State of Missouri reaches the St. Louis market, although a fair tonnage from the central-western portion gets into Kansas City and St. Joseph. The market in the former city is limited on account of the use of natural gas.

Montana (By J. B. McDermott).—During the year 1911 Montana produced 2,913,397 tons of coal. Of this total production, Carbon County registered an output of 1,226,783 tons, while Cascade County showed a total of nearly 1,000,000 tons.

The new county of Musselshell, includes the rapidly growing camps, Roundup and Klein. This county is now the third largest producer in Montana.

There has been an apparent dullness in the Montana coal trade during the year just closed due to a heavy increase of capacity and consequent overproduction. At Stockett, in Cascade County, the Great Northern Ry. Co., coal department, has installed independent haulage engines for each of their new Nos. 5 and 6 mines. The Northern Pacific Ry. Co., at Red Lodge, in Carbon County, has increased its output from 1500 tons

per day to 5000. In Musselshell County, the Republic Coal Co. has increased the capacity of its No. 2 mine at Klein from 1500 tons, the record last year, to 2000, now the average daily run; this mine is now sufficiently developed to produce 3000 tons in 8 hours. In the same county, at Mine A, of the Roundup Coal Mining Company (subsidiary of the Chicago, Milwaukee & Puget Sound Ry. Co., at Roundup, the production is now 1500 tons per day. Mine B of the Davis Coal Co. is producing from 500 to 700 tons per day. Mines in the Bearcreek district are also doing considerable development in anticipation of a heavy future output.

The first dust explosion in the history of the Montana coal industry occurred on April 15, 1911, at the No. 2 mine of the Republic Coal Co., Klein, Musselshell County. No one was seriously hurt, although a number of men were scorched.

During the year just closed there have been 31 fatal and 50 non-fatal accidents in the Montana coal mines.

North Dakota (By J. A. Bliss).—During the year 1911 there were 98 coal mines in operation in North Dakota, producing about 395,000 tons of lignite. Over a thousand men were employed during the busy season which begins in October and closes in January. During the summer months the demand for lignite falls off, and as a result part of the mines are closed down. North Dakota mines are entirely free from explosive gases, and the list of fatalities and injuries has been low during the past year.

The demand for lignite is gradually increasing and several new mines have been opened up, while some of the larger operating mines have added to their equipments. Although the estimated amount of lignite within the state exceeds that of any other state in the Union, the demand for it is largely of a local nature, and because of its abundance the selling price is so low that the margin of profit is small.

The question of timbering the mines is a serious one, as dependence has to be placed on timber shipped in from Minnesota, largely, though some is obtained from the Western States.

The state legislature has wisely seen fit to lend active support to the furthering of its lignite industry, and all public institutions are required to use coal mined within the state. An appropriation has been set aside for the purpose of establishing and maintaining an experiment station. This has been located at Hebron, where a mine has been purchased by the state and an excellently equipped building erected. This station is under the direction of the State School of Mines at Grand Forks, and its efforts are largely directed toward perfecting the briquetting process and of making commercial use of the volatile gases which run high in all lignite.

The successful development of a process of briquetting on a commercial basis and of making use of the volatile gases for illumination, heat and power would create at once a demand for lignite and open a large field for investment.

The recent advent of several new branches of railroad in the western part of the state has made accessible a new portion of the coal fields, and a greater gain in production is anticipated for the ensuing year than heretofore.

Ohio (By J. A. Bownocker).—Coal mining in Ohio during 1911 was active, save in one or two localities were strikes existed. The lowest seam in the State is the Sharon or No. 1, or as it is known in the market, the "Massillon." The production of this field, which includes Stark and surrounding counties, has been falling off for several years owing in part to the working out of the best of the seam, and in part to the competition of natural gas. During the year, four mines in this coal were abandoned and only one new one opened, the latter being in Wayne County.

The "Wellstone" or No. 2 seam is of value in Jackson County only. It was discovered in 1872 in sinking a shaft for the No. 1 seam, and the quality was such that it was soon recognized as the finest in the State. New railroad lines extended the market and the output soon became very large; later Jackson was the leading county in the State, but for years the production has been decreasing and the end is near. During 1911 one mine was abandoned and no new ones opened.

Within the past few years a large block of coal has been discovered in the northwestern part of Guernsey County and extending as far south as Cambridge. A shaft has been sunk and a tipple is now being constructed. Since the shipping facilities are adequate, other mines will probably be opened in the near future and the area may become a large producer. Probably other unknown tracts of coal will be disclosed by drilling.

The one great field of the Pittsburg or No. 8 coal in Ohio is the Belmont, including nearly the whole of that county and parts of the surrounding ones. The year was prosperous with two new mines opened and none abandoned. This field has grown rapidly in importance during the past years and now ranks first in production. Further in the southern third of Belmont County it contains the largest block of undeveloped coal in Ohio. The output of this field will grow rapidly, if business conditions warrant, and Belmont County should become one large mining camp in another generation.

Work in the Federal Creek field was fairly active in 1911, one new mine being opened at Big Run, Athens County. Shipping facilities are poor.

Coke is made from this seam near Sharpsburg, but the output is small. It is the only place in Ohio where the industry survives.

By George Harrison.—According to reports from the various coal-mining districts of Ohio for the year 1911, the production was materially reduced as compared with that for the year 1910, which reached a total of 34,424,951 tons, the greatest amount ever recorded in the history of the state mining department.

In many districts work was unusually light during the year, while in others there was a suspension amounting to from 5 1/2 to 6 months. Several causes led to the large tonnage in 1910 that will not obtain for this year, notably the long strike in the State of Illinois and the suspensions in other districts; also, the increased demand for lake shipments.

The tonnage of the Hocking Valley district as a whole fell off about 15 per cent. as compared with 1910, and that of eastern Ohio showed a loss of about 10 per cent., or a total decrease of about 1,000,000 tons. The loss in Jackson County was about 40 per cent.; in Meigs County, the same; in Mahoning County, 10 per cent.; Columbiana County, 20 per cent.; and in Tuscarawas County the decrease in tonnage amounted to about 200,000 tons, this being due to the enforced idleness of from 5 1/2 to 6 months on account of the 1910–1911 strike.

The outlook for the year 1912 is not considered to be particularly promising, and it is not expected that any material improvement of conditions will be shown over the present year. One reason for this is the expiring wage contract, which will have to be considered and renewed. Nevertheless there seems to be no reason why 1912 should not be a reasonably prosperous year for both employer and employee.

Oklahoma (By Ed. Boyle).—The total coal production of Oklahoma for the year 1911 was 2,800,000 tons, which in an extremely small output for this field. If there was a market for the coal, the present openings and development should be producing, with steady work, 6,000,000 tons per year, but it seems that during a suspension of work in 1910, the operators lost most of their contracts to competitive fields and have been unable to regain them up to this time.

The system of mining in general use in Oklahoma is criminally wasteful. Leases have usually been secured with but one object in view, which is that of getting out the cheapest coal and leaving the balance to go to waste. It is now time that the people should become aroused over this unwarranted waste and demand mining methods and conditions which will conserve the resources of the state.

Mining conditions in this state differ from those found elsewhere, chiefly by virtue of the peculiar character of the roof. This contains a large proportion of limestone, which is easily slacked by the air current

and requires great care and an unusual amount of timber to keep it secure. Nearly all the mines of this field are generating large quantities of gas and have to contend with a particularly combustible form of dust, but thanks to competent management and supervision and a strict enforcement of the mining laws, there has been a great decrease in the number of fatal accidents, which is especially gratifying when compared with the number reported in other fields operating under similar conditions.

There are a number of seams of coal in Oklahoma, varying in thickness from 24 to 72 in., which are of exceedingly good quality. Some of this coal makes an excellent grade of coke and a large number of ovens have been built in this field. At one time large quantities of coke were supplied to the smelters of Mexico, but this trade has been lost on account of the prohibitive duty, since imposed by the Mexican government. It has been found unprofitable here to attempt to manufacture coke in competition with the Colorado field for furnishing this grade of fuel to smelters throughout the West.

Pennsylvania. Bituminous.—There has not been a large amount of development in the bituminous regions during the past year owing to the small profits in recent mining. The great advances of the past few years have been in the center of the state, where large developments have been made in Indiana and Armstrong Counties.

Travelers passing along the Allegheny Valley by the Pennsylvania R. R., formerly the Buffalo & Allegheny Valley Ry., have long wondered that the left bank of the stream was well developed, while the right bank showed few houses and no mines. The work of the "Shawmut" interests give assurance that the right bank will now be well occupied and even more prosperous than the other, because the mines will be in the hands of larger and more adequate financial concerns, having a steady market for their coal.

The ratification of the plan by which the Pennsylvania Coal and Coke Company was reorganized promises a new life for that company in Cambria and Clearfield Counties. The slack watering of the Allegheny River beyond the mouth of the Mahoning, if completed, will enable coal to be loaded in barges and shipped south from many mines, which have so far been tributary entirely to the Buffalo market. This overstocked mart should feel this relief, should the much desired plan be consummated.

The new Pennsylvania Bituminous Mine Law was approved June 9, 1911. It has occasioned no little unfavorable comment. Before it was finally framed a vigorous protest was made to all the new provisions, but at length the law was enacted and the whole Code fills 136

pages. It is one of the most complete mining codes on record. Its principal fault is its obscurity. There is evidence of the stress of conflict out of which it emerged, but it is probable that the conflict alone is not responsible for its illogical arrangement. Article XI is entirely new. It deals with the installation of electricity in the mines at length, devoting 24 pages to the subject. The uncertainty of the meaning of the law made some operators make the declaration that they would remove their fire bosses from the mines. This was perhaps not stated in real earnestness. But the Chief of the Department of Mines wrote a strong letter to them urging the retention of all the fire bosses employed. It remains to be seen whether a code specifying exactly what shall be or shall not be done is as good as one leaving larger power of control in the hands of the inspectors.

The production of the bituminous mines in Pennsylvania in 1911 totaled 142,237,605 short tons. There were 515 fatal accidents in the soft-coal mines, and 1102 non-fatal accidents. In the anthracite mines 699 men were killed, while there were 1121 non-fatal accidents.

Anthracite.—The production of anthracite in Pennsylvania during 1911 broke all previous records. The total output for the year, as reported by the chief inspector, James Roderick, was 90,952,341 short tons, valued at $175,763,214. The prosperity of the anthracite industry during the year was in marked contrast to the dullness that prevailed in the bituminous fields. In most of the bituminous states the prices were lower, whereas the average price of anthracite was about 4 cents per ton higher than in 1910. The greatest returns were received from the smaller sizes of coal.

There is no doubt but that the increase in the output of anthracite in 1911 was due to the possibility that there might be a strike of miners in the spring of 1912. It is also a fact that the severe winter caused an increase in the fuel demand of the public. In 1911, it is estimated that 40 per cent. of all the hard coal marketed was small sizes. At the present time there is a large recovery from culm piles, and there is no such waste as formerly occurred.

Tennessee.—During the year 1911 the coal production in this state amounted to 6,450,000 short tons, valued at a little more than $7,000,000. This output shows a decrease of about 10 per cent. from the production of the preceding year. The chief reason for the decline in the year's output of coal, as compared with 1910, is the fact that in the latter year the coal companies of Tennessee and other eastern States were called upon to supply the fuel deficiency caused by the strike of miners in Illinois and the Middle West. The decreased production in 1911, therefore, was simply a return to normal conditions.

The production of coke in 1911 amounted to 500,000 tons, which com pares with 669,600 tons produced in 1910.

The coal industry in Tennessee employed nearly 11,000 men, and th average annual production for each person was 600 tons. Of the tot output about 45 per cent. was mined by hand, 40 per cent. was short fro the solid, 15 per cent. was undercut by machines.

Texas (By Wm. B. Phillips).—The production of coal in Texas wa practically the same in 1911 as it was in 1910, totaling 1,870,000 ton in the former year. The north central coal field of Texas is of the car boniferous, while the Rio Grande coal field is of the Cretaceous or Tertiary period.

Most of the coal mined in Texas is for railroad purposes, not much of i being sold for domestic use. The total coal area is probably about 13,000 square miles, with an original supply of 8 billion tons. As a rule the seams of coal are comparatively thin, less than 30 in., and the usual ash content is 16 per cent., with sulphur 2 per cent. The heating power of these coals is, on the average, 11,000 b. t. u. and the weight per cubic foot 87.5 lb.

The production of lignite will be about 1,000,000 tons, or nearly the same as in 1910. Workable beds of lignite occur in 43 counties and the total area is about 60,000 square miles, or one-half of the total known lignite area in the United States. The deposits run in thickness up to more than 15 ft., and every variety of lignite is represented.

Utah (By J. E. Pettit).—The production of coal in Utah in 1911 was 2,501,472 short tons. This output was about the same as in 1910, and the value of the product was practically the same. All counties in the State showed a decreased production in 1911, with the exception of Emery County, which produced approximately 120,000 tons, as compared with 40,000 tons in 1910.

Coke was manufactured in only one county (Carbon) in Utah. In 1911, 212,368 tons of coke were produced, which total output showed an increase of 66,304 tons, as compared with the coke production of 1910. This increase was attributed to the fact that the Utah Fuel Co. secured some of their old contracts from the Amalgamated Copper Co. that were rescinded 2 years ago.

The amount of explosives used was: Black powder, 398,285 lb. giant powder, 301,792 lb., which includes Monobel, Bental No. 2 and other permissible powders, a total of 700,067 lb., or 1 lb. of powder for every 3.57 tons of coal mined. This shows a decrease in the amount of black powder used during the year, and a corresponding increase in the permissible powders.

The total number of men employed inside and outside in both the hydrocarbon and coal mines and coking plants was 3798, an increase of 499 over the preceding year. The average number of days worked at the various mines was 252, and the average amount of coal produced per man was 712 tons.

There were 84 accidents in and around the coal and hydrocarbon mines, of which 16 were fatal, 18 serious and 50 non-serious. There was one fatal accident which occurred outside of the mine. Seven wives were left widows and 21 children fatherless. The per cent. of fatal accidents for the year 1911, both inside and outside, was 4.21 per 1000.

The amount of coal mined for each life lost was 156,342. The causes of the fatal accidents were: falls of rock, 4; falls of coal, 8; runaway cars, 2; boiler explosion, 1; loaded trip of cars, 1.

Virginia.—Coal mining in Virginia has made important increases within the past generation. Although not a great coal state Virginia produced about 6,500,000 tons in 1911. The State is one of the earliest producers, the figures of the United States Geological Survey showing a small output as far back as 1822. In 1850 the production was 310,000 tons and at that time only two states stood above Virginia. While other states were forging ahead, the increase in Virginia was slow until about 1895, when 1,368,324 tons were produced. In 1900 the production was 2,393,754 tons; in 1905 it was 4,275,271 tons; and in 1910 it was 5,700,000 short tons, the greatest in the history of the state.

Washington.—The coal production of Washington for the year 1911 amounted to 3,550,000 tons, having a total value of $9,052,500. The coke production in 1911 amounted to 65,000 tons, having an average value of $5.50 per ton. The mining laws of Washington have been subjected to many changes recently, with the result that the producing end of the coal industry has been placed on a safer and more satisfactory basis. The present enactments include: (1) an 8-hour day; (2) the filing of maps for each mine; (3) the weekly measurement of air at certain planes in every mine; (4) the provision of magnetic or lead-locked safety lamps; (5) the placing of sign-boards indicating the direction of exits, traveling-ways, etc. The work of training mine-rescue corps at various points in the state has progressed rapidly. A mine-rescue station was established recently at Seattle, and mining men from all parts of the State have been coming to this station in order that they might become proficient in rescue work.

West Virginia.—The production of coal in West Virginia during 1911 amounted to 60,000,000 short tons, valued at $53,800,000. The most

unfortunate particular in connection with coal mining in West Virginia is that the total output of the state sells in the market for a lower figure than a like tonnage from any other field. This is true notwithstanding the fact that the West Virginia coals, as a rule, are superior to the product of any other state. The average price of West Virginia coal in 1911 was 90 cents a ton.

West Virginia, like other eastern and southern States, profited by the coal strike that occurred in Illinois and the Middle West in 1910. The decrease in production in 1911 was unusually small, considering the fact that in the preceding year the output of the State showed an increase of nearly 10,000,000 tons. In the neighborhood of 70,000 men were employed in the West Virginia field in 1911.

The average time worked amounted to 221 days. The average production per man was more than 900 tons for the year, and 4.48 tons for the working day. The returns to the Geological Survey show that practically 96 per cent. of the output was undercut either by hand or by machine. Less than 1,000,000 tons of coal were shot off the solid. The state may well be proud of such a record.

Wyoming.—The total output of coal in Wyoming during 1911 was 6,750,000 short tons, or a decrease of about 700,000 tons as compared with the year 1910, when the production was stimulated by the miners' strike in the Middle West. The quantity of coal undercut by machines in Wyoming in 1911 was 1,948,500 short tons, or nearly 30 per cent. of the total output. During the same year, about 28 per cent. of the total production was mined by hand, and the remainder of the coal produced was shot off the solid. The price of coal in 1911 was a little higher than the price that prevailed in 1910.

Coal in Foreign Countries

Australia.—Nearly 86 per cent. of all the coal output of Australia in 1911 was furnished by New South Wales. In this state, excluding lignite and seams of the Triassic age, it is computed that the main coal-bearing rocks extend over an area of 24,000 to 28,000 square miles around the seaport of Sydney. As yet, Tasmania supplies less than 70,000 tons per annum, although there are abundant seams of marketable coal in the country. These belong to the Carboniferous and Mesozoic periods, and vary from 20 in. to 12 ft. in thickness, while brown coal and lignite occur all along the North Coast. The state of Victoria has contributed coal annually since November, 1909, when this state purchased and equipped its own coal mine on the Powlett River. The output at the present time amounts to 300,000 tons per annum.

COAL PRODUCTION IN THE CHIEF COUNTRIES OF THE WORLD.
(In metric tons.)

Countries.	1906.	1907.	1908.	1909.	1910.	1911.
Asia:						
China	.....	10,450,000	11,970,000	12,840,000	14,591,000	.....
India	9,783,250	11,147,339	12,865,408	11,870,077	12,002,416	.....
Japan	12,500,000	13,716,488	13,942,000	14,019,626	14,794,208	16,020,000
Australasia:						
New South Wales	7,748,384	7,850,000	7,992,300	8,132,196	8,304,284	8,250,000
New Zealand	1,600,000	1,831,009	1,904,276	1,941,827	2,232,520	2,106,000
Other Australia	870,000	900,000	870,000	1,183,875	1,710,930	1,740,030
Europe:						
Austria Hungary (c)	37,612,000	40,112,530	40,760,870	39,842,749	38,006,840	40,116,743
Belgium	23,610,740	23,705,190	23,678,150	23,561,125	23,127,230	23,112,062
France	34,313,645	36,753,627	37,622,556	37,971,858	38,570,473	.....
Germany (c)	193,533,259	205,542,688	215,071,345	217,322,270	221,986,376	234,259,061
Italy	300,000	453,137	421,906	395,600	(e)400,000	510,029
Russia	16,990,000	21,207,500	22,943,794	24,083,000	24,572,403	.....
Spain (c)	3,284,576	3,250,000	3,871,480	3,520,000	3,550,000	.....
Sweden	265,000	305,000	300,000	250,000	(e)210,700	.....
United Kingdom	251,050,809	267,828,276	261,506,379	263,774,822	264,505,207	268,029,000
North America:						
Canada—						
Western	3,717,816	4,780,301	4,304,600	4,245,856	6,446,336	5,500,000
Eastern	6,196,360	5,730,660	6,599,866	5,200,777	6,564,930	6,602,000
Mexico	.....	.....	.....	919,338	2,450,231	.....
United States	375,397,204	435,483,938	369,895,861	402,981,688	445,816,040	455,720,550
South Africa (a)	(e)3,900,000	3,945,043	4,621,988	4,940,192	5,500,219	.....
Other countries (e)	5,500,000	3,475,780	4,106,000	5,000,000	(e)7,000,000	.....
Totals	988,173,043	1,098,468,506	1,045,248,779	1,083,996,876	1,143,739,902	.....

(a) Transvaal, Natal and Cape of Good Hope. (c) Includes lignite. (e) Estimated.

The coal output of New Zealand in 1911 was valued at more than $5,000,000. The most important collieries are situated near Westport, on the West Coast of Midland Island. More than one-third of the total output is brown coal or lignite, and many of the workings are open-cuts. The coal from the West Coast bituminous fields is of high class and is used by the Admiralty.

Austria.—The coke production for 1911 was 2,076,978 metric tons, as compared with 1,999,106 tons for the year 1910. The production of briquettes in 1911 was 347,597 metric tons, an increase of 13,379 tons for the year. Two-thirds of the briquettes were made from lignite. Coal is produced principally in Bohemia, Styria, Silesia, and Moravia. Of 967 properties only about 309 were worked. In the neighborhood of 120,608 men were employed. About 26 million tons of the production is brown coal about equal in value to the hard coal, viz., $27,400,000.

Belgium.—Coal mining is the most important mineral industry, and there are six different regions; the most productive is the Charleroi district, yielding about one-third of the total output. The average production per underground worker in 1911 was only 230 metric tons, due probably to the small size of the seams, which on an average are only about 2 ft. thick. In Belgium the average daily wage per underground worker is less than a dollar a day.

Canada.—On the Atlantic side of the continent, bituminous coal is

being mined from thick seams of true Carboniferous age at the Sydney (Cape Breton), Picton, Inverness and Cumberland coal fields, in Nova Scotia. The coal of the Pacific Coast, generally bituminous, belongs to the Cretaceous age, and is derived from collieries at Nanaimo, Extension, and Comox in Vancouver Island. The thick seams of bituminous coal, in the vicinity of the Crow's Nest Pass, are now being worked on an extensive scale, and a large quantity of the coal mined is converted into coke for use in the smelting industry in British Columbia. All these coals are of Cretaceous age.

In British Columbia most of the large producers of coal were idle practically two-thirds of the year 1911. The production in this province, therefore, was only about 40 per cent. of the output for 1910. In 1911, the output from Vancouver Island totaled 1,775,000 tons, and had an increased value over that of 1910 of nearly $500,000. At Hosmer another level has been established along the outcrop of the coal seam. At Corbin an enormous deposit of coal has been opened at the surface, and preparation has been made for working this deposit opencast, like a quarry. In the Nicola district, the Nicola Valley Coal & Coke Co. has built a new tipple and other plants, and has located important extensions of its coal deposits. One of the most important discoveries of the year was a new field of anthracite, located at the headwaters of the Skeena River, about 120 miles north of the Town of Hazelton, British Columbia. The coal production of Nova Scotia for 1911 amounted to 4,597,631 tons.

China.—The Hungay mines, on the coast of Tonkin, produce anthracite. This coal is mixed with 20 per cent. Japanese coal and briquetted for steamer use at Canton. The Kaiping Basin coal is mined by an Anglo-Belgian Syndicate at Tongshau and Luisi in the Chikli. There are three shafts turning out about 1,300,000 ton per year. This is a coking coal. The Sai-shui coal is mined near Canton. The mines of the provinces Kwaiytung, Kwauzsi, Kweichow and Yuman are small, and worked by natives without foreign capital or supervision, and the product is inferior. In South China the need is not felt for coal, as industries on any scale are lacking, and the mines are isolated.

Germany.—Deposits of brown coal are abundant in North Germany. The deposit of *Vorgebirge* near Cologne in the Rhine Province is a large, continuous bed extending about 25 km. (15 1/2 miles) from north to south with an average width of 6 km. (5 3/4 miles). A preparation called *Kaumacit* has attracted some interest. It is brown coal rendered transportable and imperishable by a process of dry distillation, reducing the coal to from 35 to 50 per cent. *Kaumacit*. The process yields an additional 3 per cent. of tar, 17 to 26 lb. of ammonia, and 2500 cubic meters of gas, per load of 10 tons.

There are three principal coal-mining districts in Prussia: (1) The Lower Rhine and Westphalian Basin, by far the most important; (2) Silesia, and especially Upper Silesia; (3) the Rhenish district in the neighborhood of Saarbrucken and Aix-la-Chapelle. Most of the coal is derived from seams of the Carboniferous age; near Hanover there are extensive workings in the Wealden beds.

The fuel production of the German Empire in 1911 was as follows in metric tons: Coal, 160,742,272; lignite, 73,516,789; coke, 25,405,108; coal briquettes, 4,990,988; lignite briquettes, 16,836,679. During the year the value per ton dropped somewhat, as the excess of production had to be absorbed in the export trade. The number of men engaged in the coal industry in Prussia was 696,211.

Great Britain.—The British coal industry was affected by labor troubles, especially in South Wales, and by the recent railway strike. Yet in spite of these difficulties, the various coal-fields are producing up to their maximum output except in Yorkshire, Nottingham, and a mine or two in South Wales. It has been reported during the year that Yorkshire is underlaid by a vast coal-field which was previously unknown and that in the course of several years 40,000 miners would be given employment in this region. But such is not the case. On the contrary, signs are not lacking to indicate that the remarkable development begun in South Yorkshire in 1900 has already reached its climax. This fact is due to the geological conditions that make shaft-sinking extremely expensive; the coal measures being overlaid by Permian and Triassic formations which contain quicksands and large volumes of water.

At Thome colliery as much as $1,250,000 has been spent the past year in sinking two shafts to a depth of 600 ft. each, owing to the fact that it was necessary to pump out 9000 gallons of water per minute. The ultimate depth of these shafts is to be 2760 ft. At the Halfield main colliery, near Doncaster, the "Francois" system of injecting cement into the strata as a preliminary to shaft-sinking was tried for the first time in Great Britain.

During 1911, six new colliery companies started shaft-sinking in the Midland coal-field, each mine being designed for an output of at least 1,000,000 tons per year, and altogether there are now 12 shafts being sunk in this locality. The increase of production in the Midlands has overtaken and exceeded the export trade with the result that a number of the older mines closed down when the newer and better equipped operations dumped their surplus output on the market. The erection of large central power plants and the consolidation of various groups of collieries in this region have both been in evidence during 1911.

In the Lancashire coal-field, at Astley Green, the deepest colliery shaft in Great Britain will have a depth of 3360 ft. when completed.

The coal mines bill for 1911, although a non-party measure, has been discussed even more than the Eight Hour act of 1908, which latter is now giving satisfaction. The chief discussion has centered around the use of electricity in gaseous mines, the abolition of naked lights, employment of female laborers on the surface, the provision of baths and wash houses and the rescue and ambulance equipment. The bill was not opposed in regard to its provisions for insuring the safety of mines and miners, but because of the increased cost which will devolve on some of the older collieries as a result of its being passed.

The output of coal in 1910 was 264,337,786 tons, and for 1911 totaled 268,029,000 tons. The number of workers employed in the mines has passed all previous records, and shows an increase over 1910. In 1851 nineteen persons were killed for every 1,000,000 tons of coal mined in Great Britain; in 1910 the death rate was 6.37 per million tons, whereas for 1911 the rate has been reduced below 6, there having been no serious explosions.

Italy.—In this country the development of the deposits of fossil fuel, which mineral is stated to be abundant in the provinces of Arezzo, Pisa and Grosseto in Tuscany, is hindered by the cheapness of imported coal from the United Kingdom. The total output in 1911 was only 510,029 metric tons, of which all but 2500 tons were lignite. Most of the lignite came from Tuscany. A small tonnage of anthracite is mined in the provinces of Cagliari (Sardinia) and Turin.

Japan.—Coal occurs in 43 of the 49 Prefectures in Japan. It is mainly bituminous, and most of the seams belong to the Tertiary period. More than two-thirds of the total output of Japan is produced in the Island of Kyushu. In 1874 the output was less than a quarter of a million tons; in 1911 the production was more than 16,000,000 tons. A large part of the coal production of Japan goes to supply the Chinese market. Notwithstanding the fact that the mines of Japan are increasing their output each year, the demand for coal is really greater than the supply, so that all the fuel that is mined in Japan commands a good price. The Japanese collieries are equipped according to the best modern practice.

Peru.—All the different varieties of fuel exist in Peru, viz., peat, lignite, coal and anthracite. Lignite is found in the Tertiary rocks on the coast and at the summit of the Cordillera at Cajamorca. The true coal and anthracite are found in the Cretaceous rocks in various places, and a solid hydrocarbon which is neither coal nor anthracite occurs in veins, and is likewise worked and sold as mineral fuel. There are large

areas of coal in the department of Ancachs, in the Santa Valley, at Jatunhuasi, near Jauja, in the department of Junin, in the neighborhood of Cerro de Pasco, and in the departments of Huanuco, Cajainarca and Libertad. The bulk of the output is obtained from the province of Cerro de Pasco.

Philippine Islands.—Coal occurs in the Tertiary shales and sandstones on nearly every island, with the greatest development in the Visayas. The most promising coal-fields at present are situated in the provinces of Albay, Cebu, Tayabas, Sorsogon, Mindovo and Moro. Four coal seams have been found in Cebu having thicknesses of 2, 5, 10 and 13 ft. The output in 1911 amounted to 38,000 tons and was obtained principally from two mines on the island of Batan in the province of Albay.

Russia.—By-product coke ovens are being extensively introduced in south Russia. The Donetz basin produces anthracite but also mixed varieties of other coal. During the first nine months of 1911, 854,090,000 poods, compared with 701,170,000 poods in 1910, were produced and the industry is on the increase. The Ural coal district is next in importance but is too far inland to be of interest outside of Russia.

Coal in the Caucasus is not in sufficient quantities excepting in connection with the manganese deposits at Sharopan where its occurrence might save the manganese industry, by some considered ruined. There are also coal seams near Moscow and in Poland which are being worked. The visible coal reserves of Russia have been estimated at only 45,000,000,000 poods.

Next in importance to the Donetz Basin is Poland, with an output of over 5 2/3 million metric tons of true coal and brown coal. The Dombrowa Basin, in Poland, is a continuation of the great Silesian coal basin. Coal is abundant in Siberia, both east and west, and even along the line of the Trans-Siberian Ry., but the quality is poor. In the Island of Saghalien coal is worked by Russian convicts; the present output is small and is used by steamships.

Spain.—Imports of coal into Spain for the first 11 months of 1911 were 1,884,315 tons as compared with 1,828,108 tons for the same period in 1910. Coke imports for this period in 1911 were 293,841 tons as compared with 268,690 tons in 1910. Practically all imports are from Great Britain.

Spitzbergen.—Since 1904 the Arctic Coal Company, an American corporation, has been extracting coal on the west coast of Advent bay, Spitzbergen. Two beds are being exploited, one situated 590 ft. above the sea, 8 ft. thick, and the other at a height of 785 ft., from 4 1/4 to 5 1/4 ft. thick. The coal beds are intercalated in hard and compact

Miocene sandstones without any slatey layers. The seams are inclined about 4 degrees and undulate slightly, but are faulted. The coal is dry and of good quality. The operations have been principally preparatory, so that the production during the summer of 1910 was only 5000 to 6000 tons, and the mine will not be in proper going order until June, 1911. It is then proposed to export the coal to Norway, which is destitute of that mineral. The 150 workmen employed are mostly Norwegians and are paid from $1.60 to $1.84 per day. They now work all the year round, although the engineer in charge is only there from June to October.

Sweden.—All the Swedish coal obtained in 1911 was produced from the provinces of Malmohus and Kristianotad in the southern part of the kingdom. The seams, which are of Rhodanian age, are interstratified with beds of fireclay, and the two minerals are worked together. The thickness of the coal seams, including the partings of shale, varies from 3 to 5 ft.

Turkey.—Although coal is known to occur in nearly all the provinces of the empire, the only mines deserving mention at the present time are those at Eregli. Important deposits of lignite or brown coal exist in the region of Lebanon, and near Lampsacus on the east side of the Dardanelles. Coarse lignite has been found in several places near Sparta, Karaman, and in the Bulghar Dagh which may prove useful for smelting purposes. Coal deposits are known to exist in the Van district in the province of Erzeroum.

The Coal Markets in 1911

The general coal trade showed an increase of about 4 per cent. in 1911. The anthracite shipments increased nearly 8 per cent. over 1910. With the exception of the N. Y. Ont. and West., the railroads shared about equally in the increased shipments of hard coal.

SHIPMENTS OF ANTHRACITE.
(Tons of 2000 lb.)

	1909.		1910(*b*).		1911.	
	Tons.	Per ct.	Tons.	Per ct.	Tons.	Per ct.
Reading	11,920,757	19.2	12,445,733	19.2	13,265,758	19.0
Lehigh Valley	10,296,627	16.6	11,195,765	17.2	12,603,000	18.0
N. J. Central	7,938,370	12.8	8,519,135	13.1	9,218,802	13.2
Lackawanna	9,531,695	15.3	9,589,076	14.8	9,869,620	14.1
Del. & Hudson	6,136,946	9.9	6,578,356	10.1	7,206,731	10.3
Pennsylvania	5,966,543	9.6	6,250,976	9.6	6,494,733	9.3
Erie	7,461,121	12.3	7,554,198	11.7	8,800,179	12.6
N. Y. Ont. & West.	2,717,826	4.3	2,772,547	4.3	2,495,476	3.5
Del., Susq. & Sch.	(*a*)	.....	(*a*)	.....	(*a*)	.....
Total	61,969,885	100.0	64,905,786	100.0	69,954,299	100.0

(*a*) Shipments included in total of Lehigh Valley Railroad Company. (*b*) From *Coal Age*.

A complete discussion of the *coastwise trade*, anthracite and bituminous, rail and water, will be found in an article by J. J. Wolkins.[1]

Pittsburg (By B. E. V. Luty).—With ample car supply and increased capacity the production in the Pittsburg district was only a few per cent. less than in the previous year. Prices suffered throughout the year. In 1911 the $1.15 price was frequently shaded by as much as 10 cents per ton on important contracts, while occasionally odd lots forced on the market brought less than a dollar. Long-term scale contracts, based on the mining rate, while on the whole calling for low rates gave a fixed price equal to that of the last nine months of 1910. The average realized price on all the coal mined in the district fell between 5 cents and 10 cents below the average of 1910, which with a slight decrease in the total tonnage made the year an unsatisfactory one. The decrease in the Connellsville coke production, 20 per cent. from the record year 1906, was partly due to lessened output of pig iron, partly to development of other fields, and in part to the adoption of by-product ovens. The region shows no indication of ever adopting by-product practice, though it has been adopting the rectangular push oven, involving slightly cheaper operation.

MONTHLY AVERAGE PRICES FOR CONNELLSVILLE COKE FOR 1911.
(Per 2000 lb. at ovens.)

Month.	Furnace.	Foundry.	Month.	Furnace.	Foundry.
January	$2.60	$3.05	July	$1.65	$2.15
February	2.25	2.75	August	1.65	2.15
March	2.00	2.60	September	1.60	2.15
April	1.80	2.40	October	1.55	2.10
May	1.70	2.25	November	1.45	2.00
June	1.65	2.20	December	1.50	2.00

CONNELLSVILLE REGIONS COKE PRODUCTION.

Year.	Short Tons.	Year.	Short Tons.
1906	17,245,975	1909	17,785,832
1907	19,400,327	1910	18,689,722
1908	11,133,173	1911	16,334,174

COASTWISE COAL SHIPMENTS.
(In tons of 2240 lb.)

	1910.			1911.		
	Anthracite.	Bituminous.	Total.	Anthracite.	Bituminous.	Total.
New York (a)	13,991,426	10,431,636	24,423,062	14,651,401	10,749,988	25,401,389
Philadelphia	1,980,830	4,691,882	6,672,712	2,197,750	4,856,626	7,054,376
Baltimore	271,122	3,903,152	4,174,274	257,025	4,002,809	4,259,834
Newport News	.....	3,163,798	3,163,798	.....	2,678,156	2,678,156
Norfolk	.....	3,644,561	3,644,561	.....	4,698,769	4,698,769
Total	16,243,378	25,835,029	42,078,407	17,106,176	26,986,348	44,092,524

(a) New York includes all the New York harbor shipping ports.

[1] *Coal Age*, Jan. 6, 1912.

Baltimore (By Leslie Rawles).—The year 1911 has been one of reduced sales and small profits for Baltimore operators. When the present year made its advent, the hopes of the coal trade in Baltimore ran high. Operators had visions of prolonged activity in the steel industry, but these hopes have not been realized to the extent that Baltimore operators had expected. During some of the months of the year, coal has moved freely, representing shipments made under contract by Baltimore operators. And it might be said that nearly all the coal shipped in the last few months was called for under contract, but under slightly reduced prices when compared with 1910. In this section, renewals of contracts take place around Apr. 1. When this time arrived, some of the consumers came in under prices quoted by operators. Others did not, preferring to hold out and take their chances of getting the product cheaper by spot purchases. In January the spot business, which plays so prominent a part in the Baltimore market, was somewhat brisk, and again in February, operators reaped satisfactory profits from this business. There was an easing off in the demand for coal in March, and during the complete slump which followed, the spot business disappeared. Contract shipments also eased up. The early spring and summer months brought no improvement in the demand for coal. Such stagnant conditions prevailed until the advent of October, when there was a slight improvement, as evidenced by the freer movement on the railroads over which Baltimore operators ship their product. The latter part of October found the demand not quite so brisk, and, while considerable coal moved in November, it was mostly under contract. The spot business was lacking, and the market bordering on stagnation.

Cincinnati.—Prices as a whole were unsatisfactory and did not advance above the $1.50 mark for domestic grades excepting in rare instances. The prices in the Pomeroy Bend district, higher because of the differential in freight rates, reached the highest point at $1.75 to $1.85. The large tonnage of 34,424,951 in 1910 was partially caused by the long lay-off in Illinois and the increased demand for lake tonnage. The tonnage in 1911 was about 31,000,000.

Mississippi Valley (By E. J. Wallace).—With the close of 1911 the operators of Illinois have put the year down as one of the most unsuccessful in their history. At no time during the past year has the price of Illinois coal gone beyond normal, and for the greater portion of the year the actual selling price of coal was around and below the cost of production.

The principal reason for prices below and around the cost of production is that the strike of 1910 caused much business to be diverted to other fields, principally Indiana and West Virginia, and the Illinois

operators have been unable to recover this lost ground. Again, the production of Illinois coal is growing twice as fast as a logical market for it is being developed, and to-day Illinois can produce twice as much coal as there is any natural demand for in its territory.

The cost of coal in the Franklin County, Illinois field has been increased considerably on account of installations for safeguarding the work. The "wild-cat" sinking of mines has been put a stop to by the law which requires all shafts and air shafts to be of concrete.

In the central and northern Illinois districts there has been no great development. It is figured that Illinois coal, during the year 1911, brought between 3 and 5 cents per ton less than in the previous year. Comparatively no coke is produced in Illinois excepting at two or three points.

Missouri.—Practically no new mines were sunk and several of the small older mines have been abandoned. Few mines made expenses, as the demand for the coal is limited, chiefly on account of the higher grade coals produced in the surrounding states. The Missouri coal market has been lost chiefly to coal from the Carterville and Springfield districts in Illinois.

Arkansas.—The coal is gradually growing in favor an account of its superior quality. However, the cost of production is so great that only by the aid of favorable railroad rates can this coal compete with the higher grade coals of other fields, but where these rates are practically the same, Arkansas coal is almost eliminated.

River Shipments at Pittsburg.—Figures compiled by the United States Engineers indicate a banner year in the river-coal business of the Pittsburg district.

The aggregate of the coal tonnage on the Monongahela River is arrived at by taking the coal shipped up through Lock No. 1, plus the amount shipped down through Lock No. 3 and to this sum is added the amount mined and shipped in Pools 1 and 2 on the Monongahela River. The river tonnage on the Ohio River is obtained by adding the amounts shipped, both up and down that river, passing through Davis Island Dam. Shipments exceeded those of 1910 by more than 500,000 tons.

According to the Monongahela River Consolidated Coal & Coke Co., the largest shipper, one-third of all the coal they mine is shipped to the Southern trade, one-third to the Pittsburg Harbor for local consumption and one-third by rail to the Lakes and the Northwest. All told, however, less than 10 per cent. of the coal shipped and mined on the Monongahela River is sent South.

Lake Tonnage.—According to the reports of the Hocking Valley

docks at Toledo, the docks handled 2,253,069 tons as compared with 2,077,614 tons in 1910 as follows: Hocking district, 662,388 tons as compared with 666,218 tons in 1910; Kanawha & Michigan district, 886,405 tons as compared with 885,562 tons in 1910; Chesapeake & Ohio district, 651,205 tons as compared with 492,101 tons in 1910; Kanawha & West Virginia district, no tonnage as compared with 12,239 tons in 1910; Coal & Coke district, 816 tons as compared with 13,449 tons in 1910; Norfolk & Western district, 50,781 tons as compared with 6367 tons in 1910; Wellston district, 1443 tons as compared with 1678 tons in 1910.

IMPORTS OF COAL AND COKE INTO THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1906.	1907.	1908.	1909.	1910.(c)	1911.
Canada	1,427,731	1,398,194	1,107,737	1,043,419	1,675,602	980,174
Great Britain	106,771	42,830	36,989	17,225	13,124	9,278
Australia	191,758	552,918	327,441	182,271	240,889	232,969
Japan	11,996	123,720	31,792	14,344	56,278	16,031
Other countries	6,251	8,356	340	5,079	(b) 19,576	356
Total coal	1,744,507	2,126,018	1,504,229	1,262,338	2,005,559	1,238,808
Coke	128,461	132,355	129,591	170,671	156,417	69,515

(a) Of the coal imported in 1909, there were 4709 tons classed as anthracite. Nearly all the imports were for the Pacific coast. The features of 1909 are found in the large falling off of imports from Australia and Japan. The unusual increase in the receipts of Australian and Japanese coal in 1907 was due to the expectation of a fuel famine in the West. The coke received is from British Columbia with the exception of a few thousand tons from Germany. (b) Includes Great Britain. (c) From *Eng. and Min. Journ.*

DESTINATION OF COAL EXPORTS. (a)
(In tons of 2240 lb.)

	1906.	1907.	1908.	1909.	1910.	1911.
Canada	7,533,346	9,843,315	9,252,943	9,782,574	10,531,085	14,108,567
Mexico	1,084,319	1,066,502	694,099	614,310	675,980	470,674
Cuba	689,833	804,310	690,867	723,594	858,776	1,053,703
Other West Indies	319,839	474,382	374,699	378,184	487,519	577,159
Europe	81,734	220,479	234,581	255,109	250,060	} 1,222,650
Other countries	212,748	737,760	605,988	782,786	1,002,446	
Total	9,921,819	13,146,748	11,853,177	12,536,557	13,805,866	17,432,753

(a) The European exports in 1909 were chiefly to Italy, that country receiving 156,920 tons. Other countries are chiefly the South American republics. The Canadian shipments were 78 per cent. in 1909, 76.3 per cent. in 1910 and 81 per cent. in 1911.

Exports and Imports.—The export trade of the United States is increasing more rapidly than that of any other country. In ten years it has doubled in value, while Germany's trade increased 60 per cent. Bituminous coal imported is from British Columbia; Australia furnishes less than one-tenth the total.

COPPER

The United States is the greatest source of copper and in 1911 produced about 56 per cent. of the world's output. The American refineries produced in 1911[1] 1,431,938,338 lb. of all classes, excepting scrap, as compared with 1,452,122,120 lb. in 1910 and 1,438,751,056 lb. in 1909, as shown in the table.

PRODUCTION OF COPPER IN UNITED STATES ACCORDING TO CLASS.
(In pounds.)

Year.	Lake.	Electrolytic. (d)	Casting. (d)	Pig Copper. (a)	Total.
1905....	219,000,000	(c)760,000,000	46,000,000	33,495,000	(c)1,058,494,000
1906....	224,071,000	(c)860,000,000	52,000,000	29,098,000	(c)1,165,169,000
1907....	220,317,041	854,441,000	47,957,890	30,032,000	1,152,747,890
1908....	222,267,444	850,660,325	44,967,250	35,000,000	1,152,895,019
1909(e)..	226,602,134	1,101,518,458	67,471,446	43,159,018	1,438,751,056
1910(f)..	221,400,864	1,151,624,597	(g)55,673,196	46,903,463	(g)1,475,602,120
1911....	216,412,867	1,156,627,311	22,977,534	35,920,626	1,431,938,338

(a) Exported. (b) Estimated. (c) Partly estimated. (d) Included copper from scrap and junk. (e) The statistics for 1909 are officially communicated to us by the Copper Producers' Association, except that to its report of 34,123,446 lb. of casting copper we have added 33,348,000 lb. reported to us by the junk smelters. The term "Lake" copper is here used to designate all copper sold in the trade as such, regardless of the process by which it is refined. (f) Copper Producers' Association, through *Eng. and Min. Journ.*, May 6, 1911. (g) Includes 23,480,000 lb. from scrap.

The supply of crude copper received by the United States refineries during the year, from smelters is computed as follows: The smelteries of North America report in 1911, 1,337,853,657 lb. of copper in ore and scrap, as compared with 1,337,278,078 in 1910. Of these amounts, in 1911, 32,413,440 lb. were refined outside of the United States and in 1910, 33,855,800 lb., leaving amounts shown to which are added the blister copper imported from countries outside of North America.

SMELTERS' PRODUCTION.
(In pounds.)

Source.	1910.	1911.
North American ore	1,284,339,246	1,284,932,019
Foreign ore	41,976,733	34,392,091
Scrap	10,962,099	18,529,547
Totals	1,337,278,078	1,337,853,657
To foreign refiners	33,855,800	32,413,440
To American refiners	1,303,422,278	1,305,440,217
Blister copper imported	146,185,104	146,422,851
Total crude copper	1,449,607,382	1,451,863,068

[1] *E. and M. J.*, May 25, 1912.

The figures in this year show that the refineries did not produce as much copper as the smelteries account for. It is probable, however, that but little crude copper has accumulated in stock at the end of 1911. Another fact that accounts for some of the difference is that a small part of the smelters production is turned out in the form of copper sulphate (see below) and it is estimated that this may account for 6,000,000 lb. of fine copper in 1911. The stock locked up at the refineries and in the form of blister copper made from their own by-products may account for as much as 20,000,000 lb. in one year but the statistics do not take this indefinite quantity into consideration. The amount of copper tied up in the electrolytic refinery itself is of interest in this connection, and it is estimated that for the present production of the refineries of the United States (viz., 1,156,000,000 per annum) and exclusive of the refined stocks, there are probably 120,000,000 lb. of copper on hand. This amount, that is in the form of electrolyte, anodes, etc., may act as a balance wheel to the extent that a 10,000,000 lb. refinery could jump its output by more than 10 per cent. for a few months without increasing the rate of input. Fine conclusions as to the accumulation of crude copper either in transit or stock, therefore, cannot be given, from these data. The following table is based on the crude copper smelter returns, distributed over the separate states.

COPPER PRODUCTION OF THE UNITED STATES.
(In pounds.)

	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Alaska	4,703,609	8,700,000	6,610,000	4,394,887	4,057,142	5,008,171	19,412,000
Arizona	222,866,020	263,200,000	256,866,761	290,167,795	292,042,829	299,606,971	300,578,816
California	16,697,486	24,421,000	34,398,823	36,890,353	53,357,451	45,793,894	36,806,762
Colorado	9,854,174	9,565,000	13,344,118	813,896,689	10,487,940	10,127,012	8,474,848
Idaho	6,500,005	9,493,000	11,471,101	8,749,559	7,770,010	6,216,461	3,745,210
Michigan	218,999,759	[224,071,000	220,317,041	222,267,444	227,247,998	221,400,864	216,412,867
Montana	319,179,880	299,850,000	226,290,873	252,558,330	313,838,203	286,242,403	271,963,769
Nevada	(a)	426,000	1,462,450	12,174,269	51,835,309	63,877,500	65,385,728
New Mexico	5,638,843	6,262,000	8,652,873	8,523,652	5,134,506	3,632,351	1,518,288
Utah	51,950,782	49,712,000	68,333,115	70,978,952	100,438,543	125,042,381	138,336,905
Wyoming	2,393,201	146,000	2,919,137	2,384,356	89,654	180,861	(c) 130,499
Southern States	14,907,982	18,821,000	22,408,696	20,822,368	22,837,962	18,195,450	19,656,971
Other States	1,550,000	3,379,000	6,166,098	4,387,836	3,746,895	925,664	1,433,708
Total	875,241,741	917,620,000	879,241,766	948,196,490	1,105,336,326	1,086,249,983	1,083,856,371

(a) Included in "Other States." (b) From *Eng. and Min. Journ.*, May 25, 1912. (c) U. S. Geol. S.

The refinery statistics given in the tables are those collected by the Copper Producers' Association. The smelters' reports are itemized as to origin of ore and it is the most accurate indication of the mine production available. The mine production in 1911 was doubtless larger than the smeltery production in some states, especially where new mines are producers.

Consumption.—The domestic deliveries totalled 709,611,605 lb. in

1911 as compared with 749,426,542 lb. in 1910. The deliveries for export were 754,902,233 lb., as compared with 722,431,494 lb. in 1910. For comparison, these may be taken as consumption, over such a period; the visible stocks of copper in the United States, however, had decreased from 122,030,195 lb. on January 1, 1911 to 89,454,695 on Jan. 1, 1912 and the consumption does not agree with these figures taken into consideration. To reconcile the figures of production and stocks the discrepancies already mentioned as between production and refinery production plus stocks must be overcome. As we have seen above, this is a difficult matter at present and we retain the table only as it has been a feature of previous MINERAL INDUSTRIES. The exports by the Department of Commerce and Labor, also do not cover precisely the same time as the Copper Producers' Association; we have, therefore, three different sources. The difference between the reports for exports for 1911 is about 18,000,000 lb. If, therefore, deliveries for export be taken as reported by the Copper Producers' Association the consumption becomes less than given in the table.

The following table shows the refinery statistics for the year by months. The next table gives the exports and imports of copper in the form of ore, matte, and regulus, and of ingots.

The table for the consumption of copper in the United States shows wherein deliveries differ from the consumption as computed. In 1910, the exports as given by the Department of Commerce and Labor, were corrected to overcome and explain a difference that existed between the figures as given in Vol. XIX of the MINERAL INDUSTRY and those of the Copper Producers' Association, and used as a basis for the consumption. The corrected figures are now included in the table.

Refinery Statistics.				Visible stocks.			
Month.	United States Production.	Deliveries domestic.	Deliveries for export.		United States.	Europe.	Total.
Year 1910.	1,452,122,120	749,426,542	722,431,494	I,1911	122,030,195	236,629,120	358,649,373
I, 1911....	115,696,591	42,078,557	53,208,739	II....	142,439,490	236,992,000	379,431,134
II.....	109,828,297	50,518,998	45,111,019	III...	156,637,770	233,385,600	390,023,009
III.....	130,532,080	66,080,789	59,081,127	IV...	162,007,934	223,014,400	385,022,434
IV.....	118,085,223	52,407,650	62,129,599	V....	165,555,908	212,284,800	377,840,708
V.....	126,962,544	64,543,963	61,978,557	VI...	165,995,932	202,540,800	368,536,732
VI.....	124,554,312	61,655,561	71,460,519	VII..	157,434,164	195,932,800	353,366,964
VII.....	112,167,934	56,982,582	74,880,658	VIII..	138,738,858	191,891,840	329,630,698
VIII.....	125,493,667	59,935,364	69,855,660	IX...	133,441,501	191,228,800	324,670,301
IX.....	115,588,950	57,311,584	50,824,011	X....	140,894,856	191,945,600	332,840,456
X.....	118,255,442	64,068,307	60,084,349	XI...	134,997,642	176,825,600	311,823,242
XI.....	111,876,601	68,039,776	67,049,279	XII..	111,785,188	164,281,600	276,066,788
XII.....	122,896,697	65,988,474	79,238,716	I,1912	89,454,695	158,323,200	247,777,895
Year, 1911	1,431,938,338	709,611,605	754,902,233				
I, 1912....	119,337,753	62,343,901	80,167,904				

Figures are in pounds of fine copper. U. S. production includes all copper refined in this country, both from domestic and imported material. Visible stocks are those reported on the first day of each month, as brought over from the preceding month. Stocks at Hamburg and Rotterdam are included in the visible stocks for Europe.

EXPORTS OF COPPER FROM THE UNITED STATES.(a)
Ore, matte and regulus stated in tons of 2240 lb. Ingots, etc., in pounds.

Country.	1906.	1907.	1908.	1909.	1910.	1911.
Ore, matte and regulus Exported to:						
United Kingdom	206	200	168	258	.....	.....
Germany	59	188	2	.....	.....	.....
Brit. North America	36,700	82,016	55,367	50,571	.....	.....
Mexico	10,600	16,737	7,060	8,534	.....	.....
Other countries	54	.....	552	520	.....	.....
Total	47,619	99,141	63,149	59,880	43,784	57,915
Ingots and scrap (b) Exported to:						
United Kingdom	55,097,670	81,409,441	117,810,314	156,511,113	98,030,213	108,061,603
Belgium	6,475,054	3,822,551	5,560,366	6,016,861	7,176,258	5,125,C04
France	80,703,723	93,075,145	115,690,381	99,003,962	116,193,850	135,038,893
Germany	96,629,040	107,607,390	137,453,392	138,213,290	175,861,028	190,428,008
Italy	19,777,296	21,192,908	25,512,267	26,386,069	34,110,237	38,216,773
Netherlands	151,650,293	156,652,270	195,562,619	204,378,211	221,764,806	230,693,649
Russia	9,523,992	4,341,386	4,657,077	3,519,216	6,848,311	15,601,688
Other Europe	25,260,807	26,221,024	39,433,674	41,661,979	42,203,861	9,254,363
Brit. North America	4,176,135	3,747,410	3,977,142	6,790,410	5,628,487	8,931,582
Mexico	263,319	362,411	35,895	46,287	.....	.....
China	4,932,128	10,003,592	13,735,899	.....	.....	.....
Other countries	262,561	493,873	2,447,101	319,328	499,492	(c) 45,201,645
Total	454,752,018	508,929,401	661,876,127	682,846,726	708,316,543	786,553,208

(a) The exports of ore, matte and regulus are reported as gross weight, the copper contents not being stated. (b) Includes bars and plates. (c) Includes Austria-Hungary, 44,200,202 lb.

IMPORTS OF COPPER INTO THE UNITED STATES. (a)
(In pounds.)

Country.	1906.	1907.	1908.	1909.	1910.	1911.
Ore and matte Imported from:						
Brit. North America	10,329,955	12,803,069	11,187,297	9,689,829	10,034,806	12,919,644
Mexico	31,690,058	32,467,418	15,903,692	23,914,040	22,731,184	16,684,071
South America	4,140,589	8,790,621	13,025,614	20,987,197	19,425,233	15,305,335
Other Countries	2,874,289	5,657,679	16,365,340	26,496,327	33,033,752	23,717,628
Total	49,034,891	59,718,787	56,481,943	81,087,393	85,224,975	68,626,778
Pig and scrap (b) Imported from:						
United Kingdom	22,549,321	25,706,852	5,434,435	26,527,574	18,649,727	9,004,461
France	3,202,168	606,662	168,506	490,191	.....	.....
Germany	5,303,712	6,814,338	1,451,370	1,045,647	.....	.....
Other Europe	5,649,689	5,616,261	13,359,117	27,379,175	25,411,393	28,042,257
Brit. North America	30,398,369	30,902,596	30,895,737	29,196,351	29,016,785	22,442,335
Mexico	85,595,359	76,741,532	43,742,993	76,119,724	84,008,907	97,115,574
Cuba	513,240	767,184	349,560	104,182	.....	.....
West Indies (c)	399,569	401,585	184,490	223,408	.....	.....
Japan	6,752,486	9,809,569	8,329,896	23,830,140	18,482,989	20,030,447
Other countries	16,194,477	35,534,688	58,308,040	55,797,329	83,640,995	89,345,686
Total	176,558,390	192,901,267	162,224,144	240,713,721	259,210,796	265,980,760

(a) The imports reported are the copper contents of ore matte and regulus. (b) Includes also bars, ingots and plates. (c) Includes Bermuda.

CONSUMPTION OF COPPER IN THE UNITED STATES. (a)

Year.	Production.	Stock Jan. 1.	Imports.	Supply.	Exports.	Stock Dec. 31.	Consumption.
1902...	636,796,381	209,587,698	161,551,040	1,007,935,119	376,298,726	162,935,439	468,700,954
1903...	708,375,228	162,935,439	167,161,720	1,038,472,387	312,822,627	230,111,792	495,537,968
1904...	817,715,005	230,111,792	182,292,205	1,230,119,002	555,638,552	208,376,672	466,103,778
1905...	875,241,741	208,376,672	210,724,685	1,294,343,098	548,772,403	132,587,496	612,983,199
1906...	917,620,000	132,587,496	225,593,281	1,275,800,777	467,839,041	139,385,400	668,576,336
1907...	1,152,747,890	9,000,000	5,000,000	1,166,747,890	508,929,401	120,000,000	537,818,489
1908...	1,152,895,019	120,000,000	.....	1,272,895,019	661,876,127	122,357,266	488,661,623
1909...	1,405,403,056	122,357,266	.....	1,527,760,322	682,846,726	141,766,111	703,147,485
1910...	1,452,122,120	141,766,111	.....	1,593,888,231	708,316,543	122,030,195	763,541,493
1911...	1,431,938,338	122,030,195	.....	1,533,968,533	786,553,208	89,454,695	677,960,630

(a) The statistics in the above table up to 1906 inclusive are computed in the old way, namely, on the basis of the production of blister copper and the imports of copper in all forms. The stock on hand at the beginning and end of the year includes not only refined copper, but also the crude copper in transit and in process of refining. The statistics since 1906 are computed on the new and more accurate method described in *Eng. and Min. Journ.*, July 25, 1908. Briefly, in this method the basis is production of refined copper, stock of copper in final marketable form and imports of refined copper. This change in method explains the erratic appearance of the figures for 1907 as compared with those of 1906.

According to the estimate of Aron Hirsch & Sohn, Halberstadt, Germany, the world's consumption of copper exceeded the production by 76,140 long tons in 1910 and by 153,010 tons in 1911; the diminution in visible stocks was 23,684 tons during 1910 and 43,082 tons during 1911. No account is taken of scrap or old metal. For 1911, this firm estimates a production of 869,370 long tons and a consumption of 986,300 tons, of which Europe consumed 640,000 tons, America 319,791, China 4500, Japan and Asia 20,000. The largest consumer was Germany with 239,000 tons.

The Metallgesclschaft and Metallbank of Frankfurt a. M. gives the 1910 production of copper for the world as 877,600 and the consumption as 894,900 metric tons.

The world's visible stock of refined copper has also decreased as have the stocks in the United States in 1911. In a general way, consumption of copper for the world was less than the production up to 1910 when the balance swung the other way. The European consumption reached a high point in 1911.

THE WORLD'S COPPER PRODUCTION.

(In metric tons.)

Country.	1902.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910. (h)	1911. (h)
Africa (a) Cape Co.	2,794	4,704	5,563	5,105	4,003	4,298	4,550	4,720	7,016	17,252
Africa (a) Namaqua	1,727	610	2,337	2,337	2,642	2,540	2,440	2,337		
Africa (a) Other	.....	.....	.....	.....	.....	.....	.....	8,128	8,433	.....
Argentina (a)	244	137	157	157	107	224	226	610	305	1,036
Australasia (a)	29,098	29,464	34,706	34,483	36,830	41,910	40,123	34,952	40,962	42,512
Austria-Hungary (a)	1,626	1,407	1,473	1,346	1,458	1,062	3,877	6,218	2,276	2,566
Bolivia (a)	2,032	2,032	2,032	2,032	2,540	21,035	2,540	2,032	2,540	1,829
Canada (d)	17,765	19,637	19,490	21,595	19,110	2,540	24,376	21,626	23,810	25,570
Chile (d)	27,066	29,923	31,025	29,126	25,829	28,863	42,097	42,726	38,346	33,088
Cuba (d)	.....	.....	.....	.....	1,384	1,388	2,966	3,006	3,538	3,753
Germany—total (a)	21,951	32,214	30,262	22,492	20,665	20,818	20,523	32,815	25,105	22,363
(Mansfeld) (a)	(19,050)	(19,810)	(19,578)	(19,878)	(18,085)	(17,343)	(18,000)	(19,015)	(20,275)	.....
Italy (a)	3,424	3,150	3,388	2,997	2,911	3,353	3,022	2,769	3,272	2,642
Japan (f)	29,034	31,861	33,187	35,944	36,963	40,183	41,399	42,987	50,703	52,303
Mexico—total (d)	36,357	46,040	51,759	65,449	61,615	57,491	38,190	57,230	62,504	61,884
(Boleo) (a)	(10,958)	(10,480)	(11,120)	(10,341)	(11,002)	(11,506)	(12,600)	(12,426)	(13,003)	.....
Newfoundland (a)	2,906	2,753	2,235	2,316	2,332	1,758	1,453	1,402	1,097	1,174
Norway (a)	4,638	6,010	5,502	6,406	6,218	7,122	9,337	9,226	10,592	9,576
Peru (c)	7,701	9,497	9,504	12,213	13,474	20,681	15,240	16,257	27,375	28,500
Russia (c)	8,817	9,232	9,835	9,515	9,296	15,930	17,718	18,035	22,670	25,980
Spain-Portugal (a)	50,587	50,536	47,788	45,527	50,109	50,470	53,425	53,023	51,080	52,880
Rio Tinto (a)	35,032	36,382	34,016	32,795	34,642	32,833	35,517	35,938	(34,114)	(35,100)
Tharsis (a)	6,817	6,421	5,710	4,415	4,816	4,206	4,500	4,425	(3,551)	(3,450)
Mason & Barry (a)	3,383	2,469	2,997	2,764	2,504	2,662	2,804	2,403	(3,003)	(2,972)
Sevilla (a)	1,570	1,123	1,351	1,300	2,073	2,337	2,196	1,849	(1,656)	(1,558)
Sweden (c)	178	776	533	1,385	1,209	1,577	2,808	2,032	2,032	2,032
Turkey (a)	1,118	1,422	965	711	432	1,270	1,068	813	610	1,016
United Kingdom (g)	490	545	501	727	762	677	588	442	508	508
United States (d)	293,053	312,631	400,998	397,069	416,343	398,930	430,399	501,372	492,720	491,634
Total	542,606	630,590	693,240	698,931	715,510	724,120	758,065	854,758	877,494	880,098

(a) As reported by Henry R. Merton & Co., Ltd., of London. (b) As reported by Henry R. Merton & Co., previous to 1905, subsequently as reported by the *Eng. and Min. Journ.* (c) As officially reported except for 1909, for which year the figure of Henry R. Merton & Co. is used. (d) As reported by the *Eng. and Min. Journ.* (e) As reported by Henry R. Merton & Co. for 1900–1902, as officially reported 1903–1907, as per Henry R. Merton & Co. for 1908 and 1909. (f) As officially reported. (g) As officially reported, 1900–1905; subsequently as per Henry R. Merton & Co. (h) Henry R. Merton & Co., through *Eng. and Min. Journ.*

WORLD'S PRODUCTION OF COPPER. (a)

Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.
1881...	166,065	183,093	1891...	280,138	308,862	1901...	529,508	583,517
1882...	184,620	203,550	1892...	309,113	340,808	1902...	542,606	597,951
1883...	202,697	223,481	1893...	310,704	342,562	1903...	630,590	694,910
1884...	223,884	246,840	1894...	330,075	363,920	1904...	693,240	764,758
1885...	229,315	252,828	1895...	339,994	374,856	1905...	698,931	770,221
1886...	220,669	243,295	1896...	384,493	423,917	1906...	715,510	788,492
1887...	226,492	249,716	1897...	412,818	455,147	1907...	724,120	798,205
1888...	262,285	281,179	1898...	441,282	486,529	1908...	758,065	835,623
1889...	265,516	292,741	1899...	476,194	525,021	1909...	854,758	942,408
1890...	274,065	302,166	1900...	491,435	541,561	1910...	877,494	966,998
						1911...	880,098	970,308

(a) The statistics for 1881–91 are as reported by Henry R. Merton & Co.; 1892–1910 as per *The Mineral Industry*. (b) Henry R. Merton & Co.

In the United States, a number of new producers have gradually been added, namely, Miami, Chino, Ray, Bonanza, Mason Valley and Braden, and are gradually increasing their production which will not be fully established until into the year 1912, if then. This expected capacity is estimated at 170,000,000 lb., annually. On the other hand, there were

during the year 1911, a number of producers that, for one reason or another, suspended operations. Among these were the Balaklala, Granby, Imperial, Butte and Ballaklava, and Shattuck-Arizona. Still other producers, as the United Verde and the Mammoth, operated at much reduced rate, and it is estimated that these curtailments amounted to 6,000,000 lb. per month.

An increase in the refiners' production occurred in 1910. The capacity of electrolytic refineries has now increased about 46,000,000 lb. as given in the following table:

ELECTROLYTIC COPPER REFINERIES OF THE UNITED STATES.

Works.	Location.	1910 Capacity. Pounds.	1911 Capacity. Pounds.
Nichols Copper Company	Laurel Hill, N. Y.	†330,000,000	†330,000,000
Raritan Copper Works	Perth Amboy, N. J.	*310,000,000	*320,000,000
Baltimore Copper Smelting and Rolling Company	Baltimore, Md.	*264,000,000	†288,000,000
American Smelting and Refining Company	Perth Amboy, N. J.	*180,000,000	†180,000,000
U. S. Metals Refining Company	Chrome, N. J.	*180,000,000	†180,000,000
Balbach Smelting and Refining Company	Newark, N. J.	*41,000,000	*48,000,000
Anaconda Copper Mining Company	Great Falls, Mont.	*65,000,000	*65,000,000
Tacoma Smelting Company	Tacoma, Wash.	†28,000,000	†28,000,000
Calumet & Hecla Mining Company	Buffalo, N. Y.	*50,000,000	*55,000,000
Total		1,448,000,000	1,494,000,000

* Official figures furnished by the respective companies. † Estimated from previous records.

During September, October, November and December, 1911, an unexpected shrinkage in the refinery production took place, when compared with the smelter returns. There may have been restraint placed upon some of the large refinery outputs but it appears likely that with the low stocks and with the promising demand that any surplus would have been taken care of in these months.

A statement of the production of the world for the years 1850 and 1909 by countries is given by James Douglas[1], showing that in the former year the production of 109,708,000 lb. was chiefly from Great Britain, Chile, and Russia. The present production all told (887,088 metric tons) for 1911, is about twenty times as great and the productions of all countries, excepting Great Britain and Austria, have increased since 1850. Great Britain then was credited with 27 per cent. and Austria with 6 per cent. of the total, whereas, now the former makes only .06 per cent. and the latter 1.14 per cent. of the total. An interesting statement comparing the relative consumption of copper and pig iron, and also the productions, is given by James Douglas.[2] Since 1896 to 1911, inclusive, the average number of tons of iron produced in the United States for every ton of

[1] *Min. and Sci. Press*, May 21, 1910; MIN. INDUSTRY, XIX, page 153.
[2] *E. and M. J.*, Vol. XCIII, No. 16, p. 776.

copper was 53 and the average number of tons of pig iron consumed for every ton of copper was 83. The corresponding ratios for the world's figures to 1911 are 78 and 79. The theory that the consumptions of these metals are parallel would appear to be supported. The resources of iron ores are greater to supply the demand than are those of copper ores to supply the demand for copper; as the lower grade copper ores are drawn upon more and more, the value of the red metal will depend upon the difficulty of supplying the demand from these sources.

The effect of the increase in production of the porphyry coppers, which are low-grade properties, but produce the metal cheaply on a large scale, was feared in 1911. The ore reserves, estimated mill capacities, and copper contents are given in the following table:

	Mill Capacity, Tons per Day.	Ore Reserves, Tons.	Grade Per Cent. Copper.
Chino	5,000	54,970,646	2.24
Inspiration	(a)5,000	50,000,000	2.0
Miami	3,000	18,232,000	2.58
Nevada Consolidated	8,000	40,853,371	1.66
Ray Consolidated	8,000	77,314,470	2.17
Utah	21,000	301,500,000	1.53

(a) Assumed.

The estimates of ore reserves made sometime in 1911 with corresponding assays, are probably as accurate as engineers can make them. The capacity of the plants with corresponding extraction expected is a difficult matter to predict accurately; the cost also depends on many uncertainties; the character of ores is not the same combination of gangue and mineral in all cases and a few per cent. in the extraction mean a difference in cost and waste. Many improvements have reduced costs in the case of plants like the Nevada Consolidated where increased tonnage lowered the concentrating cost per ton 25.7 per cent. The Utah concentrators are extracting 70 per cent. of the copper contents and yet these mills probably lost 40,000,000 lb. of copper in the year 1911. They are, however, working on the lowest-grade ores and getting as good an extraction as other mills working on higher-grade ores. One exception to this is the Miami where an extraction of 73 per cent. is claimed.

The outcome of the attempt to consolidate some of the Lake Superior Copper mines was a failure, although unquestionably many economies might have been effected, especially as this district is long past the prospecting stage. Profits are much dependent on a large scale of operations in this district but the officials were hampered by suits and legal complications that made consolidation unadvisable.

Copper Sulphate.—In 1909, the exports of copper sulphate from the

United States were 2,965,703 lb. worth $133,341. In 1911, the exports were 7,421,439 lb. worth $319,255.

Austria Hungary and Italy are the largest consumers. Prices in the United States were $4.50 per 100 lb. The production is estimated at 33,454,000 lb. in 1911 against 26,356,788 lb. in 1910 and 45,000,000 in 1909. This includes the sulphate made in refining base lead. Some copper refiners now avoid making this by-product.

Copper Markets in 1911

New York.[1]—The explanation of the apprehensions at the outset of the year in contrast with the hopefulness prevailing at the close, is to be found in a few simple factors. Producers exercised a commendable restraint; instead of forcing the output of their mines, it was, in most cases, decided that a curtailment would serve the industry best. The low level of the market stimulated consumption throughout the world. Copper replaced other metals for the most varied uses. The growth of consumption for miscellaneous purposes was so gradual as to evade observation from the standpoint of its effect upon prices. Stocks of pig copper which had accumulated at the refineries were gradually worked off. This reduction in the output of the refineries was expected much earlier than was actually the case, the delay being due to accumulations of furnace material, which had served as a working stock, and which were first of all consumed before the refineries allowed their tanks to stand idle. The market backed and filled for 10 months of the year within a range of not more than $10 per ton, due largely to a persistent policy of hand-to-mouth buying, prompted by the expectation of a much lower level of prices.

In January, the market was dull and competitive selling brought prices down to 12 1/8 cents by the beginning of February, when there was a recovery, due to heavy purchases for foreign account, which resulted in an advance to 12.25 to 12.30 cents. The market showed resistance whenever it sank to the neighborhood of 12 cents. This was the situation up to May, when unfavorable statistics caused a break through 12 cents, and about the middle of that month the lowest point of the year was reached with 11.80 to 11.90 cents. Europe took the lead with heavy purchases, and followed by a more liberal demand for home trade, quotations climbed steadily until 12.50 to 12.60 cents was realized toward the end of June. Not long thereafter, the Morocco difficulty began to cloud Europe's political horizon, and an urgent liquidation started in all speculative markets. However, the unsettlement of the stock exchanges had a relatively slight effect upon the price of copper, notwithstanding

[1] *E. and M. J.*, Jan. 6, 1912.

the fear of a serious contraction in the absorbing power of Europe a correspondingly smaller exports. Early in October, the quotation sto at 12.05 to 12.10 cents, the low point. Consumers on both sides of t Atlantic had allowed their stocks to run down to a minimum. Buying a large scale could not longer be put off. Aside from a temporary setba toward the end of October, the movement had started, which took t market out of the rut in which it had stuck practically since the panic

LAKE COPPER SALES.

Company.	1910.			1911.		
	Pounds.	Amount.	Average.	Pounds.	Amount.	Average.
Ahmeek	9,574,512	$1,244,603	12.99	14,336,750	$1,832,892	12.78
Alloues	3,582,241	454,415	12.685	4,482,267	547,719	12.822
Baltic	17,549,762	2,234,645	12.74	15,370,449	1,927,036	12.54
Centennial	1,249,763	164,024	13.124	1,393,830	178,997	12.842
Champion	19,224,174	2,447,844	12.74	15,639,426	1,960,758	12.54
Franklin	1,254,428	167,192	13.328	1,200,789	150,300	12.516
Isle Royale	5,958,931	755,517	12.68	6,134,643	759,279	12.38
La Salle	472,100	.....	.....	211,398	26,851	12.70
Mohawk	11,412,816	1,493,816	13.09	12,091,056	1,527,108	12.63
Osceola	16,744,258	2,184,083	13.04	17,470,124	2,222,776	12.72
Quincy	22,517,014	2,974,086	13.208	22,252,943	2,831,799	12.725
Superior	3,181,041	401,765	12.63	3,103,049	392,621	12.652
Tamarack	10,039,970	1,301,798	12.97	7,162,872	910,111	12.71
Trimountain	5,694,868	725,139	12.74	6,120,417	767,332	12.54
Victoria	586,941	71,900	12.26	1,030,993	126,498	12.27
Totals	125,389,628	$16,219,062	12.856	.....	.....	.....

AVERAGE MONTHLY PRICES OF COPPER MANUFACTURES.
(In cents per pound.)

	1910.		1911.	
	Copper Wire.	Sheet Copper.	Copper Wire.	Sheet Copper
Jan	15.35	18.50	14.06	18.50
Feb	15.25	18.75	13.50	18.50
Mar	14.75	19.50	13.25	18.50
Apr	14.75	19.50	13.75	18.50
May	14.10	18.50	13.75	18.50
June	14.19	18.50	13.75	18.50
July	14.00	18.50	13.90	18.50
Aug	14.00	18.50	13.81	18.50
Sept	14.00	18.50	13.75	18.50
Oct	14.00	18.50	13.50	18.50
Nov	14.25	18.50	13.75	18.63
Dec	14.25	18.50	14.94	19.13
Year	14.41	18.69	13.81	18.56

1907. Favorable statistics in November and December helped to accele ate the pace, and by the end of the year prices had advanced to 14 cent under the influence of an enormous volume of orders. While Europe buyers regained sufficient confidence in the future to anticipate the

requirements a few months ahead, the domestic consumers followed the market only reluctantly. A buying power is therefore in reserve, which may tax the resources of producers.

During 1911, there was a reduction of about 100,000,000 lb. in the world's visible supply of copper. In 1912, a new production of about 150,000,000 lb. is anticipated, besides which there will be whatever the older producers choose to add.

AVERAGE PRICE OF STANDARD COPPER (G. M. B.'s) IN LONDON. (a)
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1902....	48.43	55.16	53.39	52.79	54.03	53.93	52.89	51.96	52.68	52.18	51.03	50.95	52.46
1903....	53.52	57.34	63.85	61.72	61.73	57.30	56.64	58.44	56.82	55.60	56.30	56.36	57.97
1904....	57.500	56.500	57.321	58.247	57.321	56.398	57.256	59.952	57.645	60.012	65.085	66.375	58.884
1905....	68.262	67.963	68.174	67.017	64.875	65.881	66.887	69.830	69.667	71.406	74.727	78.993	69.465
1906....	78.869	78.147	81.111	84.793	84.867	83.994	81.167	83.864	87.831	97.269	100.270	105.226	87.282
1907....	106.739	107.356	106.594	98.625	102.375	97.272	95.016	79.679	68.375	60.717	61.226	60.113	87.007
1908....	62.386	58.786	58.761	58.331	57.387	57.842	57.989	60.500	60.338	60.139	63.417	62.943	59.902
1909....	61.198	57.688	56.231	57.363	59.338	59.627	58.556	59.393	59.021	57.551	58.917	59.906	58.732
1910....	60.923	59.388	59.214	57.238	56.313	55.310	54.194	55.733	55.207	56.722	57.634	56.069	57.054
1911....	55.604	54.970	54.704	54.035	54.313	56.368	56.670	56.264	55.253	55.176	57.253	62.063	55.973

AVERAGE PRICE OF LAKE COPPER PER POUND IN NEW YORK. (a)

Year	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1902.	11.322	12.378	12.188	11.986	12.226	12.360	11.923	11.649	11.760	11.722	11.533	11.599	11.887
1903.	12.361	12.901	14.752	14.642	14.618	14.212	13.341	13.159	13.345	12.954	12.813	12.084	13.417
1904.	12.533	12.245	12.551	13.120	13.000	12.399	12.505	12.468	12.620	13.118	14.456	14.849	12.990
1905.	15.128	15.136	15.250	15.045	14.820	14.813	15.005	15.725	15.978	16.332	16.758	18.398	15.699
1906.	18.419	18.116	18.641	18.688	18.724	18.719	18.585	18.706	19.328	21.722	22.398	23.350	19.616
1907.	24.825	25.236	25.560	25.260	25.072	24.140	21.923	19.255	16.047	13.551	13.870	13.393	20.661
1908.	13.901	13.098	12.875	12.928	12.788	12.877	12.933	13.639	13.600	13.646	14.386	14.411	13.424
1909.	14.280	13.295	12.826	12.938	13.238	13.548	13.363	13.296	13.210	13.030	13.354	13.647	13.335
1910.	13.870	13.719	13.586	13.091	12.885	12.798	12.570	12.715	12.668	12.788	12.914	12.863	13.039
1911.	12.680	12.611	12.447	12.275	12.214	12.611	12.720	12.634	12.508	12.370	12.769	13.768	12.634

AVERAGE PRICE OF ELECTROLYTIC COPPER PER POUND IN NEW YORK. (a)

Year	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1902.	11.053	12.173	11.882	11.618	11.851	12.110	11.771	11.404	11.480	11.449	11.288	11.430	11.626
1903.	12.159	12.778	14.416	14.454	14.435	13.942	13.094	12.962	13.205	12,801	12.617	11.952	13.235
1904.	12.410	12.063	12.299	12.923	12.758	12.269	12.380	12.343	12.495	12.993	14.284	14.661	12.823
1905.	15.008	15.008	15.125	14.920	14.627	14.673	14.888	15.664	15.965	16.279	16.599	18.328	15.590
1906.	18.310	17.869	18.361	18.375	18.457	18.442	18.190	18.380	19.033	21.203	21.833	22.885	19.278
1907.	24.404	24.869	25.065	24.224	24.048	22.665	21.130	18.356	15.565	13.169	13.391	13.163	20.004
1908.	13.726	12.905	12.704	12.743	12.598	12.675	12.702	13.462	13.388	13.354	14.130	14.111	13.208
1909.	13.893	12.949	12.387	12.563	12.893	13.214	12.880	13.007	12.870	12.700	13.125	13.298	12.982
1910.	13.620	13.332	13.255	12.733	12.550	12.404	12.215	12.490	12.379	12.553	12.742	12.581	12.738
1911.	12.295	12.256	12.139	12.019	11.989	12.385	12.463	13.405	12.201	12.189	12.616	13.552	12.376

(a) From *Eng. and Min. Journ.*

The London Copper Market

The year opened with optimism. Statistics showed a substantial decrease in the European visible supply. American statistics disclosed an unwelcome increase in the output of the refineries during December, coinciding with a decrease in consumption, and the immense quantities shipped from America to Europe. With trade requirements covered, January closed with values £54 17s. 6d. and £55 13s. 9d. respectively.

In February, European statistics were satisfactory, selling pressure had ceased, trade requirements were large, and speculative interest was encouraged. Later, rumors of adverse American statistics for January intimidated buyers and short selling was encouraged; the result being a decline, but further decline was arrested by persistent renewal of trade demand. Thereafter fluctuations were within narrow limits, but influenced partly by liquidation by tired holders and partly by apprehension as to the effect of a decision of the Railroad Commissioners in the United States on the freight-rate question. In March, statistics disclosed a decrease of 442 tons in the European visible supply, which was insufficient to counteract the downward trend. Throughout the month fluctuations were within narrow limits. April closed with a persistent decline, due to lack of speculative enterprise. European statistics for first half of April disclosed a reduction of 2664 tons in the visible supply, which tended to check the decline. The Amalgamated Copper Co.—after long abstention—entered the market at the reduced price of £56 10s. for Europe. In May, some buoyancy on the Stock Exchange, and a reduction of 1535 tons in the visible supply combined to infuse some speculative interest into the market, and resulted in a temporary advance. In June the expansion in European consumption and evidence of increasing consumption in America were followed by enhanced prices. In July, demand was more readily satisfied by disappointed speculative holders than by the leading producers whose accumulation of stock was an element of weakness in the market. Manufacturers were consequently cautious in entering the market. Even the mid-monthly statistics, showing a decrease of 2273 tons in the European visible supply, failed to dispel the prevading apathy. Throughout the month of August the consuming industries were well employed, but sellers were always found ready to supply the metal at current values, which moved within narrow limits. In September, declining values caused heavy losses, wherein copper-mining securities participated largely. There was little pressure to sell, nor would the leading producers make any concessions in price. In October, the Amalgamated group of American producers reappeared as sellers of the refined metal at prices

not sufficiently low to tempt the trade. A great expansion of business and an improvement in prices set in.

Copper Mining in the United States.

Alaska.—(A. H. Brooks.[1])—The total production of copper in Alaska since 1901, when systematic mining of this metal began, is about 56,700,000 lb., valued at about $8,170,000.

Copper mining in Alaska received an enormous impetus in 1911, due chiefly to the activity in the Prince William Sound region and the Kotsina-Chitina belt. The production was more than five times that of 1910. In the Ketchikan district productive mining was continued in the Mount Andrew, It, Rush & Brown and Jumbo mines during 1911. The Red Wing mine was equipped with a compressor and drills and preparations were made to begin shipping ore. Considerable development work was done on the Victory property, near Seal Bay.

Of the Prince William Sound properties the Beatson-Bonanza and Ellamar were operated on a productive basis throughout the year. Some shipments were also made from the Threeman property, on Landlocked Bay. This mine has been systematically developed on three levels, and the completion of a new wharf in 1911 places it in a position to become a regular producer. The Fidalgo Bay Copper Mining Co. has also arrived at the stage of shipment, an aerial tram having been completed from mine to wharf in 1911. The Seattle-Alaska Copper Co. installed a concentrating plant at its mine on Latouche Island, which has been developed by an adit said to cross-cut the ore body 400 ft. below the outcrop. Some developments were also made on the Barrack Girdwood claims on Latouche Island. The Knight Island Copper Co., at its property near Drye Bay, installed an air compressor and drove two tunnels in which an ore body said to be 12 ft. wide is exposed at a depth of 85 ft. below the outcrop.

The improved transportation facilities furnished by the Copper River Ry. led to more development work on copper properties in the Kotsina-Chitina region than in any previous year, but details in regard to this work are lacking. The shipping of ore from the Bonanza mine began in April and continued throughout the rest of the year. A plant was installed at the end of the railway for concentrating the lower-grade ores of the talus slope at the Bonanza mine. Much development work was done on the Mother Lode property, which is on the McCarty Creek side of the Bonanza divide, and a large body of high-grade ore is said to have been blocked out. Systematic development work was

[1] *E. and M. J.*, Jan. 6, 1912.

also undertaken at the Nikolai property, also in the McCarty Creek basin. At Copper Mountain, near the western end of the belt, the ore body was cross-cut at a depth of 1000 ft. below the outcrop. It appears that developments in this vicinity have been sufficient to block out a large amount of ore, and the depth to which the lode has been traced is an encouraging feature, indicative of the permanency of at least some of the ores in this part of Alaska.

The Bonanza ore is unusually rich, cargos assaying 71 per cent. copper, and 8 to 20 oz. silver. The ore is the main source of traffic on the Copper River and Northwestern Railway. The mine is owned by the Kennicott Mines Company, capitalized at $3,000,000. Alaska ore is shipped to the Tacoma plant of the A. S. & R. Co., which has increased its facilities.

Arizona.—The production of Arizona was the largest of any state in the years 1908, 1910 and 1911; the last year records a record production of 300,578,816 lb. Comparing the total productions, Arizona ranks third, Montana being first and Michigan second.

(By James Douglas[1]).—The last year was one of activity in building new and remodeling old works in Arizona. The porphyry mines progressed beyond the development stage, and two of them, the Miami and the Ray Consolidated, commenced to produce, while the increased quantity of concentrates, and the decreasing quantity of lump ore in the older districts, necessitated the addition of reverberatories to the cupola plants.

In the case of the Arizona Copper Co. a decision was reached to abandon the old and to erect new smelting works exclusively on the reverberatory system. It is understood that the new works will be designed, not so much for increased production, as for greater economy in smelting. Along these lines the Copper Queen, which now receives from 300 to 400 tons of concentrates daily from the Moctezuma Copper Co., and which makes between 100 and 200 tons of flue dust daily, will put two reverberatories into operation. The Calumet & Arizona Co. is remodeling the old works at Douglas, increasing the capacity and adding roasting and reverberatory furnaces. At the United Verde, Mr. Clark is planning new works in a more central situation to the ore production of the district.

However, production in the older camps was well maintained, while the new mines probably brought the production of the state to a somewhat higher figure than the maximum of any previous year. The Copper Queen produced slightly less copper than in 1910. No startling developments were made underground, but new discoveries more than kept

[1]*E. and M. J.* Jan. 6, 1912.

pace with extraction. The total production of the Douglas smeltery is estimated at 111,000,000 lb., of which about 26,000,000 lb. was contributed by the Moctezuma Copper Co. The combination of the Calumet & Arizona with the Superior & Pittsburg cannot but prove beneficial to both companies, and it is already reflected in a notable increase in the output of copper. The combined companies will probably make a production of about 50,000,000 lb. The only other notable producer in the district was the Shattuck-Arizona, with an output of copper in its ores—which were smelted at the Copper Queen smeltery, at Douglas—of between 4,000,000 and 5,000,000 lb. The Courtland camp, in the Dragoon Mountains, which, 3 years ago, promised to be a prominent producer, did not realize expectations. Its copper output has probably been 2,000,000 lb.

In the Clifton district the most promising development was the discovery of profitable secondary ores at considerably greater depth than was heretofore assigned as the limit in depth of the altered zone. These deeper ores occur along certain fault planes in the Humbolt mine of the Arizona Copper Co., and the West Yankee of the Detroit Copper Co., and encourage the hope that the district may possess resources heretofore unsuspected. The production of the district, including the Shannon, and small quantities of ore from outside companies, will amount to about 70,000,000 lb.

In the Globe district the Old Dominion smeltery turned out about 5,000,000 lb. more of copper than in 1910. The total output will probably foot up to 25,000,000 lb. or over. This was almost exclusively from the ores of the Old Dominion and the United Globe mines, there being less custom ore in the Globe market than formerly. A notable feature of development during the last 2 years was the depth to which secondary enrichment extended along the main Old Dominion fissure and the size of the ore bodies at a depth of even 1600 ft. from surface. At the Old Dominion, as elsewhere, the increased quantity of concentrates and the decreased quantity of lump ore points to modifications in the company's smelting methods.

The Imperial Copper Co. mines were idle. Little copper was mined in the northern part of the territory.

In regard to the so-called porphyry mines, the Miami took the lead as a producer, and began shipping actively in April. It is currently estimated that its production will reach about 3,000,000 lb. per month, but that consummation will not be reached until the current year has partially expired. Its neighbors, the Live Oak and the Inspiration, confined energies to underground development and to planning large concentrating works. The Ray Consolidated is reported to be treating 7000

tons per day, securing 18 per cent. concentrates. It expects to treat ultimately 13,000 tons per day. The Hayden plant of the American Smelting & Refining Co. on the Gila River will have two reverberatory furnaces and two basic converters in operation in April, 1912.

The Miami District is described by W. R. Ingalls.[1] Four mines have been opened, viz., the Miami, Inspiration, Keystone and Live Oak, the first being the only producer.

The report of the Miami Copper Co., for the year ended Dec. 31, 1911, shows a total of 445,036 tons of 2.48 per cent. Cu ore milled, and 20,065 tons of 40.36 per cent. concentrates produced, equivalent to 36.39 lb. of copper per ton of ore, or 73.37 per cent. mill extraction. From the concentrates were produced 15,385,783 lb. of fine copper. The cost of copper on board cars at Miami is shown in the accompanying table.

COSTS AT MIAMI, 1911.

	Per Ton Ore.	Per Pound Cu.
Mining	$1.2134	$0.03500
Milling	0.6274	0.01810
General	0.1706	0.00492
	$2.0114	$0.05802

In order to protect the ore from admixture with waste, and to permit complete extraction of ore without leaving pillars behind, it was found necessary to mine the outer ore by square sets and slicing so as to protect the central mass by a mat of timber. As this outer ore contains oxidized copper which cannot be efficiently saved in milling, the extraction obtained is not considered as good as will be had later. Similarly, the cost of mining, $1.21 per ton, is not considered as good as will be attained after the top layer of ore is removed by the caving system.

The capping has already begun to cave over the ground extracted by square-set timbering.

The mill was started on Mar. 15, 1911, with one section, at the end of the year five sections were running, milling 2500 tons per day, and the equipment of the sixth section was complete except for the fine-grinding machinery.

During the year 22,888 ft. of drifts and 10,047 ft. of raises were driven. The work consisted chiefly of completing the 420-ft. tramming level and opening up the ground in the northwest portion of the ore body, the limits of which had been defined by churn drilling. Ore reserves are estimated at 18, 232,000 tons, averaging 2.58 per cent. copper. Drilling has been discontinued except on the Red Spring claim, where new ore of commercial grade has been found.

[1] *E. and M. J.*, Jan. 13, 1912, p. 119.

Besides the expense, exclusive of general mining expense, of putting the concentrates on board cars at Miami, which on the 14,970,337 lb. of copper already sold is taken at 5.310 cents per pound copper, other expenses are shown as follows: General mining expense, 0.714; freight on concentrates, 0.629; smelting, refining and freight on pig copper, 2.326; selling expense (including freight and insurance on refined copper), 0.171; New York office expenses, 0.175; total, 9.325 cents per pound, from which must be deducted 0.222 cents for proceeds of silver and miscellaneous rentals, leaving a net charge of 9.103 cents per pound. The copper sold at 13.03 cents per pound.

MOCTEZUMA COPPER COMPANY.

Year.	Pounds Copper.	Net Earnings.
1904	11,061,649	$598,922.36
1905	10,160,016	533,117.66
1906	12,714,726	1,195,424.18
1907	9,640,390	833,236.25
1908	15,522,580	524,902.00
1909	26,487,776	1,104,454.00
1910	22,632,180	480,690.55
1911	.....	1,206,183.00

COPPER QUEEN STATISTICS. (a)

Year.	Production, Pounds.	Net Earnings.	Dividends.
1904	50,151,552	$2,960,659.70	$800,000
1905	64,625,955	5,609,486.30	2,300,000
1906	79,219,655	7,625,854.76	6,500,000
1907	63,341,055	4,471,137.08	3,800,000
1908	76,125,162	.....	3,000,000
1909	84,429,791	5,271,278.20	4,025,000
1910	76,428,908	4,623,856.12	6,300,000
1911	111,445,229	4,155,011.00	5,200,000

(a) The difference between earnings and dividends is represented by expenditure on increased plant facilities, and undistributed. Since 1887 the company has paid a total of $45,585,000 in dividends.

DETROIT COPPER MINING COMPANY.

Year.	Production, lb.	Net Earnings.	Dividends. (a)
1904	16,424,394	$603,340.00	.....
1905	14,632,117	532,684.28	.....
1906	20,347,497	973,456.42	.....
1907	17,974,581	814,874.11	$80,000
1908	24,223,172	1,072,016.00	480,000
1909	24,591,021	1,153,269.66	760,000
1910	23,056,292	1,079,546.90	1,840,000
1911	22,704,398	930,496.00	800,000

(a) The company had paid a total of $8,357,590 in dividends up to the end of 1910.

As to finances, a condensed balance sheet is here given. The bonded indebtedness was reduced $18,000 in 1911 by conversion of bonds. While the cost of development has been debited to capital, development account has been credited with 31 cents per ton of ore hoisted, which, if continued, is estimated to be sufficient to liquidate fully all past and future development costs.

The Detroit Copper Mining Company, publishes the following data regarding its 1910 concentrating operations: There were 474,037 tons of ore milled carrying 2.977 per cent. of copper. The concentrates amounted to 69,907 tons assaying 15.426 per cent. copper, a concentration of 6.78:1, and a recovery of 76.42 per cent. of the copper contents. The tailings assays showed 0.82 per cent. of copper, and the water used amounted to 554 gallons per ton of ore; 517,087 tons of ore were treated all told. The production, 1911, etc., is given in the two tables.

In the report of Phelps, Dodge and Company for the year 1911, it is stated that Copper Queen mines did 62,150 ft. of development work and 884 ft. of diamond drilling. Sulphide ore was found in Cambrian limestone in the Spray mine. Economies in timbering and power are reported. There were 854,463 tons of ore smelted giving 24.94 per cent. matte fall of 36.7 per cent. copper. The Moctezuma mines given here are in Sonora. The reserves of broken ore were increased to 302,000 tons. The Moctezuma concentrator treated 517,352 tons of 3.1708 per cent. ore, and produced 113,222 tons of 11.932 per cent. concentrates (4.569:1), the saving amounting to 85.19 per cent., and the tailings assay being 0.5579 per cent. The Copper Queen smeltery treated these concentrates and also Moctezuma ore, which together were 13.1 per cent. of the ore smelted.

PRODUCTION OF COPPER QUEEN SMELTERY.

	Tons Treated.	Copper lb.	Silver oz.	Gold oz.
Copper Queen ores and precipitates	629,268	75,661,152	834,658	15,423
Moctezuma ores and concentrates	111,462	25,511,582	361,296	735
Custom Ores	106,751	10,272,489	598,941	10,996
Total, 1911	847,481	111,445,229	1,794,895	27,154
Total, 1910	832,314	116,061,102	1,395,504	20,412

The Ray Consolidated interim report dated Oct. 25, 1911, gives the following details of its operations near Hayden and Kelvin. The eastern section of the mine is adjacent to No. 1 shaft now operating. The ground developed includes 3,000,000 tons of ore ready for stoping. During September, the cost of stoping averaged 65 cents including part of current development, crushing, and delivering on cars ready for sending to the

COPPER QUEEN MINES PRODUCTION.

	Tons.	Copper lb.	Silver oz.	Gold oz.
Ore mined and shipped to Douglas	588,527	73,154,430	599,192	12,015
Precipitates shipped to Douglas	377	359,339	212	2
Lessor's ore shipped to Douglas	5,958	483,301	270,108	1,808
to El Paso	11,359	67,162	357,941	3,070
Sulphide ore shipped to El Paso	586	38,106	.....	.....
to Globe	12,325	387,390	.....	.....
Total, 1911	619,132	74,489,728	1,227,453	16,895
Total, 1910	596,193	76,428,908	608,096	12,430

CONCENTRATOR STATISTICS, DETROIT COPPER COMPANY, 1911.

	Tons.	Per Cent. Copper.
D. C. Co., ore concentrated	500,000	2.869
Ore purchased	.....	2.889
Concentrates produced	66,012	15.876
Saving, percentage	.....	73.05
Tailings assay	.....	0.848
Tons milled per ton of concentrate	7.57	.....
Water used per ton of concentrate	522.6 gal.	.....
Actual running time for year	95.32%	.....
Ore milled per 24 hours	1,737	.....

mill. No. 2 shaft in the western part of the property will not be ready to handle full tonnage until the ore also comes from No. 2 level which is being connected. The average copper contents of all ore mined has been a little under 2 per cent., and at present, 1.85 per cent., but it is to be maintained at 2.17 per cent. The mill is built in eight sections of which four are handling 800 tons each per day. To October 1, 1911, 9,000,000 lb. of copper was produced; at present 2,500,000 are being produced with a recovery of 69 per cent. The American Smelting and Refining Company purchased at cost, the smeltery at Hayden. Pending completion of this plant, the concentrates are sent to El Paso, but the saving per pound of copper by shipping to Hayden will be approximately 1 1/2 cents. The December output of copper was 2,660,000 lb.

Arizona Copper Co.—The report for the year ended Sept. 30, 1911, shows a total production of bessemer copper of 34,569,019 lb. as compared with 32,161,205 lb. for the previous year. This is the largest output since the company began operation. The total ores mined from the company's properties during the year amounted to 744,746 dry tons, which indicates a yield of 46.4 lb. of copper per ton of ore. Of the total ores mined, 94 per cent. were concentrated, the remaining 6 per cent. being direct-smelting ores, the latter figure including pyrites for the acid plant and siliceous ores for converter lining. Of the total copper produced, 82 per cent. came from sulphide and 18 per cent. from oxide ores.

The mines making the largest production were: the Humboldt, producing 281,417 tons, of which 263,901 tons were concentrating ores, 8673 tons direct smelting, and 8784 tons siliceous ores for converter lining; the Coronado mine, which produced 142,571 tons of sulphide ores, of which all except 20 tons was concentrated; and the Metcalf mine, which produced 102,666 tons of oxide concentrating ores from surface and underground workings.

During the year the oxide concentrator treated 104,764 dry tons of ore, the tailings from which, amounting to 90,526 dry tons, were treated in the leaching plant. From these ores 11 1/2 per cent. of the total copper production was obtained. The sulphuric-acid plant supplied the leacher with 3779 tons of acid. The total ores and concentrates smelted during the year amounted to 150,991 tons, with fluxes and by-products re-smelted the amount was 230,833 tons. The average yield of copper from ores and concentrates was 11.22 per cent.

The proposed new works of the Arizona Copper Co., which includes the re-modeling both of the concentrating and the smelting plants is discussed in *Eng. and Min. Journ.*, Dec. 16, 1911.

STATISTICS OF UNITED VERDE COPPER COMPANY.

	1907.	1908.	1909.	1910.	1911.
Copper production, lb	33,012,339	36,183,089	36,694,063	38,663,800	33,167,987
Silver production, oz	.....	494,574	495,480	563,133	461,145
Gold production, oz	11,730	20,334	17,022	19,267	15,239
Cost of mining	$1,068,293	$796,529	.....	.....	.....
Cost of smelting	1,281,674	1,215,993	.....	.....	.....
Charged to depreciation	298,243	320,407	.....	.....	.....
Price received for copper, per lb	18.167c.	13.343c.	12.97c.	12.69c.	12.59c.
Price received for silver, per oz	64.158c.	52.453c.	.....	.....	52.4 c.
Price received for gold, per oz	$20.50	$20.50	.....	.....	20.72c.
Cost of copper, per lb (a)	10.54c.	9.046c.	.....	.....	.....

(a) Value of the precious metals is not credited against cost of producing the copper.

CALUMET & ARIZONA MINING COMPANY.

Year.	Net Earnings.	Copper Produced, lb.	Price Received.	Gold and Silver Value.	Gross Product.	Cost Per lb.
1904	$1,682,518	31,638,660	12.562c.	$195,926	$4,170,374	7.86c.
1905	2,314,268	31,772,896	14.932c.	178,843	4,923,172	8.21c.
1906	4,827,872	37,470,284	17.96c.	238,464	6,968,127	5.71c.
1907	2,114,047	30,689,448	18.102c.	210,846	5,765,636	11.22c.
1908	857,700	28,048,329	12.948c.	234,358	3,859,854	11.00c.
1909	1,214,495	27,630,050	13.531c.	211,760	3,950,311	10.38c.
1910	1,119,848	28,029,506	12.9316c.	265,364	3,889,927	9.72c.
1911	893,639	21,476,739	12.4922c.	301,369	2,983,665	8.33c.

Calumet and Arizona.—The development work amounted to 26,195 ft. in 1911 as compared with 30,850 in 1910. The tons of ore mined per foot of this work were 10.1 in 1911. In the work are included 1378 ft. of drilling, all on the Oliver. The congested condition at the old smelter

was relieved by two more converter stands and basic practice introduced. The new smelter was decided on at the end of the year and will contain two 48×40 blast-furnaces, four 19×100 reverberatories, twelve 21 ft. Herreschoff roasters and six 12 ft. diameter converters.

The Ray Central has developed the Globe Isabella mines by 25,930 ft. of work including 5403 ft. of churn drilling. The underground work consisted of 3215 ft. of drifts, 175 winzes and 253 ft. raises. This is reported by Weed and Probert in September, 1911. It is purposed to mine the developed rich ore first while preparing to cave the lower-grade ore body. Of the 6,700,000 tons, 10 per cent. may be lost in mining, but it is estimated that a profit of 90 cents per ton will be derived from this, based on a cost of 10 cents per pound, or $2.95 per ton. Also the high grade costing $5.75 per ton or 7 cents per pound of copper, should give a profit of $5 per ton for 673,000 tons.

California.—Owing to litigation concerning smelter fumes, the production fell off nearly 20 per cent. from 1910, as it did also in the year from 1909. The Farmers' Protective Association of Shasta County, alleging injury to crops, was successful in enforcing conditions which it was difficult for the smelters to fulfill at a profit. Nevertheless, they spent a great deal of money in trying to remedy the troubles. The Bully Hill and Balaklala finally closed in 1910, the latter installing the Cottrell process and diluting the sulphurous gases. The Mammoth, also, experimented but reduced its production. In Humboldt County, the development of copper claims on Horse Mountain gave an impetus to exploitation. A plant for the reduction of copper ores was being installed at the close of the year.

The record of smeltery operations at the Balaklala Consolidated Copper Co., Coram, Shasta County, Calif., as shown by the annual reports, gives the ratio of ores smelted as one ton of siliceous ore to 14.1 tons of sulphide ore. The total burden to blast-furnaces was composed of 77.6 per cent. sulphide ores; 5.6 per cent. siliceous ores; 4.15 per cent. matte; 2.75 per cent. slag; and 9.9 per cent. limerock. The percentage of ore treated by blast-furnace was 99.38 per cent., 0.62 per cent. of siliceous ores being used as converter lining. Coke consumed equaled 1 ton for every 17.1 tons handled, or 1 ton for every 14.9 tons of copper-bearing material treated. The percentage of coke to total burden was 5.90 per cent., and of coke to sulphides, 7.58 per cent. Flue dust produced equaled 1 ton for every 49 tons of copper-bearing material treated, or 56 tons of burden. Of the total metal contents of the material treated, the flue dust contained: 1.49 per cent. of the gold; 1.21 per cent. of the silver; and 2.24 per cent. of the copper, or approximately 2.12 per cent. of the total value in dollars. The average furnace burden per day was

453 tons, and of sulphides 352 tons. Two 52×240-in. water-jacketed blast-furnaces were operated during the time this record was made. Later reports show ratio of tons smelted as follows: sulphide ores, 70.6 per cent.; siliceous ores, 11.3 per cent.; flue dust, 2.6 per cent.; and limestone, 17.2 per cent. The cost of smelting was about $3 per ton treated.

Colorado (By Geo. E. Collins).—There has been no change in the output of copper, the production of which continues to be incidental to the mining of other metals, and particularly of gold and silver. A number of small producers of purely copper ores may be noted in the San Juan. The principal of these, the Frank Hough, has been idle throughout the year on account of the destruction by fire of its surface equipment. It is probable that this mine will be re-opened by tunneling through Engineer Mountain during 1912. The greatest production, however, is at Leadville, where copper occurs intermixed with the well-known complex sulphides of that district.

Idaho.—The principal point of interest in mining in Custer County in 1911 has been the leasing of the Empire Copper Company's property at Mackay, the shipments being 220 cars. This is a case of a large scale resulting in forced production of low-grade ore that did not pay, but the paying by leasers argues a large deposit. The ore is shipped to Salt Lake. The Lost Packer mine was operated through the year and produces in its 100-ton hot blast pyrite furnace, the highest gold-bearing matte in the United States. The smelter was run only at short periods but produced $500,000 in gross value of matte. The vein is a true fissure in granite and has been proven to a depth of 800 ft. by tunnels. Unfortunately, it is situated 125 miles from a railroad.

The Copper Queen at the head of Agency Creek, Lime County, shipped a number of car loads of copper-gold ore.

Michigan.—Michigan now ranks as third producer but in point of total production, it is second. Development work and production were continued as in 1910. The value of the reserves in Mr. Finley's report, was based on existing and proven conditions and did not take into account the possibilities of undeveloped territory. That these are very great, is shown by the history of the mines in the district. Lately drilling has disclosed the sandstone contact farther east than supposed. The amygdaloid beds, though of lower grade, are extensive and uniform and are destined to bear in the future, the greater output of which they now bear 70 per cent.[1] Only two conglomerate beds have mines, and the one has been a practical failure. Lindgren estimated in 1909 that at 15 cent copper, the Michigan mines would last only 15 years, but to-day this estimate is undoubtedly to be extended. Under 1911 conditions, probably one-

[1] R. C. Allen, State Geologist in the Michigan Acad. of Science.

third the copper produced in the State was being made at a loss. The amygdaloid is being worked where its tenor does not exceed 13 to 16 lb. of copper per ton and the reserves of such ore have been enormously increased. With proper economies and the utilization of present facilities rather than new installations the district is likely to produce after the present "porphyries" are gone. The proper valuation of mineral rights such as those in the copper range has not yet been finally determined.

DIVIDENDS PAID BY MICHIGAN MINES.

Mine.	1909.	1910.	1911.	Mine.	1909.	1910.	1911.
Calumet & Hecla	$2,700,000	$2,400,000	$2,400,000	Quincy	$440,000	$550,000	$440,000
Copper Range Con.	1,536,930	1,357,104	1,357,104	Tamarack	.....	.....	.....
Mohawk	300,000	200,000	175,000	Wolverine	300,000	600,000	540,000
Osceola	769,200	673,050	637,050	Totals	$5,045,940	$5,647,836	

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Adventure	2,182,608	1,380,480	1,606,208	1,552,628	1,244,872	90,870	*Nil*	*Nil*	*Nil*
Ahmeek	.....	350,000	1,552,957	3,077,507	5,527,672	6,280,241	9,198,110	11,844,954	15,196,127
Allouez	.....	.....	1,167,957	3,486,900	2,934,116	3,047,051	4,031,532	4,655,702	4,780,494
Atlantic	5,505,598	5,321,859	4,049,731	1,439,082	*Nil*	*Nil*	43,483	19,018	*Nil*
Baltic	10,580,997	12,177,729	14,384,684	14,397,557	16,704,868	17,724,854	17,817,836	17,549,762	15,370,449
Cal. & Hecla	76,490,869	80,341,019	83,812,370	94,529,821	88,055,723	81,660,723	74,593,553	72,672,469	72,861,925
Centennial	.....	641,294	1,446,584	2,253,015	2,373,572	2,196,377	2,583,793	1,572,566	1,493,834
Champion	10,564,147	12,212,954	15,707,427	16,954,986	16,489,436	17,786,763	18,005,071	19,224,124	15,639,426
Franklin	5,309,030	4,771,050	4,206,085	4,571,570	4,401,248	3,703,421	1,615,556	966,353	820,203
Isle Royale	3,134,601	2,442,905	2,973,761	2,937,098	2,667,608	3,011,664	5,719,056	7,567,399	7,490,120
Mass	2,576,447	2,182,931	2,007,950	2,106,739	2,078,677	1,766,930	1,723,436	1,321,885	1,326,898
Michigan	275,708	2,746,127	2,891,796	2,875,341	2,665,404	3,000,206	1,979,305	36,682	327,773
Mohawk	6,284,327	8,149,515	9,387,614	9,352,252	10,107,266	10,295,881	11,248,474	11,412,066	12,091,056
Osceola	16,059,636	20,472,439	18,938,965	18,588,451	14,134,753	21,250,794	25,296,657	19,346,566	18,388,193
Phoenix	202,823	1,162,201	273,219	*Nil*	*Nil*	*Nil*	*Nil*	*Nil*	*Nil*
Quincy	18,498,288	18,343,160	18,827,557	16,194,940	19,796,058	20,600,361	22,511,984	22,517,014	22,252,943
Superior	.....	.....	.....	.....	.....	21,244	1,789,315	3,181,041	3,236,233
Tamarack	15,286,093	14,961,885	15,824,008	9,832,644	11,078,604	12,806,127	13,533,207	11,063,606	7,494,077
Trimountain	9,237,051	10,211,230	10,476,462	9,507,933	8,190,711	6,034,908	5,282,404	5,694,868	6,120,417
Winona	1,036,944	646,025	*Nil*	278,182	1,285,863	*Nil*	*Nil*	*Nil*	*Nil*
Wolverine	8,999,318	9,764,455	9,464,418	9,548,123	9,273,351	9,555,233	9,971,482	10,469,253	9,630,639
Victoria	.....	.....	.....	.....	1,207,237	1,290,040	1,164,564	1,164,564	1,303,331
Others	75,000	50,000	.....	.....	100,000	.....	2,862,233	899,647	294,873
Totals	192,299,485	208,392,485	218,999,753	224,071,103	220,317,041	222,123,688	230,971,051	223,179,539	216,119,011

Operations in the Lake Superior District (By Carl L. C. Fichtel[1]).—The Keweenaw Copper Company found poor ground and development was suspended about December 1.

The Calumet and Hecla opened on the same lode, viz., the Kearsarge, at the 300-ft. level and found encouraging results.

The Ojibway opened 19 stopes and started 150 tons per day shipments to the Tamarack mill. The Mohawk operated normally and the south end of the property was found improving. Shafts 3 to 4 of the Ahmeek,

[1] Abstract *E. and M. J.*, Jan. 6, 1912.

which entered the list of dividend producers were established in the lode and much development was done. The steel shaft house of the Allouez No. 2 will be ready in 1912 and both shafts ready for a large tonnage.

The old workings of the Wolverine maintained normal working and good progress was made in the drift to open new territory eastward between shafts 3 and 4 on the twenty-eighth level. Centennial extended drifts toward the Wolverine ground. Good drill cores were obtained on the Mayflower and Old Colony on a lode supposed to be the same as the one on the St. Louis tract opened by the Calumet and Hecla.

The Calumet and Hecla worked normally at all shafts on the conglomerate except No. 11 which was delayed by the burning of the shaft house. On the amygdaloid, alterations were made on shaft equipments and it is expected that 2000 tons per shaft per day will be the ultimate tonnage with these changes. At No. 16 a 24×48 Blake crusher was installed. Shaft No. 21 on the Kearsarge lode was operated continuously. Power economy was given much consideration and an exhaust steam turbine at the mill to drive a 7500 kw. generator is planned. The mill is to be enlarged with a re-grinding department. The re-treatment of tailings, a most valuable asset, is to be the main object. Another object is the treatment of ores from many mines in one plant.

The C. and H. report for 1911 shows 2,909,972 tons of rock mined at an operating cost of $1.84 as compared with 2,795,514 tons in 1910 at $1.92. The copper per ton of rock was 25.77 lb. in 1910 and 25.47 lb. in 1911. The cost per pound of copper produced was 8.52 cents. The total dividends paid including 1911 are $115,850,000.

The Tamarack adopted a policy of breaking developed ground and curtailed its development, thus reducing operating expense.

The Quincy put down its No. 9 shaft to 1400 ft. with indifferent results. The Hancock did much development work and its shaft was put down to 3100 ft. and cut a good copper-bearing lode. At the Allouez Centennial mill, a test run of 6 months showed a recovery of 18.21 lb. of copper per ton. The Isle Royal Company was put on a paying basis and did development work in its southern shafts Nos. 4, 5 and 6.

The Atlantic mill was used by the Superior Company. The Copper Range Consolidated acquired control of the Atlantic property. The three properties of the original group operated normally. A betterment was shown in the lower levels of the Trimountain. Drilling on various properties was successful. The South Lake and Indiana started sinking. The Algomah developed the largest body of copper ore other than the native, ever exposed in the district; native copper, however, was found below the 104 ft. level. The new Lake shaft was completed and the property will probably enter the list of producers.

The report of the Osceola Consolidated Mining Co., for the year ended Dec. 31, 1911, shows 1,246,596 tons of rock stamped at a cost of $1.14 per ton for mining, transportation, stamping and taxes. From this 24,452,912 lb. of mineral were obtained, which contained 75.198 per cent., of fine copper. The cost per pound of refined copper produced was 9.28 cents per pound, as against 9.37 in 1910, 9.47 in 1909, and 10.25 cents in 1908. The cost of stamping alone may be judged from the fact that 63,449 tons of North Kearsarge rock was stamped at the Tamarack mills at a cost of 27.28 cents per ton.

The new equipment installed during 1911 at the Osceola mill consists of Woodbury classifiers and jigs, Wilfley tables and Hardinge mills. The results show that the new washing system gives an extraction of about 79 per cent,, as compared with about 73 per cent. formerly, or a saving of 1.1 lb. of copper per ton on the present grade of Kearsarge rock. The capacity per head is about the same, with an added cost of treatment of about 1 cent per ton of rock. The grade of mineral secured is substantially the same. The decrease in the copper content of the rock stamped is due to the falling off in content of the North Kearsarge ore.

The Mohawk Mining Co. of Michigan, in its 1911 annual report states that 902,859 tons of ore were hoisted and 802,548 tons milled, producing 15,760,700 lb. of concentrates, or 12,091,056 lb. of refined copper. This indicates that 11 per cent. of the ore hoisted was discarded by sorting; that the ratio of concentration was approximately 102 tons ore into one ton of concentrates; and yield of refined copper per ton of rock stamped, 15.07 lb. The cost of mining, transportation, and stamping was $1.406 per ton of rock stamped or 9.33 cents per pound of refined copper; smelting, freight, marketing and office expense, 0.81 cents and construction 0.255 cents per pound copper; a total cost of 10.399 cents per pound.

The underground cost was 81.6 cents per ton of ore hoisted. Of the underground costs, pumping and hoisting cost 3.58 cents per ton hoisted; and power drills 7.2 cents per ton. The rock-house expense, crushing and sorting, is about 6.65 cents per ton hoisted. Transportation of ore to mills was 12.1 cents per ton. Stamping was 21.9 cents per ton stamped. Development work consisted of 15,458 ft., or 1 foot for every 50.5 tons of unsorted ore.

The Quincy Mining Co. has paid dividends for 48 years, with a total of $19,880,000. 1,382,524 tons of rock were milled producing 32,550,440 lb. of mineral. Development is still well ahead of stoping, and the deepest openings in the mine show rock up to the average of recent years. The average depth on the dip of the lode is about 5200 ft. compared with 8600 ft. in the deep workings of the Lake Superior mines. The most notable change in methods in the Quincy mines is the sub-

stitution of small light-weight drilling machines using high-pressure air for the heavy machines previously used. About 100 of the new machines have been installed. Dr. R. H. Richards has shown that a better saving of copper can be made by more careful hydraulic classification. To this end, No. 3 classifier is being re-built, from which interesting results are expected.

Montana.—In 1911 the state was second to Arizona, although the total output of the Butte district exceeds that of any state or district. The normal production in recent years has been 300,000,000 lb. Recent observations on the Butte district are given by W. R. Ingalls.[1]

Much ore has been obtained below 1600 ft. although most of it has come from above that depth, and the grade appears to be as good with depth as before. The development also is such that with the efficient technical organization, the mines are capable of producing copper as required and with low costs.

(By F. L. Fisher[2]).—The absorption of the various operating companies, formerly constituting the property of the Amalgamated Copper Co., by the Anaconda Copper Mining Co. was completed in 1911. The only other companies now are the Tuolumne, East Butte, Davis-Daly North Butte, Butte & Ballaklava and Alex Scott. The reduction of the output of the mines to three-quarters of their normal capacity, which was instituted in 1910, was continued through 1911 on account of the low price of copper.

The 23 mines operated during 1911 by the Anaconda company were the Anaconda, Never Sweat, St Lawrence, Mountain Con., Bell, Diamond, High Ore, J. I. C., Original, Moonlight, Tramway, Stewart, Gagnon, East Gray Rock, West Gray Rock, Silver Bow, Berkeley, Mountain View, Pennsylvania, Leonard, East Colusa, West Colusa and Badger State. The shaft of the Speculator mine of the North Butte company was extended to the 2800-ft. level and richer and more extensive ore bodies discovered under the somewhat barren zone between the 2000- and 2300-ft. levels. The shaft of the Granite Mountain mine was enlarged to three compartments from the 1800-ft. level to the surface. Production was maintained at the rate of about 2,000,000 lb. of copper monthly.

The shaft of the Butte & Ballaklava mine was repaired between the 1200- and 1400-ft. levels, where it was damaged by water during the High Ore fire, and shaft sinking was continued near the end of the year; development work was done on those veins where no injunction in the suit of the Anaconda company prevented. The Ballaklava company brought a counterclaim against that company attacking its title to the Right Bower claim.

[1] *E. and M. J.*, Vol. XCIII, p. 550.
[2] *E. and M. J.*, Jan. 6, 1912.

Production at the East Butte company's property was about 1,000,000 lb. of copper monthly. Extensive improvements were made upon the smeltery at the mine; work was begun on a new dust chamber and a slime pond was constructed for working over the old tailings dump. Extensive development work was done on the 1000- and 1200-ft. levels. At the property of the Tuolumne Copper Co. improvements were made and ore blocked out. A complete new hoisting apparatus was installed consisting of a Nordberg engine with capacity for hoisting from the 3500-ft. level and provided with self dumping skips; a new steel gallows frame and steel engine-house were constructed, and new ore bins, having a capacity of 1000 tons, were built; a railroad spur was laid from the bins to the main track, a quarter of a mile, effecting a saving of 60 cents per ton in haulage rates. About 100 tons of ore, mined on the 1200-, 1400- and 1600-ft. levels, were shipped daily to the Washoe smeltery at Anaconda.

At the Colorado mine of the Davis-Daly company development work was done on the 1400- and 1700-ft. levels and high-grade ore shipped at intervals. Development work was done at the Alex Scott company's mine, principally on the 1500-, 1600- and 1800-ft. levels, and an average of 1.5 cars of ore was shipped daily to the East Butte smeltery. At the Ophir mine of the Butte Central Copper Co. the shaft was sunk to the 1000-ft. level and a pump station cut there. Mining was confined chiefly to the silver ore on the 100- and 300-ft. levels from which several shipments were made. Plans were perfected for the building of a concentrator at the mine. The Washoe sampling works, which was destroyed by fire early in 1910, was replaced by a new all-steel and concrete plant.

In the Corbin-Wickes district the Corbin Copper Co. mined ore from the tunnel level. The concentrator was started again in November. Operations at the Boston & Corbin mine were confined to blocking out of ore in readiness for the proposed concentrator. Sinking the shaft to the 1500-ft. level was begun. At the property of the Wickes-Corbin Copper Co. 4 ft. of ore were cut in Tulare Tunnel No. 2, and in the Bunker Hill No. 4 tunnel six veins of various width were passed through.

North Butte copper has had a gradual decrease in ore reserves as stated in the quarterly reports. The levels of the Edith May, and Jessie veins above the 2400-ft. level were developed largely before 1911. Sinking the Speculator shaft occupied most of the year. The work on the lower levels is now being pushed. During 1911, 420,203 tons ore were shipped and 410,694 (dry) tons treated; 27,114 tons were first-class ore. This ore produced 24,816,669 lb. of fine copper, 1,134,300 oz. of silver and 1,280,947 oz. of gold. Average settlements for the year were copper at

12.561 cents per pound; silver 53.769 cents per ounce and gold $20 per ounce.

The report of the Anaconda Copper Mining Company for the year 1911 states that the Butte and Boston and the Parrot operating departments were abolished, all operations being carried on under the Anaconda and the Boston and Montana Departments. The Anaconda shaft which was out of commission the entire year, is being repaired and will be sunk to connect at 2800-ft. level with the High Ore mine. Through the Never Sweat shaft ore was hoisted from the 1800, 2000, and 2200 levels but the shaft will not be used as a main working shaft after connecting elsewhere. The St. Lawrence shaft will also be sunk to the 2800-ft. level of the High Ore, and all water then pumped through the latter. The ore mined in the St. Lawrence was from the 400- to 800-ft., and also from between the 1400- and 2000-ft. level. The Mountain Consolidated shaft was in operation only a few months and is being sunk to establish levels at 2400 and 2500 ft. The 2300-ft. level was started in ore. The High Ore stopes caught fire which was extinguished after two and a half months and work was resumed. The shaft was equipped with skips and five pockets at the lower levels. The engine equipped with new cylinders is to use compressed air. The sinking of the Belmont shaft was accomplished to 2089 ft., or 67 ft. below the Anaconda 2400-ft. level. The connection makes the fifth with the Anaconda workings. In this work three veins were encountered, one of which promises to be of importance as it contains several feet of high-grade ore and all of it is pay ore. The Gagnon shaft will be maintained only for timber and air, as the Original shaft is now connected with the lower workings of the Gagnon mine. The ore from three mines formerly hoisted through independent shafts are now brought up through the Steward shaft. The Steward mine was stoped and developed between the 1700- and 2300-ft. levels; a long continuous shoot of high-grade ore was developed since it was encountered in 1910.

Under the B. and M. department development work was extended on the Mountain View, the Bagden State, the Tramway and the Pennsylvania. The Berkeley shaft was sunk to 1000 ft. in depth and connected with the Silver Bow, and ore was mined on all levels from the 400 to the 1000 ft. A remarkably continuous shoot of ore was opened for 642 ft. on the 600 and extended into the oxidized zone. In the West Colusa, the extension to greater depth of an ore body between the 1200- and 1800-ft. levels was proven. Ample ore was obtained in the levels above the 1800. In the Leonard, a zone of ore about 400 by 500 ft. on the 1600- and 1800-ft. levels is estimated to contain 50 per cent. pay ore.

The extraordinary repairs to shafts and machinery, the fighting of the

fire at the High Ore, etc., were all expenses charged to operating. The benefits of consolidation are only partly reflected as yet in the cost per pound of copper. Refinements in operating methods and new ore treatment are being extended. The Washoe plant at Anaconda has changed one section to a new type of concentrating machinery which increases the saving in copper and also the capacity.

The mines of the Anaconda in 1911 produced 3,844,070 tons of ore and 4602 tons of precipitates. The Reduction Works treated 3,154,036 tons of ore and copper-bearing material at Anaconda and 1,101,778 tons at Great Falls. Of these amounts 3,756,235 tons were from the company's mines and 499,077 tons were purchased, and 5014 tons were precipitates from the Old Works. The amount of fine copper produced was 191,573,780 lb. at Anaconda and 67,833,313 lb. at Great Falls. The silver produced all told was 9,731,561 oz. and the gold 48,950 oz.

Nevada.—In the report made to the Directors of the Nevada Consolidated, Pope Yeatman reports that the total ore reserves on Jan. 1, 1912, were 40,853,371 tons averaging 1.662 per cent. copper. The report covers a period of 15 months on account of the change to calendar years. All mining was done by steam shovels and the most of it came from the copper Flat or Eureka pit, now 137 ft. below the capping. Toward the end of the year also, some ore came from Liberty. The ore concentrated during the 15 months ending Jan. 1, 1912, was as follows:

	Tons.	Assay.
Eureka pit	3,261,482	1.79 per cent. copper.
Liberty pit	62,956	1.768 per cent. copper.
Veteran mine	5,584	3.050 per cent. copper.
	3,330,022	1.791 per cent. copper.

With reduced cost of concentration it was found profitable to use ore lower in grade than before. Silicious carbonate ores amounted to 205,795 tons, but only 39,235 tons were smelted and the remainder stored.

Results obtained during the 15 months ended Jan. 1, 1912, at the Nevada Consolidated:

Tons of ore treated (dry)	3,338,242
Average copper content, per cent	1.80
Percentage of extraction (Cu)	67.59
Ratio of concentration	11.34
Gold content (oz. per ton)	0.013
Silver content (oz. per ton)	0.079
Percentage of extracton (gold) per cent	57.72
Percentage of extraction (silver) per cent	45.92
Average gold and silver recovered per ton of ore	17.35
Average copper in concentrate, per cent	13.80

The Steptoe Valley S. and M. Co. worked the plant at larger capacity and also at low operating cost. Concentration costs were 15.9 cents less than before, a reduction of 25.72 per cent. The cost of roasting concentrates has been reduced 8.3 cents or 12.71 per cent. per ton. Large tonnages in the reverberatories were obtained by changing to Coa-

linga oil firing. The daily charges were increased from 223.5 tons per day in 1910 to 359 tons per day for December, 1911. The costs for smelting were lowered to about 1.9 cents per ton of roasted concentrates.

The cost per pound of copper all told has been 7.17 cents. The copper produced in this period of 15 months was 78,541,270 lb.

In the Mason Valley district, the smelter of the Mason Valley Mines Co., at Thompson, was ready at the end of the year. Ore supplies are from the Mason Valley mines and also the Nevada Douglas. The matte shipped contains about 42 per cent. copper. The plant has two blast furnaces, 46×200 in.

The Nevada Douglas property is 41 miles from the smelter. Its ores are mostly carbonates and also native copper; sulphides are, however, encountered. The maximum depth in the Youngton district has not exceeded 700 ft. The shipping ore of the various properties runs as follows: Mason Valley mines 4 per cent. copper; Nevada Douglas 5 1/2 per cent.; McConnell mine, 5 per cent. The Mason Valley according to the president's annual report has done 15,136 ft. of development work not including 700 ft. of drilling.

New Mexico.—The development on the Chino deposits of Santa Rita is the only important one worthy of recording. At Chino, 1500 tons per day are being put through the concentrator, making 20 per cent. concentrates. It is expected that four sections will be in operation by the middle of 1912, and ultimately to treat 8000 tons per day. The mines in the Burro Mountains, including the Chemung, the Burro Mountain, and the Savannah, were rather sluggishly explored, and no steps were taken toward producing.

The report of the Chino Copper Co., operating mines at Santa Rita and a mill at Hurley, N. M., covers the period of 19 months ending Dec. 31, 1911. True operating was carried on in the last quarter of the year, during which 986,375 lb. of copper were produced, which sold at 13.3 cents per pound. The expense of production per pound of copper was: Mining, 0.774 cents; freight on ore, 0.461; milling, 5.391; treatment, refining and freight, 3.175; selling commission, 0.133; total, 9.934 cents per pound. Ore reserves on Dec. 31, 1911, are given at 55,000,000 tons, containing 2.24 per cent. of copper. It is estimated that of this amount, 32,000,000 tons can be mined by steam shovel, based on mining by this means to a depth of not over 200 ft. As it may be possible to mine by opencut to greater depths, this tonnage may be exceeded. The first section of the mill was started on Oct. 16, 1911, about 15 months after beginning construction work, the second about the end of December, the third on Apr. 1, 1912, and it is estimated the other three will be running before Sept. 1, 1912. To date, $2,234,993 has been spent on account of the new

mill. Up to the end of 1911, about 2,000,000 cu. yd. of material had been removed, of which 1,700,000 was waste, the remainder was partly oxidized ore, most of which was stored, the remainder going to the concentrator, where a recovery of about 22 lb. of copper per ton of ore gave the production and costs mentioned. The concentrates are sent to the smeltery at El Paso.

Tennessee.—The two companies, viz., the Ducktown Sulphur and Iron Co. and the Tennessee Copper Co. had an active year. At the Tennessee Copper Company, at the end of 1911, acid was being made at the rate of 200,000 tons per year. In 1911 there were 444,625 tons of ore mined and 13,808,940 lb. of copper produced, or 31.65 lb. per ton. Including 1,380,000 tons of probable ore, the ore in sight on Jan. 1, 1912 was 5,033,000. The total operating cost is reported as $3.44 per ton of ore and $.1088 per pound of copper, the latter after allowing for the silver and gold. The silver was 90,001 oz. and the gold recovered was 608 oz.

From the figures given in the report, it would appear that the actual cost of smelting was about 89 cents per ton of charge, 33 cents being for fuel. The cost of converting is given as .709 cents per pound of fine copper.

A fire occurred on June 12 which caused shutdown of the acid plant and smelter for one month. Two new Gay-Lussac towers were completed during the year and a larger recovery of niter made thereby.

Texas.—No developments took place in Texas copper ores. Smelting was attempted with some of the Permian ores in 1884; the question of economic treatment of these ores is discussed by Wm. B. Phillips.[1] The copper plant near El Paso will be smelting early in 1912. The plant covers 3 acres at Towne, Texas. It is equipped to burn oil for its reverberatories and its Wedge roasters. It is provided with two basic-lined converters.

Utah.—It is estimated that 13 per cent. of the copper of the United States, including Alaska, is produced within a radius of 20 miles of Salt Lake City, and 51.2 per cent. of the developed or partially developed porphyry copper ores are within this radius.

(By Edward R. Zalinski).[2]—Due to the larger output of the porphyry mines at Bingham, there was an increase in copper production in 1911.

The Utah Copper produced steadily, remodeled its Arthur mill in part, and completed the Bingham & Garfield R. R. New steam shovels were added, bringing the number at work up to 21, the greater part of which was used in stripping operations. Additional shovels were put on overburden on the Boston Consolidated side in June bringing the number

[1] *E. and M. J.*, Dec. 16, 1911, p. 1181.
[2] *E. and M. J.*, Jan. 6, 1912.

of shovels worked on overburden to 19, while an average of two shovels were worked on ore. Steam-shovel operations now extend vertically 1500 ft. Underground mining was done on the Boston Consolidated, but in order to discontinue this as soon as possible stripping operations were pushed, and 3,947,748 cu. yd. of capping, were removed from the Utah and Boston groups in the first three quarters. Additional territory was acquired, covering ground between the original holdings and the Boston Consolidated.

Five sections of the Arthur mill were remodeled, and Chilean mills substituted for the Nissen stamps. Garfield roughing tables and Janney classifiers were added. The plant was not operated at full capacity, but only to such an extent as to take care of the excess of ore produced above the capacity of the Magna plant. The operation of four of the remodeled sections was about equal in capacity to that of the 13 original sections. An extraction of up to 72 per cent. on ore running 1.46 per cent. copper was reported. When the Arthur plant is completely remodeled it will have a capacity of about 8000 tons, which with the 12,000 or more tons treated by the Magna will bring the capacity up to 20,000 tons. The Magna and Arthur mills employ 1460 men.

The Magna plant was operated steadily, but not crowded. A few minor changes were made, including the installation of larger motors, etc. The Copperton plant was dismantled.

The report of D. C. Jackling, General Manager of the Utah Copper Company states the following:

UTAH COPPER STATISTICS.

Ore Reserves, Jan. 1, 1912	301,500,000 tons.
Average copper content	1.532 per cent. copper.
Total 1911 tonnage treated	4,680,801
Average grade of all ore milled	1.51 per cent. copper.
Average cost of milling, per ton	41.68 cents
Average cost of mining, per ton	44.79 cents
Average price received for copper sold	12.646 cents
Net 1911 profits	$6,237,928

The total amount of underground work, 8930 ft. makes 36 miles all told, in the original copper group. Also 33,504 ft. in 1911 were driven in the original Boston Consolidated group and making also about 36 miles all told, since commencement of development. Of these totals 30.6 miles remained accessible at the end of the year, the balance being destroyed by underground or surface work. During the year 8533 ft. of drilling were done and previously 10,015 ft. The average thickness of ore developed at the end of 1911 was 418 ft. Of the ore reserves 229,830,000 tons are fully developed, 62,040,000, being 2 per cent.; 92,130,000 1.6 per cent. and 75,660,000 tons, 1.30 per cent. During the life of the Utah and Boston, 15,885,521 tons, assaying 1.602 per cent. copper, were mined inclusive of 1911. The capping removed during 1911 was 5,450,604 cu.

yd. The extraction at Magna plant averaged 69.28 per cent. and at the Arthur 70.49 per cent. The copper contents produced at both plants were 98,436,224 lb. being 25.62 per cent. of the concentrates.

The Utah Consolidated Mining Co. during 1911 shipped to the International Smelting and Refining Co., 162,522 tons of copper ore carrying copper 2.891 per cent. silver .98 oz.; gold .094 oz. per ton. The development of copper ore in 1911 was not satisfactory, the new ore being below the eighth level and low grade. The copper ore reserves are 239,650 tons averaging 2.24 per cent. copper; .603 oz. silver, and .603 oz. gold. The International plant has a daily capacity for copper ores of 1500 tons.

Copper in Foreign Countries

Africa.—The threatening decrease in the yield from the older mines of the Cape Copper Co. promises to be offset in the not distant future by increasing productivity among the newer mines in the Congo. As yet, these mines are greatly hampered by difficulty of access, by insufficiency of skilled labor, and by climate, though railroads are steadily advancing and their traffic facilities are constantly becoming more reliable.

The Congo.—After many months of disappointment and delay, enough of the plant ordered by the Tanganyika Company to be erected at Lubumbashi was assembled to permit smelting to begin on June 30, and to be maintained without interruption after Sept. 16, 1911. At first, the output was only 12 tons of copper per day; the whole output to the end of 1911 was 3000 long tons (including copper shipped in ore). The first shipment of 400 tons of pig copper reached Antwerp late in 1911, at a total cost of about £65 per ton, netting a loss at the price of copper then current. It is expected that by securing cheaper coke from the Wankie collieries, or from a newly discovered coal field at Luano, belonging to the company, and by other economies in metallurgical practice, the cost will be reduced to £34, to which must be added £9 freight to Antwerp. Of the 100 or more deposits known in the Katanga district, 10 of the largest are under development by the Tanganyika Company, of which the accepted estimate of proved ore is 10,000,000 tons, averaging 10 per cent. copper. Practically no precious metal is found in the ore. The ore is siliceous, and much of it is finely pulverized, which accounts both for the large consumption of coke, and for the high losses in flue dust, that have characterized the smelting operations up to this time. For the oxidized ores, a metallizing process (reduction to metallic copper, followed by concentration) is in contemplation.

Rhodesia.—In southern Rhodesia are no producing copper mines but several gold mines are yielding copper as a by-product. During 1910,

the Valley mine in Gwanda produced 41.72 tons of copper, valued at £2290, in the form of matte. During 1909, 97.21 tons were produced in Rhodesia, valued at £5144. The total output from southern Rhodesia, including 1906, when production began, and ending with 1910, was 359.01 tons valued at £23,058. In northwestern Rhodesia, sandstone and limestone impregnated with copper minerals, occur extensively, especially near the border of the Congo Free State, where the Katanga district lies. The Bwana M'Kubwa mine in northwestern Rhodesia, has an ore consisting of bornite and copper glance. It also has some carbonate ores, for the treatment of which a plant using a recently perfected process will be erected. The Tanganyika Concessions, Ltd., the major portion of whose interests are in the Congo, has also a large copper deposit at Kansanshi in Rhodesia, near the Congo border, where it has a smelting plant. It is at present smelting ore from the Star of Congo mine, in the Katanga district of the Congo State.

Transvaal.—The quantity of copper produced in the Transvaal is small, the total value of the production for 1911 being estimated at about £61,000 as against £71,914 for the previous fiscal year.

The famous mines of the Cape Copper Co. near O'okiep on the Atlantic Coast of Cape Colony near the Orange River are approaching exhaustion. The O'okiep mine, which was one of the largest bonanzas of high-grade copper ore known, is soon to be abandoned. The company has another mine in the northern Transvaal which should at least take its place as a producer. This is the Messina mine, north of Pietersburg near the Limpopo River. Here copper glance exists in shear zones in granite. About 12,500 ft. of length of ancient workings show on the surface, about 2000 ft. has been partially explored to the eighth level and a continuous production has been kept up since 1903. The total ore mined up to June 30, 1911, was 65,788 tons, yielding 9699 tons of concentrates containing 53.1 per cent. copper.

Coal suitable for reverberatory smelting has been discovered within 20 miles. Mine development has been most favorable. The main lode where cut at the 800-ft. level was 27 ft. wide and the ore assayed 17 per cent. copper; 1500 ft. has been driven on profitable ore on the fourth level and one stope 50 ft. long and 25 ft. wide averages 22 per cent. copper. The ore actually blocked out amounts to 102,000 tons assaying 10 per cent. copper and the probable ore above the 800-ft. level to 430,000 tons assaying 10 per cent. copper. The estimated profit will be £600,000 with copper at £55 per ton. There is a small concentrating plant at the mines and the picked ore and concentrates exported assay from 40 to 50 per cent. copper while there is an accumulation of some 13,000 tons of middlings assaying 12 per cent. copper.

The total costs for the year ended June 30, 1911, during which time 18,573 tons of ore were mined and milled, were: Mining $3.80; dump ore 30 cents; milling $1.56; development redemption $1.20; Johannesburg charges $1.02; total $7.88. The mill power is now furnished by a gas engine supplied with gas made in a suction producer from local charcoal, but two seams of coal, one 3 ft., the other 5 ft. wide, have been found on the company's coal land. These are expected eventually to produce fuel for power and smelting.

The Messina Development Co. has arranged an additional issue of £250,000 in 6 per cent. stock guaranteed by the Camp Bird, Ltd., for the purpose of raising working capital with which to develop and equip the mine, enlarge the mill to a capacity of 120,000 tons per annum, and establish smelting works, which have not been built on account of lack of transportation facilities. Arrangements have been made with the South African government by which the railroad now being built will, it is expected, reach the mine by the end of 1912. The Messina copper mine is situated on the government farm, Berkenrode, in the Zoutpansberg district of northern Transvaal. The other properties of the company are in the same same district and adjacent to Berkenrode. The Messina company has acquired 200 coal claims, each 150×400 ft., about 15 miles southwest of Messina.

According to the report of the Cape Copper Co. for the year ended April 30, 1911, the O'okiep mine produced 15,978 tons of 12 per cent. ore, as against 15,196 tons of 14.52 per cent. ore for the previous year. Ore reserves are estimated at 8000 tons of 20 per cent. ore. The Nababeep mines produced 55,202 tons of 5.51 per cent. ore, and the Nababeep North 4803 tons at 6.07 per cent. Reserves here are 172, 200 tons of 5 per cent. ore. Both the O'okiep and Nababeep mines are now being tested in depth by means of boreholes. Explorations are being conducted at the Narrap mines from which 5777 tons of 4.34 per cent. ore was sent to the smelting works during the year under review, and at Springbok and Carolusberg, from which 1165 tons of ore were taken, averaging 8.28 per cent. copper. The Cape Copper Co. paid 5 per cent. for the year on both the preference and ordinary shares, the smallest dividends since 1902. However, in this connection it may be pointed out that since 1888 the common shares have returned their holders 469 per cent., and since 1891 the preference shares have paid 399 per cent.

Argentina.—The Famatina Development Corporation, Ltd., an English company, owning 700 acres of mining land in the Mejicana district in the province of Rioja operated a 100-ton smeltery equipped with converters and produced about 400 tons of copper, 75,000 oz. of silver and

4000 oz. of gold. The smeltery encounters great difficulty in preventing losses of both copper and silver in flue dust. A financial reorganization was determined upon early in 1912. An aerial tramway 21 1/2 miles long was built by the Argentine government from Chilecito at the foot of the Andes to the district. Railroads connect Chilecito with Buenos Aires, La Plata and Rosario. This is but one example of the government's projects to aid in the development of the mineral resources of the country.

Australasia.—The copper-mining industry made no great strides, and the position at the close of 1911 remained much the same as in preceding years, while, looking ahead, it cannot be said there is any probability of a material increase in 1912. However, it must not be thought that Australia is failing to open up new supplies of ore, for in point of fact the number of producing mines has increased but these new mines merely compensated for the shortage created by decreasing production from some of the older ones, and by the interruption of operations at the established mines by labor troubles, etc.

It is impossible to estimate exactly the amount of copper metal originating in the Australian Commonwealth because of the inconsistent figures published by the several states. The combined exports of the Commonwealth in 1911 consisted of 13,154 long tons of high-grade ore and 40,300 long tons of ingot and matte, a slight increase over the exports for 1910. Merton & Co. credit Australasia with 41,840 long tons of copper in 1911, as against 40,315 tons in 1910. Hirsch & Son's figure for 1911 is 44,600 long tons.

The statistics published in the official reports of the several states are as follows, the copper output of New Zealand and Victoria being negligible:

New South Wales.—The bulk of the copper produced was obtained from the Great Cobar, Ltd., the Kyloe, and the Grafton copper mines. There were few fresh developments to record. There are numerous deposits of copper ore in this state, but their exploitation and development are hampered owing to the failure of the legislature to sanction the necessary lines of railway communication. The Great Cobar, Ltd., has purchased the Cobar gold mines, and with the large quantity of auriferous copper ore now available, a steady increase in the output should result. The output by this company for the year was 6548 tons of copper or 300 tons more than in 1910.

Operations at the Great Cobar were hampered throughout the year by shortage of skilled labor. The main shaft was sunk to a depth of 1280 ft. and developments on this level indicate a greater width of payable ore than on the two preceding levels. The mine has, at present, a reserve

of 2,000,000 tons. The surface plant was enlarged during 1911, resulting in economy of operation. Since 1876, this plant has treated 2,844,596 tons of ore and recovered 89,009 tons of copper. A number of mines, Nymagee, Mount Hope, and Girilambone, did not resume work during 1911, while several others, Crowl Creek, Bogan River, Grafton and Kyloe afforded an increased output. The electrolytic Refining and Smelting Co., at Port Kembla produced 13,132 tons of electrolytic copper, mainly from mattes and ores imported from Queensland.

Queensland.—The principal centers are Mount Morgan and Cloncurry.

The production from the Mount Morgan mine during 1909 was 5799 tons; 1910, 7062 tons; 1911, 6973 tons. The decreased output in 1911 was occasioned by unavoidable delays in effecting the necessary additions to the copper reduction works rendered necessary by the substitution of the basic ore from the Many Peaks mine for the barren iron-stone formerly used. With the new plant in full operation the lost ground will soon be made up. The reserves of auriferous copper ore in this mine have been revalued, and, after allowing for the year's depletion, a slight increase in the former estimate is disclosed. The reserves are estimated at 1,543,000 tons carrying 3.5 per cent. copper and 8 dwt. gold, and 2,070,500 tons carrying 3 per cent. copper and 2.5 dwt. gold.

The extension of the railway to the Cloncurry field has been respon-

PRODUCTION OF COPPER IN AUSTRALASIA.
(In tons of 2240 lb.)

Year.	New South Wales.				Queensland.		South Australia.	Tasmania.				Western Australia.			Combined value.
	Ingots, Matte and Regulus.		Ore.		Copper, Tons.	Value.	Copper, tons. (a)	Value.	Blister copper.		Copper ore.		Copper ore, tons.	Copper (b) tons.	
	Tons.	Value.	Tons.	Value.					Tons.	Value.	Tons.	Value.			
1905	7,962	£522,276	630	£5,127	7,221	£503,547	6,548	£426,511	8,610	£704,287	1,151	£52,939	2,389	.....	£16,266
1906	8,964	781,645	791	7,882	10,077	916,546	8,208	718,609	8,708	862,444	2,234	72,480	7,430	.....	50,337
1907	8,963	716,759	1,135	11,015	12,756	1,028,179	7,931	690,000	8,247	832,691	788	36,975	3,727	1,602	203,376
1908	8,679	496,564	392	6,248	14,961	893,535	5,628	338,000	8,833	603,063	1,185	6,588	2,503	479	57,091
1909	6,857	423,642	109	1,095	14,494	853,196	5,637	334,584	8,638	586,419	1,388	21,619	6,959	833	104,644
1910	8,435	450,491	4,455	35,766	16,387	932,459	5,102	306,120	8,193	553,822	671	13,150	6,309	1,281	95,928
1911	10,618	578,198	1,482	11,904	20,384	1,151,351	5,922	332,500	6,022	385,797	2,286	22,852	9,825	829	78,364

(a) Small amounts of copper ore, and sulphate not included here. (b) Ingots and matte.

sible for a large increase in the copper production of Queensland. This field is remarkable for the extent of its oxidized deposits, an area of 250 by 80 miles being studded with holes from which the working miner has removed parcels of ore; as yet, only in a few instances have sulphide bodies of any extent been found. Consequently it has yet to be demonstrated that the life of the field is likely to extend for any lengthy term. The two largest mines in this field are the Mount Elliott and the Hampden Cloncurry, and during the latter part of the year they contributed between them a monthly output of about 1200 tons of blister copper. The total output of the Cloncurry field in 1911 was 8511 tons of blister copper, valued at £477,346.

The Mount Elliott, in 1911 yielded 5446 tons of copper, 10,494 oz. of gold and 5156 oz. of silver, the average contents of the ore being 13 per cent. copper, 5 dwt. gold and 2.5 dwt. silver. Much of the output was mined in large faces, or quarried, and, being almost self-fluxing, was converted into blister copper at Townsville, at a cost of £25 per ton of copper. No extension of the main shoot has been discovered, and at a depth of 550 ft. (down the dip) the grade of the ore has diminished to 3 per cent. copper, although the vein holds its original width. The Hampden mine yielded 1580 tons of copper, 1535 oz. gold and 20,776 oz. silver from 17,211 tons of ore. The results at neighboring mines point to the advisability of a consolidation of ownership and the erection of concentrating plants. The Queensland Copper Co. closed its smelter at Mount Perry, and will devote more attention to the development of ore bodies in its mines. The Mammoth Copper Co., of Klondyke has decided to build a smelter, which will go into operation by the middle of 1912. The Chillagoe company bought the Einasleigh mine.

South Australia.—Operations proceeded at the Wallaroo & Moonta mines at the same steady pace as in former years. Little work was done at the outside mines, so that practically no custom ores were available for purchase. The yield from all sources was 5922 long tons of copper valued at £332,500. At the Moonta mines 13,771 tons of ore were broken, of which 11,931 tons averaging 4.08 per cent. copper were treated; at the Wallaroo mines, 56,600 tons were broken and 54,647 tons, averaging 3.73 per cent., were treated.

The South Australian government purchased the Paramatta and Yelta mines, near Moonta, for £6000. These mines were worked by a French company from 1903 to the copper slump in 1907, producing about £250,000 worth of copper. The Yelta has a smeltery, connected by a tramway with the Paramatta. The area bought is 1349 acres which, with the exception of the land required for actual operations, will be cut up into farms of 20 to 30 acres.

Tasmania.—The Mount Lyell Co. was practically the only producer, and the output of the company was adversely affected by labor troubles, the smelteries being idle from Sept. 23 to Dec. 11. During 1911 the company produced only 6022 tons of blister copper as compared with a normal output of 8500 tons in previous years. This company normally produces about 500,000 oz. of silver and 10,000 oz. of gold per year. The exploratory work undertaken during the year resulted in bringing the ore reserves up to 1,025,000 tons; the lode opened at the 1100-ft. level of the North Lyell, was of good dimensions, and showed a higher percentage of copper. For the half year ended Sept. 30, 1911, the costs per ton of ore treated were as follows: Mining operations, including stripping charge on opencut ore, $2.22; smelting, $2.32; converting, $0.33; total, $4.87. These figures in the half year ended Mar. 31, 1911, were: Mining, $1.91; smelting, $2.10; converting, $0.28; total, $4.29. The total cost per ton of fine copper was $207.15, of which $94.38 was for mining, $98.68 for smelting, and $14.09 for converting. Figured per pound of copper, the cost was 9.24 cents, of which 4.21 was for mining; 4.41 for smelting; and 0.62 cent for converting. The average selling price was 12.39 cents per pound.

Western Australia.—Copper ores have not been worked extensively, principally because they are not gold bearing. The Mt. Malcolm mine at Eulaminna is responsible for half the production. The ore body is from 5 to 40 ft. thick. The oxidized ores extend down 110 ft., where the ore changes to black sulphides and iron-pyrites. The Phillips River district dates back to gold discovery in 1892; the rocks are metamorphic sedimentaries; the deposits are of two classes: basic and siliceous. They are low-grade but many promise to be large and suitable for concentration. In the West Pilbara field the Whim Well Copper mine is a freehold property, and the chief producer. The flat ore body varies from 2 to 30 ft. thick and extends 3000 ft. The ore is oxidized in a country of talc schist.

Canada.—There is practically no production of refined copper in Canada, so that the output of the metal is represented by the copper contents of smelter products, matte and blister copper, together with the amount of copper estimated as recoverable from exported ores. On this basis the total production in 1911 was 55,848,665 lb., valued at $6,911,831 as compared with 55,692,369 lb. valued at $7,094,094 in 1910, an increase in quantity of 156,296 lb., but a falling off in total value, owing to the slightly lower price of copper in 1911. The total copper contents of ores shipped in 1911 was approximately 67,282,590 lb., being 3,123,189 lb. from Quebec, 21,402,221 lb. from Ontario and 42,757,180 lb. from British Columbia. This record is of special interest

as illustrating the distinction between ore contents and smelter recoveries.

Of the smelter recovery in 1911, Quebec province is credited with 2,436,190 lb. as against 877,347 lb. in 1910. This is altogether from pyrite ores, which are mined primarily for their sulphur contents. Ontario's production in 1911 was 17,932,263 lb. as compared with 19,259,016 lb. in 1910, all being from the nickel-copper ores of the Sudbury district. The production in British Columbia, notwithstanding the failure of the domestic coke supply due to strikes in the coal mines of the Crows Nest Pass district and the consequent shutting down of the Granby smelter for nearly 5 months, shows a slight increase, being estimated at 35,480,212 lb. in 1911 as against 35,270,006 lb. in 1910. The British Columbia Copper Co. operated at larger output, using imported coke, and production from Coast mines, particularly the Britannia and Marble Bay, was specially active. The increased production from these mines more than balanced the falling off at Granby.

The total exports of copper contained in ore, matte and blister, etc., according to Customs Department returns were 55,287,710 lb. valued at $5,467,725, which agrees fairly closely with the record of production. The total imports of copper in 1911 were valued at $4,936,459 and included crude and manufactured copper to the extent of 35,155,550 lb. valued at $4,632,452; copper sulphate 2,191,899 lb. valued at $88,419, and other copper manufactures valued at $215,588.

British Columbia.—At the Consolidated Mining & Smelting Co.'s works, at Trail, West Kootenay, a Dwight-Lloyd sintering plant was put in. This was found to give most satisfactory results when roasting gold-copper concentrate from the Le Roi No. 2 Co.'s Josie mine, Rossland. It has not been so successful, however, with lead-zinc ores and concentrates, with which a second roasting was found necessary. Some changes in the arrangement of the Huntington-Heberlein roasters and converter pots are being made at Trail. The operation of the electrolytic lead refinery there was continued with excellent results. At Grand Forks, Boundary district, the Granby company constructed flumes, bins and an incline trestle (with conveying belts) up to a height of 100 ft., for disposal of granulated slag, thereby making the old dump again available for use for this purpose. Approximate figures indicate that in 1911 the total value of metal products at the Consolidated smeltery was $4,750,000, in the following proportions: Gold, $2,745,000; silver, $715,000; lead, $770,000; copper, $520,000.

A prominent feature of the year was the development by the Granby Consolidated company at its Hidden Creek mine of a tonnage of ore—variously calculated by different engineers at 6,000,000 to 12,000,000 tons—estimated to contain a minimum of 2 per cent. of copper for the

smaller tonnage and 1.65 per cent. for the greater. Some large shoots in this mass of ore give an average assay return of 5 per cent. of copper. Regarding the acquisition of the Hidden Creek property, final payment was made Dec. 15, and the last payment for the minority stock on Jan. 15, 1912. The total consideration for this minority stock was $50,000 cash and 1500 shares of Granby stock. All the stock of the acquired property has been purchased. Full details of plant construction have not been decided upon. Underground development is progressing satisfactorily and one shoot is believed to contain 300,000 tons of 5 per cent. copper ore. The cost of putting the property on a producing basis depends upon a number of conditions, but principally that of desired capacity; it is expected that $1,000,000 will be required for this. The engineers have recommended an ultimate capacity of 2000 tons per day.

During the year ended June 30, 1911, the mines at Phœnix shipped 950,563 tons of ore, as compared with 1,175,548 tons for the previous year. Additional ore to the amount of 1,248,298 tons has been blocked out. Considerable attention was paid to exploration and development of outside properties and deep diamond drilling was done at the Phœnix mines. The property is in a better physical condition than the year previous, and while a less tonnage was produced, the cost of mining and smelting was reduced and more tonnage was placed in sight than was extracted.

A summary of the year's business of the Granby company shows that 17,858,860 lb. of fine copper was produced and sold at an average price of 12.32 cents per pound; 343,178 oz. of silver was sold at 53.5 cents per ounce, and 41,707 oz. gold at $20. The total amount realized was $3,219,271. The total cost at mines and smeltery, including freight, refining, selling, general expenses, and foreign ore purchased, was $3,001,856. The cost per ton was $2.77 and the cost per pound was 11.1 cents, after deducting value of gold and silver. The net profit for the year was $217,415, and net surplus $2,533,304. New construction and equipment at the smeltery amounted to $3843, mine development was 9894 ft.; diamond drilling, 8207 ft.; Granby ore smelted, 959,563 tons; foreign ore smelted, 24,783 tons.

The annual report of the British Columbia Copper Company for the year ended Nov. 30, 1911, shows surplus earnings of $133,929, which left the company with a deficit of $14,002 after payment of $147,931 dividends. The previous surplus having been $731,266, the surplus at the end of the year was $717,264. Proceeds of metal shipments and other income totalled $1,968,159, which compares with 1,466,749 in 1910. Custom ore purchases reserved for contingencies totalled $500,966,

which added to operating expenses of $1,533,263, brought the surplus to $133,929, or $4.52 per cent. on $2,958,545 compared with 8.67 per cent. earned in 1910. Eight months of the year the company was harassed by the coal strike which necessitated the use of Connellsville coke and added greatly to the cost of operation. The production for the year was 9,944,087 lb. of copper, 31,114 oz. of gold and 134,266 oz. of silver. Full description of this company's new smeltery at Greenwood, written by the General Manager, appeared in the *Engineering and Mining Journal*, May 20, 1911.

Ontario.—Practically all of the copper raised in Ontario comes from the nickel mines of Sudbury, where the pyrrhotite is mixed with chalcopyrite. The output in 1911 was about 9000 tons, or slightly less than in 1910. There are many other deposits of copper ore, mainly sulphide, on the north shore of Lake Huron, west of Lake Superior, and elsewhere, but for the present they are non-productive.

Yukon.—The Canadian Geological Survey reports native copper, originating from sulphides, in widely distributed bed-rock near the Alaska line, at Discovery on the White River. One slab was found 8×4 ft.×4 in., weighing 6000 lb., also some smaller ones.

Chile.—The Braden Copper mines, about which so much interest has been aroused, have at last actually begun the production of copper. A full description of the mines and smeltery, written by Pope Yeatman, Consulting Engineer for the company, appeared in the *Engineering and Mining Journal* of December 9, 1911.

The mines are situated in the Andes Mountains in the Province of O'Higgins. They are reached by a state-owned railroad from Valparaiso to Rancagua, 158 miles, and, thence, by the company's narrow-gauge railroad, 45 miles long. The mining area includes 101 claims covering 1214 acres, in addition to other rights-of-way and various water rights. The concentrator and smeltery are at about 6700-ft. elevation and the mines at about 1000 ft. higher.

The rocks of the district are eruptives, mainly andesite, surrounding a body of tuff which represents the old crater of a volcano. The diameter of the volcanic vent is 4000 ft. and the circumference, measured along the outside, is about 2 1/2 miles; the outline is mainly circular. The ore bodies are around the periphery of the crater, in the andesite, at its contact with the tuff. The minerals, comprising mainly chalcopyrite, bornite, chalcocite, and associated carbonate and silicate derivatives, occur in brecciated fissures in the andesite. The ore bodies dip at about 70° toward the center of the crater. Three of these ore bodies have been worked: The Fortuna, the Teniente and the Fortuna No. 4. In the case of many other ore bodies, future development i

planned. The Fortuna ore body is typical of the others; it is lens-shaped in horizontal section, about 1500 ft. long, with a maximum width of 260 ft., and of unknown depth, but ore is developed to a distance of 1421 ft. below the outcrop. The amount of development work done on the various deposits up to the middle of 1911 comprised about 30,000 ft. of drifts, raises and shafts. The mining method to be employed is a modified caving system. Stopes are started by the overhand method and remain filled with broken ore on which the men work. Pillars are left between the stopes and these will eventually be caved and taken out. The ground stands firmly and breaks easily, and no timber is required. Small hammer drills will be used for stoping. No hoisting of ore will be necessary, as the ore bodies are easily reached through adits. For the same reason, no pumping will be required. Greater depths can be reached in a similar manner; for example, a 6000-ft. tunnel starting at a point close to the mill will intersect the Fortuna ore body at a depth of 900 ft. below the deepest ore yet developed.

A careful measurement of the tonnage of ore developed as on July 1, 1911, gives the following figures: Main Fortuna ore body, 9,799,794 tons averaging 2.80 per cent.; Fortuna No. 4 ore body, 212,312 tons of 3.58 per cent. ore; Teniente No. 3 ore body, 62,510 tons of 2.38 per cent. ore; total 10,074,616 tons averaging 2.70 per cent. copper. The future for continuation of ore in depth is considered very promising for a number of reasons related to the occurrence and genesis of the ores.

The concentrator is designed on lines similar to the Utah and Nevada plants, but with coarser preliminary crushing and the introduction of jigs to catch the coarse minerals. The plant will include gyratory crushers and rolls, shaking screens, jigs of the Woodbury type, Chilean regrinding mills, classifiers, Wilfley tables and Frue vanners. The first unit of the mill was put into operation early in 1912. The ratio of concentration will be about 9 into 1. Analysis of the concentrates will be approximately: Silica, 24; alumina, 12: iron, 19; lime, 2; sulphur, 22; copper, 16 per cent. Smelting operations have been carried on to a limited extent and have proved that the above concentrates can be easily smelted at a reasonable cost. A plant has been designed consisting of blast-furnace, 44×180 in., with a 7-ft. ore column, and a Pierce-Smith basic-lined converter for converting matte into blister copper. Experiments have shown that if the converter slag be returned to the blast-furnace the concentrates are almost self-fluxing and require but little additional lime. A briquetting plant is under construction so that concentrates will be briquetted before charging into the blast-furnace. A leaching process for oxidized ores or for roasted sulphide ores is in contemplation, tests on this process having proved entirely

satisfactory. The plan includes the manufacture of sulphuric acid from the sulphide ores, and an electrolytic precipitation plant suitable for handling the copper either from the blast-furnace or from the leaching plant. Power is produced cheaply by an hydraulic plant on the Cachapoal. A power house has been designed and machinery purchased sufficient to supply 6000 kw. per day. This will be delivered to the smelting plant over a line 18 miles long, at 33,000 volts.

The following estimates of cost include all disbursements necessary for operation and conduct of the business. The average assay of the ore is taken at 2.70 per cent.; concentrator extraction, 65 per cent.; smelter extraction, 95 per cent.; yielding 33.5 lb. of copper per ton of ore; average price of copper, 12.5 cents per pound. It is estimated that mining will cost $0.70; transport to mill, $0.10; milling, $0.50; smelting, $0.55; converting, $0.135; refining $0.25; freight, $0.20; commission, etc., $0.04; total $2.48 per ton of ore. This is equivalent to 7.4 cents per pound. of copper yielding a profit of 5.1 cents per pound or $1.71 per ton of ore. It is expected that the leaching process may reduce the cost by one-half to three-fourths of a cent per pound but this will require additions to the plant which will not be authorized before a thorough working test on a large scale is made at the mines, in a plant now being erected for that purpose.

The Poderosa Mining Company, of Collahuasi, Chile, is probably the largest producer of high-grade copper ore in the world. In 1909 its output was 25,782 tons, averaging 18.36 per cent. copper, which realized £181,211. In 1910 the shipments of ore amounted to 23,341 tons averaging 21.5 per cent. copper, and a little over 7 oz. silver per ton, of which the net proceeds, after payment of freight, insurance, smelting charges, etc., were £191,871. The gross profit for 1910 was £55,872 compared with a loss of £24,877 in 1909. The profit in 1910 was derived almost entirely from operations during the last 6 months of the year as during the first half of the year operations were carried on under abnormal conditions, the mine being partially shut down for several months to allow of the enlargement and timbering of the shaft, installation of cages, pumps, etc., the completion of various outside work, and the cleaning out and re-timbering of many of the underground workings. The net profits for the year was £38,471. In March, 1911, an unprecedented rainfall caused the flooding of the mine, which turned out to be more serious than at first believed, owing to the numerous falls of roof of large extent in the workings, which greatly delayed the unwatering and cleaning up of the mines.

Cuba.—Exports of copper ore from Cuba in 1910, amounted to 46,470 metric tons, valued at 417,055 pesos, a notable decrease from the output of 1909, which was 181,547 metric tons, valued at 896,230 pesos

Imports of manufactured copper in 1910 amounted to 1739 metric tons, valued at 809,127 pesos.

Germany.—The amount of copper ore mined in Germany in 1911 was 868,600 metric tons, valued at 24.79 marks per ton, as compared with 925,957 tons of an average value of 25.28 marks in 1910. The great majority of this ore (840,675 tons in 1910) comes from the Mansfeld mines in Saxony, at which over 15,000 men are employed. The smeltery attached to these mines in 1911 produced about 21,000 metric tons of refined copper, and 110,000 kg. of silver. This smeltery employs 2600 men. Other ores, generally of higher grade, are produced in the Rhine district, Westphalia and the Hartz. Imports of ore and matte in 1911 amounted to 23,327 tons, resulting in a total production of 37,452 tons of refined copper in that year, compared with 34,926 tons in 1910. An additional amount of 146 tons of black copper was made in 1911 (1735 tons of 1910). Exports in 1911 included 27,396 tons of ore and matte, 7106 tons of copper bullion, and 19,669 tons of bars, sheets and wire. The average contents of the ore hoisted at Mansfeld is 2.6 per cent. copper and 145 grm. silver per metric ton.

Japan.—The output for 1911, as officially reported to the *Engineering and Mining Journal*, was 52,303 metric tons, compared with 50,703 tons in 1910. Japan thus holds third place in the list, surpassed by only the United States and Mexico. The majority of the output comes from three large mines, Kosaka, Ashio, and Besshi. The first has always had to contend with metallurgical difficulties, and is believed now to have passed its prime, unless new ore bodies are discovered. The Ashio now yields 600 tons of metal per month from ore averaging 3 per cent. copper. Reserves for 10 years are reported. Many improvements and enlargements were made during 1911; a railway was built to the mines, the concentrating works enlarged, mechanical equipment improved, additional hydraulic power secured, and the refinery doubled in capacity. At the Besshi mine, similar improvements were made in the equipment. A new adit 15,000 ft. long was begun, to cut the deposit 1876 ft. below the present workings. The Besshi ore averages 10 per cent. copper, and is so free from contaminating minerals that no refinery is necessary. During 1911, pot-roasting was adopted in place of briquetting, and a unique method of nodulizing fine ore preliminary to roasting was developed by the superintendent of the smelting works. At present, Japan exports about 40,000 tons of copper per year besides supplying a growing home consumption.

Mexico.—The production of copper in 1911, based on returns compiled by the *Engineering and Mining Journal* amounted to 61,884 metric tons, as compared with 62,504 tons in 1910. It thus appears that the

copper mines were unexpectedly fortunate in escaping delays and depredations of the revolution. The great majority of the total output is contributed by three concerns: The Greene Consolidated Copper Co., at Cananea, Sonora; the Moctezuma Copper Co., (subsidiary to Phelps-Dodge, of Arizona) at Nacozari, Sonora; and the Boleo Copper Co., at Santa Rosalia, Lower California.

The report of the Greene Consolidated Copper Co., for 1911 states that the total production was: Copper, 44,897,466 lb.; silver, 1,295,297 oz.; gold, 5892 oz. The production from its own ores was 37,101,119 lb. of copper, 795,160 oz. of silver and 3610 oz. of gold. The company mines produced 751,462 tons of ore at a cost of $2.46 per ton. The important developments of new ore have all been in the limestone areas. These ores are reported to contain from 3 to 5 per cent. copper, from 1 to 4 oz. of silver and from 10 to 30 cents in gold per ton. They are direct smelting ores, highly basic in character. No attempt was made to press production during the year, but, on the contrary, the aim was to curtail. However, on account of the development of basic high-sulphur ores, more silica was needed; consequently operations were resumed at the Henrietta mine, which was closed in 1910, and the Cobre Grande mine was re-opened. Toward the end of the year, work was also started at Puertecitos, with a view to increasing the output when necessary.

During the year, 528,412 wet tons of ore were milled. Of this, 415,199 wet tons were company ore; the concentration ratio was 2.40:1. The total cost for concentration was 72.2 cents per wet ton. An average of 77.69 per cent. of the copper in the ore was saved in concentrates. In the reduction works, 580,060 dry tons of new copper-bearing material were treated at a cost of $2.57 per ton. The total cost of mining and beneficiating, including every cost until the refined copper was sold, was $5.257 per ton of ore.

The report of the Moctezuma Copper Co. for 1911 states that 13,668 ft. of development work was done, of which 9202 was in preparation for ore extraction, and 4466 was purely exploration. Two important new ore bodies were developed. The mine delivered 524,336 tons of ore, and increased its reserves of ore broken and ready to be drawn by 181,000 tons. The mill treated 70,000 tons more in 1911 than in 1910, owing to improved water supply. The operations were as follows: Ore milled, 517,352 tons, assaying 3.171 per cent. copper; concentrates produced, 113,222 tons, assaying 11.932 per cent. copper; tailings assay, 0.558 per cent. copper; recovery, the average of four different methods of computation, 85.19 per cent.; tons ore milled per ton concentrate, 4.569; ore milled per day, actual running time, 1466 1/2 tons.

Technical details of smelting practice at the Boleo mines will be found in the *Mining and Scientific Press*, May 18, 1912.

Newfoundland.—Operations at Tilt Cove for the year ended June 30, 1911, resulted in a net profit of £4733, one-half of which went to the Tilt Cove Copper Co., and the other half to the Cape Copper Co. There were 26,822 tons of ore at 3.42 per cent. copper extracted, and 2255 tons of 8.07 per cent. ore. Reserves are estimated at about 40,000 tons of low-grade ore.

New Caledonia.—About 12,000 tons of ore were made available during 1911 for delivery to Dilah smelters. Ore carrying 7 per cent. copper brings 15 francs per ton, and 8 per cent. brings 20 francs at the mines.

Russia.—The production of copper, as reported by the *Mining Journal*, was 25,909 metric tons in 1911, compared with 21,188 tons in 1910; this was the largest output yet attained, the increase being mainly due to the Kyshtim company in the Urals. The two largest producers are the Bogosloff company, and the Kyshtim company, each of which contributes about one-fifth of the whole copper production of Russia. The latter has recently completed its refinery. The Demidoff is another important mine. In the Caucasus, the leading concerns are the Caucasian Copper Industries, with works at Dzansulsky, and the Caucasian Industrial and Metallurgical Co., at Allaverdsky.

PRODUCTION OF COPPER IN RUSSIA.
(In poods. 1 pood = 16.381 kg. = 36.114 lb.)

Years.	Ural.	Caucasus.	Siberia.	Kirghiz Steppe.	Altai.	Finland.	Various.	Total.
1896	167,574	149,698	.....	1,868	13,239	23,640	.....	356,019
1897	220,783	162,534	.....	3,586	15,427	21,360	.....	423,690
1898	236,863	173,993	.....	2,440	16,341	15,445	.....	445,082
1899	253,610	171,568	.....	5,754	15,292	13,664	.....	459,888
1900	241,148	227,079	.....	11,273	11,322	13,354	.....	504,176
1901	217,063	247,348	.....	21,993	13,193	17,311	.....	516,908
1902	279,135	213,273	.....	25,238	7,431	13,231	.....	538,308
1903	265,116	262,919	.....	17,902	7,546	10,126	.....	563,609
1904	265,915	296,666	.....	30,513	7,344	.....	.....	600,438
1905	225,800	223,800	67,000	.....	.....	.....	53,900	570,500
1906	288,600	232,300	40,500	.....	.....	.....	74,700	636,100
1907	457,904	310,244	68,957	.....	.....	.....	65,253	902,358
1908	522,584	296,379	155,116	.....	.....	.....	54,616	1,028,695
1909	530,778	391,200	150,582	.....	2,940	.....	57,984	1,126,584
1910	565,957	470,970	196,019	.....	3,480	.....	57,017	1,293,443
1911	808,982	486,964	228,007	.....	1,709	.....	55,979	1,581,641

The output of the Spassky company, in the Akmolinsk district of Siberia, is increasing, while that of the Yeniseisk company is diminishing. The Atbasar Copper Fields, Ltd., developed a large tonnage, and proposes to erect a smelter in 1912, to go into operation in 1913.

The Caucasus Copper Co., which has spent over $8,000,000 of English and American capital in the development of mines and plant situated 50 miles from Batum, has recently been able to show a profit on its

operations. The copper minerals are disseminated in an extremely fine state of division; reserves now opened are estimated at 5,000,000 tons, carrying 3 1/4 per cent. copper. The present plant treats 500 tons per day, but extensions are now under construction, with the ultimate view of treating 1500 tons or more per day. The present concentrates average 8 1/2 per cent. copper, 37 per cent. silica, and 28 per cent. sulphur. After roasting, the ore is smelted in oil-fired reverberatory furnaces, and the copper is all sold in Russia, under a protective tariff of 5 rubles per pood (7 cents per pound). In point of tonnage developed, this ranks with the large mines of the world. Description of the mining and concentrating methods at this mine will be found in the *Mining and Scientific Press*, of Jan. 27, 1912.

The Kyshtim mines, controlled by the Kyshtim Corporation, of England, are situated 25 miles south of the town of Kyshtim, on the Trans-Siberian railway, to which they are connected by rail. Four important mines have been developed, and a smelter erected at them; also an electrolytic refinery at Kyshtim. One of the mines yields barren pyrites, which commands a good price for its sulphur. The others yield ore of the following average composition: Gold, 0.1 oz.; silver 1.0 oz. per long ton; copper, 3 to 4 per cent.; sulphur, 40.8 per cent.; silica, 12 per cent., iron, 34.1 per cent.; lime, etc., 3 to 6 per cent. Reserves are estimated at nearly 2,000,000 long tons. The present, or Karabash, smelter was finished in 1911. Technical details will be found in the *Engineering and Mining Journal*, June 22, 1912.

Spain.—The production of copper in Spain and Portugal is estimated by Merton & Co. at 52,878 metric tons in 1911, and 51,100 tons in 1910. The official figures for 1910 are: Refined copper, 85 metric tons; matte, 2684 tons; precipitate, 14,056 tons. The Rio Tinto Company is the largest single producer in Spain, the output from which mines for a number of years is given in the following table, in long tons:

OUTPUT OF RIO TINTO MINES.
(In Long Tons.)

Year.	Pyrites for Shipment.	Ores for Local Treatment.	Total Mined.	Average Copper Contents.	Copper Produced at Mines.	Pyrites Sold.	Washed and Other Sulphur Ores Sold.
1900	704,803	1,189,701	1,894,504	2.744%	21,120	665,967	.....
1901	633,949	1,294,827	1,928,776	2.627%	21,100	641,935	.....
1902	627,967	1,237,322	1,865,289	2.517%	21,659	595,092	.....
1903	688,919	1,229,619	1,918,538	2.390%	21,565	667,748	118,174
1904	672,344	1,276,475	1,948,819	2.340%	21,218	663,744	157,810
1905	627,336	1,202,768	1,830,104	2.363%	19,530	660,724	308,184
1906	655,328	1,268,388	1,923,716	2.411%	21,287	632,307	477,843
1907	641,858	1,265,090	1,906,948	2.417%	21,251	607,944	619,814
1908	604,275	1,115,610	1,719,885	2.265%	24,256	589,815	668,477
1909	604,799	1,184,188	1,788,987	2.349%	24,364	600,946	569,604
1910	637,020	1,509,945	2,146,765	2.097%	22,790	578,443	683,605
1911	649,215	1,536,390	2,185,605	2.144%	21,880	662,259	841,964

The amount of copper in the pyrites sold was 10,663 tons in 1910 and 11,505 tons in 1911, making the total copper production of the company in those 2 years 33,423 tons and 33,385 tons, respectively.

The Cordoba Copper Co. operates the Cerro Muriano and North Cerro Muriano copper mines, 10 miles northeast of the city of Cordoba, in the south of Spain. The ore deposit is difficult to follow and work, and the smelting and concentration are not simple problems. Operations have been substantially extended during the past year or so, and much better results have been obtained. The amount of ore raised during the year ended September 30, 1911, was 86,285 tons, as compared with 61,887 tons the year before, and the reserve has been increased by 66,495 tons, to 199,398 tons, containing 3.59 per cent. copper; an increase in grade of 0.31 per cent. The erection of converters has been completed, and the output is now in the form of blister copper. Of the ore raised and treated at the company's own plant, 6630 tons were hand picked and sent direct to the smelter, 2240 tons of rich fine was screened and sent to the sintering pots, and 63,609 tons sent to the concentrators. The last produced 8238 tons of concentrate assaying 11.1 per cent. copper, which was sent to the sintering pots and briquetting machines; and 9455 tons of middling, assaying 1.47 per cent. which was sent to the Murex plant, where 936 tons of concentrate was obtained averaging 12.2 per cent. copper. The remaining ore was hand picked and dressed by contract, and the products sent to the smelter and sintering departments. As the sintering pots were being worked to their maximum, other fine had to be treated by briquetting, the binding material being slime. The smelting plant treated 10,122 tons of picked ore, 7886 tons of sintered material, and 2427 tons of briquettes, and yielded 3439 tons of matte averaging 57 per cent. copper. By means of the wet concentrators and the Murex plant, the percentage of recovery is 75; the coarse tailing averages 0.65 per cent., the fine tailing 1.17 per cent., and the slime 1.53 per cent. copper. The magnetite used in the Murex process is not subsequently eliminated, because iron is required in smelting, and the magnetite in this case costs no more than hematite, which would have to be purchased in any case. The oil used is a thick coal-tar product, which, if any tar refinery were a customer for it, would be more valuable for that purpose than for the Cordoba company. The tailing will be treated by the process in the course of a short time, and the recovery will then be greatly improved.

Sweden.—The Stora Kopparbergs Bergslags Aktiebolag, whose activities date back to the fourteenth century, made net earnings in 1911 of $872,720, of which 16.1 per cent. came from copper, 20.2 per cent. from iron and steel, and the remainder from timber and paper pulp. The

amount of pyrites mined in 1911 was 21,592 metric tons, from which were made 326 tons of copper sulphate, and 6159 tons of sulphuric acid.

United Kingdom.—The principal mines are in North Wales. The precipitate from Anglesey sold for over $150 per ton for a little less than 200 tons. The copper from Cornwall and Devonshire, a by-product, was almost negligible. Two new mines began operations last year; one at Alderley Edge, Cheshire, and the other in the North Riding of Yorkshire. The output for 1911 was 4500 tons.

The Briton Ferry smelting works in Wales, belonging to the Cape Copper Co., were in constant operation during 1911, and an electrolytic refinery will soon be ready.

In Ireland, a company is undertaking development of the Ardtully mines, between Kilgarvan and Kenmare, County Kerry. Assays are reported to show 22 to 49 per cent. copper, and returns on the first ton smelted were on the basis of 15 per cent. copper and 15 oz. silver. Copper deposits previously developed in Ireland have been of low grade.

Metallurgy of Copper in 1911

By L. S. Austin

Trend of Improvement in 1911.—There has been a remarkable development of practice in basic converting during the year 1911, the methods of lining and construction having been previously worked up. In many cases the old standard shells have been lined with basic bricks. Much of the old equipment used in acid converting has been rendered useless so that one feels that a plant depreciation allowance of 10 to 15 per cent. is justified. From this point of view many former installations may have been too permanent and expensive. In reverberatory practice improvement has shown itself in the increased use of fuel-oil and in increased tonnage per furnace. It is recognized in blast-furnace work that the charge should be of uniform constitution, and to this end increased care should be taken in bedding ore. Where a given plant is using both the blast and reverberatory furnaces, the precaution is now taken of screening, so that the dust-free ore goes to the blast-furnace, the fines to the reverberatory.

Handling Ores at the Reduction Works.—To take ores as they are delivered from cars, to sample and store them, and finally to deliver them to the furnaces is now effectively accomplished by troughed conveying-belts. The system, as installed at Cananea, Mex., has proved to be flexible in use as well as reliable and gives a uniform product for the furnace. Three or four years of use have shown that these belts are durable. For delivering ore 1/4 mile to roasters, the belt has proved

to be better than the electric tram. The cost of conveying may be estimated at 4 cents per ton.

Handling of the Fumes from Smelting Works.—These fumes may be disposed of completely by removing the contained dust, the remaining sulphurous acid gas being used for the manufacture of sulphuric acid. This implies a market for the acid and hence is inapplicable in remote districts. In such places the dust has been removed and the clear fume dissipated into the air from the top of the stack. At Anaconda, Mont., the dust is largely settled out, the arsenic in the fumes condensed and recovered and the sulphurous gases allowed to escape. To ensure settling it has been found that a low velocity of the escaping gases of 4 to 5 ft. per second permits of the settling out of the dust. When the velocity is as high as 10 to 15 ft. per second, the eddy currents in the flue keep the dust so stirred as to prevent satisfactory settling. Beside sulphurous acid a little sulphuric acid forms. This latter has been neutralized by mixing with the flue gases the fumes from a furnace roasting zinc ore. The zinc oxid volatilized in roasting neutralizes the sulphuric acid.

Copper Blast-furnace Development.[1]—E. P. Mathewson gives a history of the development of the blast-furnace from the earliest times. The earlier forms show a hearth or crucible blown by the various hand appliances. Finally, with the introduction of power blowers, we find the hearth heightened by a shaft. The cross-section of the furnace has worked itself out in two directions. Either it was circular and for increased capacity of larger diameter and having more powerful blast, or to increase capacity without increase of blast-pressure was accomplished by lengthening the furnace as in the Raschette type. The tuyères, at first introduced at the back of the furnace and opposite the breast, were finally placed on all sides in the round furnaces and on the two long sides in the rectangular ones. Mr. Mathewson's own latest invention is a furnace of exaggerated length and of great capacity.

"The Practice of Copper Smelting" by E. D. Peters, McGraw-Hill Book Co., New York, 1911. "This book was written to replace Dr. Peters' former one—Modern Copper Smelting." It includes studies of actual operations at representative plants, and investigations, comparisons, and discussion of the results attained, the whole having been brought to the date of publication. The subject-matter includes, ores and their sampling, methods of copper extraction, the behavior of ores at high temperatures, the roasting and smelting of ores, blast-furnace smelting, reverberatory smelting, copper converting and refining, the disposal of flue dust and smoke, a discussion of copper slags and certain miscellaneous information.

[1] *Eng. and Min. Jrnl.*, XCI, 1057.

Roasting and Sintering

An Improved Rake and Arm for McDougal Roasting Furnace.[1]—A device, designed by Wm. Kelly and H. N. Thompson at the Washoe smeltery of the Anaconda Copper Mining Co., is shown in Fig. 1 which gives a cross-section, a plan, a side view and an end view. On the cross-section two ledges are shown where the ore lodges and forms a non-conducting cover. The rakes or blades are slid on the arms and the ore cannot pack in the joint between the two; hence the blade may be applied or removed with but little trouble. It has been in satisfactory use for over 2 years.

Blast-furnace Practice

Slag-disposal System at the Granby Smeltery.[2]—Central storage-bins for the granulated slag have been built below the level of the top of the present slag-dump and the slag is sluiced to these by launders. After dewatering, the slag is taken by an inclined troughed belt-conveyor

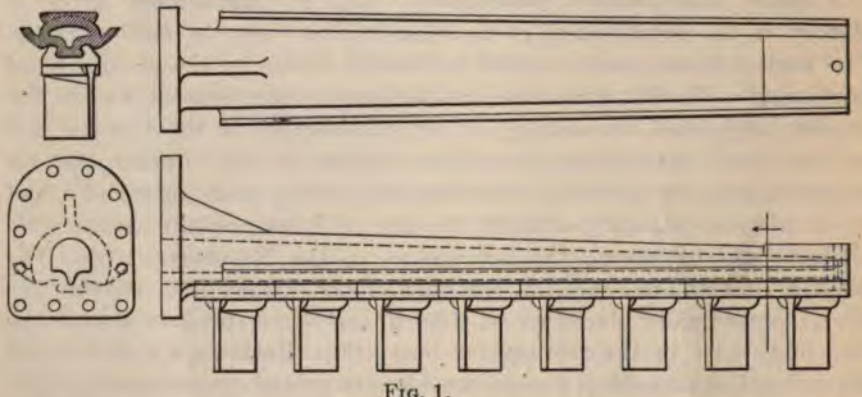


Fig. 1.

to an elevated trestle and discharged on a movable tripper. The space beneath the inclined elevator will be first filled, then that below the elevated trestle. An extension belt will be used for dumping beyond the end of the level trestle when the ground beneath it has been filled. It is expected that the system will provide for the disposal of eight to ten million tons of slag.

Blast-furnace Smelting with Oil.[3]—Thomas Kiddie gives the results of tests with fuel-oil at Van Anda, Texada Is., B. C. The furnace charge consisted of Boundary ore, iron ore and slag. Beginning at 11

[1] *Min. and Eng. Jrnl.*, XCI, 455.
[2] *Met. and Chem. Eng.*, IX, 415 X, 113.
[3] *Eng. and Min. Jrnl.*, XCII, 434.

A. M. the burners were used until a sufficient heat was attained, then two more, so that by noon slag began to flow. By 2:30 P. M. the slag was hot and was flowing at the rate of a pot per minute. He computes that 60 gallons of oil were used at starting, and during the 2 1/3 hours of running, 157 gallons were needed, or 14.6 gallons per ton of charge, equal to $0.438 per ton. He estimates that with certain changes this cost may be reduced to $0.30 to $0.35 per ton, and that a further saving of $0.09 per ton can be made in labor. The tests appear to be all too brief to establish these results.

The Use of Fuel-oil for Smelting.[1]—E. H. Hamilton gives his experience in the use of crude oil and residuum for retorting, assay furnaces, roasting, reverberatory smelting and even for use at the blast-furnace.

He gives the calorific efficiency of California crude oil, an average sample of numerous cars, at 10,383 calories, a lower figure than those generally presented. He estimates that one barrel (42 gallons) was equivalent to 1 ton of good coal.

In 1890 at Pueblo he used oil continuously for roasting in Brown-O'Hara roasters and obtained a properly nodulized or agglomerated product. In 1904 he used California crude oil for roasting copper ores in Edwards roasters. In this case the ore was nearly roasted by its own heat, and when the temperature began to get too low, the oil jets were turned on for a few minutes.

In the reverberatory work the silica in the slags ranged from 45 per cent. downward; in fact from his previous experience at Anaconda, with a slag of 45 per cent. SiO_2, he was confident that these furnaces could not have been run with coal-firing, and that at Anaconda with oil-firing he could have smelted 300 tons daily. He found that 19 to 20 per cent. of the heat was absorbed in smelting and 50 per cent. was taken up by the boilers. In a furnace 100 ft. long oil-firing could be regulated to produce a white heat and complete combustion before reaching the boilers. For heating the oil at the tanks and while it runs through the pipes to the furnaces, as well as for running the oil pumps, 7 per cent. of the steam produced by the boilers is used.

He once employed oil when there was a shortage of coke for the blast-furnace and kept running for several days. To ensure effective burning, he was obliged to first heat the blast at the tuyère where the oil was introduced. It was possible to see the flames from the oil in two or three tuyères on each side of the oil-burning one. The principal difficulty was found to be in the slagging of the tuyère.

Blast-furnace Tops.[2]—N. H. Emmons gives in his paper particulars of the development of closed furnace tops as applied to the blast-furnaces

[1] *Eng. and Min. Journal*, XCI, 224.
[2] *Trans. A. I. M. E.*, XLI.

of the Tennessee Copper Co., when the gases from the furnaces were used for sulphuric acid making. To this end the furnaces, putting through 500 tons daily, were run with hot tops. The average temperature in the down-take was 633° C., and at the time of charging it dropped 50 to 100° C. The escaping gases carried 5 per cent. of SO_2. An interesting feature of the furnace operation was that with faster running the temperature of the escaping gases was lower. The gases were passed to Glover towers, and at that point the temperature was found at times to be higher than at the furnace. This was explained by the assumption that the gases, as they escaped from the surface of the charge, contained unconsumed sulphur as well as CO, and that as air leaked into the closed top, it burned those gasses, thus developing heat.

After trying various tops made with structural steel and brick, a type of closed top was evolved of 13-in. brick walls bound with heavy angle-irons at the corners and upright plates or posts between the doors, while the lintels of the cast-iron doors were 20-in. I-beams, and these were surmounted by a semi-cylindrical 1/4-in. boiler-plate top and down-take. All metal parts were so arranged as to be exposed to the cooling of outside air and were not buried in the brickwork. The side doors open inward and are operated by a simple two-throw weighted switch stand.

Blast-furnace Matte Smelting of a Low-grade Ore.[1]—C. A. Heberlein gives his experience in smelting an ore of the following composition: SiO_2, 31.4 per cent.; Fe, 20.10; Al_2O_3, 5.5; S, 11.7; CaO, 14.6, and Cu, 0.28 per cent.

The furnace was 36×120 in. at the tuyères with height to feed floor of 7 ft. It was run by an overflow trap spout to a forehearth of 6 ft. inside diameter. The ore, regarded as self-fluxing, needed 7 per cent. of coke and gave a concentration of ten into one. The blower delivered 6300 cu. ft. of air per minute at 18 oz. pressure to the square inch and the furnace took 56,000 cu. ft. per ton of ore treated while running at the rate of 160 tons in 24 hours or 5.3 tons per square foot of hearth area. The slag assayed SiO_2, 45.6 per cent.; FeO, 23.9; CaO, 21.5; Cu, 0.095 per cent. with Ag 2.1 oz. per ton and a trace of gold. At the same time there was produced a matte carrying 3.08 per cent. copper with 10.8 oz. of silver and 0.96 oz. of gold per ton. This indicated a loss of Cu 23.7 per cent.; Ag 13 and Au 3.3 per cent. It would appear then that when copper is present to so little as 0.3 per cent. it is still able to make a fair extraction of the silver and a fairly close removal of the gold. The furnace to do the most satisfactory work should have been at least 10 to 12 ft. high instead of 7 ft. The corrosive matte produced was hard on the water-jacketed cast-iron spouts and on the tap-jacket.

[1] *Eng. and Min. Jrnl.*, XCI, 617

Electric Smelting

Electric Smelting of Copper Ores and Blast-furnace By-products.[1]—Von. J. Schilowski gives the results obtained in the electric smelting of a few pounds of ore in an apparatus consisting of a crucible fitted with carbon electrodes to which was attached a small settling and cooling chamber.

He smelted 8 lb. of ore containing SiO_2, 20.3 per cent.; Cu, 10.1; Fe, 27.1; Al_2O_3, 9.4; As, 9.3 and S, 15 per cent. This yielded a matte of Cu, 21.4 per cent.; Fe, 50.3 and S, 28.4 per cent.; also a glassy easily-flowing slag containing only 0.15 per cent. Cu. The arsenic was sublimed as arsenic and arsenious sulphides (As_2S_3 and As_2S_2) and the condensed resultant contained 60 per cent. arsenic. The current employed was equivalent to 27,500 kw. hours per ton of copper produced.

In a second experiment, using 27.5 lb. of charge of two-thirds roasted and one-third raw ore, a matte of 53.1 per cent. Cu was yielded with a slag containing 0.6 per cent. Cu. Some speiss was produced which separated at the bottom of the crucible. Repeating the work, a slag of 0.45 per cent. Cu resulted. In both these latter experiments the power consumption averaged 17,300 k. w. hours per ton of copper.

We conclude that with a lower grade charge the slag is cleaner, but the power consumption is greater than where a high-grade matte is produced.

Reverberatory Practice

Reverberatory Smelting.—With an improved draft, which may be as much as 1.6 to 2.0 in. water it has been found possible to burn coal more rapidly, and consequently to increase the working temperature of the reverberatory furnace. This has worked in two ways: first, by increasing tonnage and secondly by enabling more silicious slags to be produced—slags that are at once more infusible and more viscid than those of lower silica contents. Thus, where 38 to 40 per cent. silica was carried, now at the higher temperature we find slags of 45 per cent. SiO_2 at the Garfield plant and 43 per cent. at the Steptoe smeltry. This higher silica makes for reduced operating cost and cost of flux.

The method of claying or fettling at Cananea (Mineral Industry, XVIII, 185) is now criticised, first because cold air enters the furnace when claying is undertaken, and secondly because much of the silicious fettling floats away and escapes with the slag.

Oil Fuel at Ely Smelter.—Preliminary excavation has been completed for the oil storage reservoir which will hold the reserve supply for the reverberatory furnaces at the Steptoe smelter at McGill, Nev. The excavation nearly circular in form, is 125 ft. in diameter. The fall of

[1] *Metallurgie* (1911), 617.

100 ft. from the reservoir to the furnace building will deliver the oil to the reverberatories under a gravity pressure of 46 lb. to the square inch. This will be increased by pumping to a pressure of 85 lb. to the square inch at the burner. Numerous changes have to be made in the furnaces to allow of the use of oil for fuel. The storage capacity of the reservoir now being built is figured at 22,000 barrels. Other tanks and reservoirs will be built as the use of oil increases at the plant, until a storage capacity of approximately 55,000 barrels is reached.

Developments of Smelting Practice at Cananea.[1]—In addition to the description of smelting practice at Cananea (MINERAL INDUSTRY, XVIII, 185), the manager, Dr. L. D. Ricketts, gives additional points hereunder presented:

Preparatory to roasting, the ores are bedded, the mixture from February to July, 1911, containing 76.1 per cent. concentrate, 22.1 per cent. of fine sulphide ores and 1.8 per cent. of limestone. The composition of the bedded charge was Cu 5.2 per cent., Fe 28.4 per cent., S 29.9, SiO_2 23.6, and Al_2O_3 3.7 per cent. Upon roasting, this yielded calcines containing Cu 6.3 per cent.; Fe 34.5; S 7.7, SiO_2 28.6, Al_2O_3 4.4 per cent. For this self-fluxing charge there are two beds of 3000 tons each. The ore is received on belts from the railway bins, sampled on its way to the beds, and is bedded. In quantity as needed, it is reclaimed from the beds and conveyed by belt to charge-hoppers over the McDougal roasters. These hoppers are large enough to hold 18 hours' supply, so that reclaiming can be done on one shift.

Of the two reverberatory furnaces the second, recently constructed, uses 12-in. I-beams for buck-staves and has an arch 20 in. thick. At the same time, the waste-heat boilers have been increased from three, used for the first reverberatory, to eight provided for the two, these boilers having an average capacity of 235 hp. each. This enables one boiler to be cut out for repairs as needed. In operating the two reverberatories with the boiler plant specified, the temperature of the waste gases leaving the boilers is 287° C., and after passing through the economizers 176° C. Besides this four banks of Green economizers, with a total of 1152 tubes, have been installed. The reverberatories are oil-fired, using an oil that, in the independent boilers at 150 lb. pressure per square inch, evaporates 14 lb. water per pound of oil, or allowing for steam used in atomizing the oil, 13.5 lb. per pound of oil. Thus far, the new reverberatory has needed but few repairs, the most serious of them due to a "runaway" or accidental escape of the contents of the furnace, which cut off some of the buckstaves. The old reverberatory has needed a great deal of repairing because of its thinner (12 in.) arch.

[1] *Eng. and Min. Jrnl.*, XCII 673.

Of repair expense $8000 applies to resetting three waste-heat boilers that had been in service for five years, and to certain changes in the economizers. It was found that the fine-dust settling on the economizer tubes is easily blown off, but that the automatic scrapers commonly used rub it into a paste, which makes a thick tough insulation on the tubes, and finally builds up until so thick that the scrapers will not operate. The scrapers have accordingly been removed and doors which can be readily opened, have been inserted at the sides of the economizers. Every day the dampers are set to cut out each economizer in succession and the fume is blown from its tubes with a jet of compressed air. It takes less than an hour to clean the four economizers, or 15 minutes per economizer per day. It has been found to be most profitable to burn no more than eight barrels of oil per hour per furnace. With this quantity repairs are less, and it is hoped to cut the fuel consumption to 0.8 barrel per ton of charge—in fact, this has been accomplished over short periods. Average practice, however, indicated the consumption of one barrel per ton of charge. This higher result has been due to shut-downs for repairs since, in again starting up, it has been found that 500 barrels of oil are used before the furnace is in good working condition and able to take its normal charge. Tests made upon the furnace gases indicate but little flue-dust loss either from the reverberatory or McDougal furnaces. The blast-furnace loss is notable though a fairly good dust-chamber is provided. This loss hardly warrants heavy expenditure for a new dust-chamber, especially since it is intended to remove the fines from the ore before they go to the blast-furnaces, these fines being reserved for treatment through the roasters and reverberatories.

In the converting department experiments have been made by blowing through one tuyère of a converter a limited amount of fine high-grade copper concentrate. Of this material, principally copper glance, silica and alumina, about 50 per cent. will pass a 200-mesh screen, and already several thousand tons have been smelted. By treating it in converters one step in smelting would be removed, and since the matte is low-trate, so thatwould be abundant heat to melt the sulphides in the concen-grade, there all the converting would be from the stage of white metal.

Tests were made both before and after installing a suitable steel dust chamber. This chamber, 104 ft. long, of cross-section to give a velocity of the gases of not more than 3 ft. per second when all five converters were running, was connected to a stack common to all the converters. For cheapness and for better radiation of the heat the flue was of steel, and the whole installation cost $25,000. The losses were determined by passing a portion of the flue gases through woolen bags with the following

results: Before installing the flue chamber, and when not blowing concentrate, as the result of 13 tests of 5 hours each, the solids escaping were 196.23 lb.; the copper contents 12.66 lb., the silver contents 1.498 oz., the gold contents 0.00512 oz. After installing the dust-chamber and while blowing concentrate into the converters, the solids escaping were 140.3 lb., the copper contents 4.19 lb., the silver contents 0.696 oz.; gold contents 0.00107 oz. per ton; this being the average of nine tests of 9 hours each. During the first tests 74 converter hours per day were averaged, and during the second 97 converter hours per day.

The Burning of Reverberatory "Ash."[1]—Lindsey Duncan gives the result of his experimental run when burning the ash or droppings from the grate of a reverberatory furnace at the Steptoe plant of the Nevada Consolidated Copper Co. This imperfectly burned or coked material contained fixed carbon 68.9 per cent., volatile constituent 5.9, ash 19.2, moisture 6 per cent., corresponding to an efficiency of 6340 calories. On a test upon 11,385 lb. of this ash there was evaporated 57,241 lb. of water giving 5.03 lb. of water per pound of "ash" burned. The ash burned with a short blue flame, and for rapid combustion a forced draft in conjunction with light and frequent firing was found necessary. The drawbacks to its use were: that it reduced the capacity of the boiler 40 per cent. as compared with coal, that it clinkered so much that fires had to be cleaned four to six times per shift, that to force the boiler, coal must be used, and finally that 50 per cent. more labor is needed than when burning coal.

The Use of Liquid Converter-slag in Reverberatory Practice.[2]—Dr. Ricketts is of the opinion, that when pouring irony converter slag into the reverberatory near the back or bridge-end of the furnace, it settles through the lighter silicious slag produced by the furnace. It spreads out upon the matte and will travel far toward the front before it mixes with the lighter slag above. The reverberatory slag, if very silicious and infusible, would be rendered but little more fusible by the basic slag beneath, and so the latter would give but little assistance in fluxing the lighter kind. If, however, this converter slag can be added to silicious ore to form a self-fluxing mixture, this can at once be added to the furnace-charge at a temperature say of 650° C.

In 1908 experiments were made at the Copper Queen Reduction Works with a view of sintering converter slag in mixture with flue-dust in a drum in shape of a truncated cone. This produced a remarkable sinter and it is this solid mixture that it is proposed to add to the reverberatory charge.

[1] *Eng. and Min. Jrnl.* XC, 1302.
[2] *Eng. and Min. Jrnl.*, XCII, 695

Converter Practice

Control of Converter Temperatures.[1]—In operating basic-lined converters, referring to the question of controlling the heat through the addition of cold materials, Dr. L. D. Ricketts says: "It occurred to me that the generation of the heat in a converter containing a large mass of matte is a function of the speed of oxidation, and that therefore if oxidation were conducted at a slower pace during the period when intense heat is apt to be generated, we should have a ready method of controlling heat." To do this, he advises the use of large converters, and a constant-pressure blowing-engine with an individual throttle to each converter.

Basic Converters.[2]—In the Pierce-Smith type, the converter had at first one opening or throat for both skimming slag and pouring copper. As now built there are two openings, one for skimming and a second some 2 ft. below for pouring copper. The throat, formerly at one end, has in one case been changed to be near the middle. With material like dry concentrate, which spreads readily when charged, the location of the throat does not matter, but in the case of silicious ore, of 1-in. size and less, some is apt to adhere to and build up on the end near the throat. This change has been so favorable that the plan is now to alter all the converters as rapidly as practicable. It has been found possible to increase the tuyères from 32 to 37 in number; and yet with the central outlet the converter works as quietly and faster than with the older plan. Thus, with 32 tuyères, blowing from white-metal to copper, 10 tons of copper per hour was the capacity; with 37 tuyères it becomes 14 to 15 tons. A urther change has been the introduction of removable pipe-tuyères in place of the fixed square tuyères at first used. If a tuyère becomes slagged or filled with copper it can be removed and replaced by a new one; no longer is it necessary to drill out the tuyères.

At several plants, equipped with acid-lined converters, these have been changed to basic-lined ones by the substitution of magnesia brick. The shells are of the regular 12 1/2 by 8-ft. size with a capacity of one-third to one-fourth of those of the Pierce-Smith type. The ganister used in the narrow belt at the tuyère-region, 18 to 20 in. thick in the radial direction, is often made of a mixture of lime and oil with the magnesia; it has been found, however, that the addition of 5 per cent. sodium silicate is more satisfactory. It sets slowly and forms a stable and coherent mass when hard. The drying of the mass must be done slowly else it will scale off, though even then it stands well.

Basic converting will in the future result in a modification of design

[1] *Eng. and Min. Jrnl.* XCII, 695
[2] *Eng. and Min. Jrnl.*, XCI, 707, 964; Mineral Industry, XIX, 192

of the converter plant. Matte and slag will be charged by less expensive methods than by a traveling crane, and the speedy change and transfer of shells will be no longer required. There will be no crane-men exposed to the sulphurous gases escaping from the converters. In plants designed for acid converting and now changed to basic, one is struck by their deserted look. The lining department is idle, and because of more continous operation, two-thirds only of the converters are running.

Basic Converting Practice.[1]—Ore carrying as little as 20 per cent. excess silica can be used, the lower limit being due to the large quantity needed for fluxing the forming iron oxide. Such a quantity tends to cool off the converter and to fill it up. As much as 8 tons has been successfully added at a time to a 26-ft. Pierce-Smith converter, though it is a question whether this is not too much.

The slag should be skimmed clean otherwise oxidation on the succeeding blow proceeds slowly. Thus, on one charge when skimming was clean, the rate of oxidation was 163 lb. per minute and this dropped to 95 lb., 55 lb., and 28 lb. when all the slag possible was not removed. Following the succeeding blow, the slag was carefully removed when the rate increased to 106 lb. per minute, which was as much as could be expected of a charge approaching the white-metal stage. Slower oxidation also takes place if a large amount of fluxing ore is added at once.

The slag must be thin since it has to be poured without rabbling. If the slag is viscous, due to too much silica, then the converter is to be turned back and blowing resumed. With a 3-ton charge of ore running 70 per cent. silica, the time of blowing is 40 minutes, though that varies and is to be learned by experience. If the slag is viscous, due to being too basic, then a further addition of silica and further blowing will correct the trouble.

In blowing the successive charges of matte and silicious ore, the converter eventually contains a charge, which has been brought to the stage of white metal, after which begins the difficult operation of blowing to blister copper. Only the exact amount of silicious flux must be added, so that when the slag is high, a thin liquid slag results. If too much flux is added the slag is too thick or viscous, if too little some iron remains unoxidized. Both cases mean too much slag in the converter, the charge runs hot, foaming begins and material is shot out. To a certain extent this can be corrected by charging cold copper scrap, but this delays operations. There is a certain temperature at which the charge works fastest; if too hot foaming results, if too cold the tuyères tend to clog and

[1] *Eng. and Min. Jrnl.*, XCI, 707, 964; MINERAL INDUSTRY, XIX, 192.

so hold back the air; both of these conditions resulting in slower working.

In the large basic converters it is hard to tell the finish by the flame or by the copper particles in the flame, and accordingly the rod method is used. To do so the punching bar is quickly inserted at a tuyère and withdrawn, and the appearance of the copper now sticking to the bar, shows how near the charge is to the finish. When finished this copper should be free from white-metal, full of minute pin-holes, and usually slightly blistered.

These large converters have produced over 3000 tons per lining on matte of less than 40 per cent. copper, three runs giving 3170, 3276 and 4031 tons. The corresponding charges were 31.4, 40 and 33 tons on a converter having a nominal capacity of 75 tons.

The quantity of magnesite used up for converter lining was at first 11 lb. per ton of copper produced, later the loss dropped to 9.7 lb. per ton, and this figure has been further improved.

Taking a summary of basic converter performance during November 1910, we find: To produce 1 ton of copper there was used 2.45 tons of matte, 0.31 tons concentrate and 0.56 tons of silicious ore. This yielded, besides the 1 ton of copper, 1.67 tons of slag, indicating a loss of nearly 20 per cent., presumably of sulphur and moisture. Of copper there was produced 32.3 tons per charge or 45 tons per day. Of the total time 33.4 per cent. was occupied in blowing matte to white-metal, 27.8 per cent. in blowing white-metal to copper, while the waits between blows needed for charging and skimming was 27.8 per cent., making the actual operating time per run of 83 per cent. Of the remaining 17 per cent., 8 per cent. was occupied in pouring copper and 9 per cent. in waits between charges. The average number of blows per charge, that is the number of times the converter was turned up to blow after skimming, was 12.8 and the duration of these times varied from 10 to 70 minutes with an average of 45 minutes.

For this month 107.5 hp.-hours was required and there was delivered 4.94 tons or 147,600 cu. ft. of air per ton of copper, at the rate of 9700 cu. ft. per minute and at a pressure of 9.17 lb. per square inch.

Introduction of Ore into the Converter.—To blow fine ore into the converter through the tuyères was attempted years ago, but by using a separate pipe and with a higher pressure than that of the blast it is now successfully driven in. A sufficient supply can be introduced at one or two of the tuyères to supply the requirements for silica. Lately, concentrates have been introduced in the same way, but this seems to be a rather expensive way of treatment as compared with ordinary furnace smelting.

Smelteries

Plant of the Mammoth Copper Mining Co., Kennett, Cal.[1]—This plant is described in Mineral Industry, XVII, 252; further operating details are as follows:

The ore from the Mammoth Mine contains about 4 per cent. of zinc, which is volatilized and neutralizes any sulphur trioxide formed in the blast-furnace operations. It carries about 4 per cent. Cu. and on the average 2 to 2 1/2 oz. Ag and 75 to 80 cents Au per ton.

The blast-furnaces have water-cooled closed tops because they are run with much over-fire. Pyritic smelting is followed, using 5 per cent. coke. Owing to the high percentage of sulphur in the ore the furnaces have been run without fuel, but to insure long campaigns and greater tonnage a small amount of coke has been used. Normally four furnaces are in operation, two on ore producing a low-grade matte, and two a higher grade matte which is sent to the converters. Both converter slag and the low-grade matte are cast in endless-chain casting-machines. The thicker part of the converter slag is returned to the blast-furnaces, but the thin slag from the first skimming is poured directly into the furnace forehearths.

Each blast-furnace has 425 tons capacity, and with the cooling system used, only two furnaces have been run in summer; in winter, however, three furnaces have been operated. The plant uses alternating-current, this being purchased from the Northern California Power Co. The seven Connersville blowers are driven by induction motors; the Nordberg compressor for the converters by a rope-drive from a 750-hp. constant-speed induction motor. The compressor runs at constant speed. It has a regulator, which, when the air pressure gets too high, lifts a piston which shuts off the air-inlet valve according to the amount of air being used at the converters. Thus, the energy is regulated, not by changing the speed, but by the amount of air entering the compressor.

The converters are operated by hydraulic power, but no accumulator is used. Instead of this a Mason regulating-valve is employed. This causes the water to be passed back to the supply-tank whenever the pressure gets too high at the converter cylinders, and so the excess of water goes round and round in the pump. This pump is driven at constant speed by an induction motor. The system is reported to work well.

Slag Car.[2]—William B. Pollock & Co. of Youngstown, Ohio, build a large car having a conical slag-pot of 260 cu. ft.—say of 25 tons capacity. The pot can be dumped and righted again in less than 1 minute by aid

[1] *Eng. and Min. Jrnl.*, XCI, 614.
[2] *Eng. and Min. Jrnl.*, XCI, 660.

of a 16-in. cylinder that can be operated either by steam or by air. The slag-cars at the Greenwood Copper Smelting Works, British Columbia, are of 250 cu. ft. capacity.

Straightening Blast-furnace Water Jackets.[1]—At Kennett, Cal. when jackets have been distorted in use, provision has been made for straightening them. They are placed flat in a rectangular reverberatory having a short chimney at one end. The roof of this furnace consists of bricks clamped together to form the cover, just as is done for the outlet-flue of an ordinary reverberatory furnace. The jacket when placed is brought to a red heat by the use of oil-jets. The roof-cover is now removed and the jacket lifted out by means of a track-derrick, then deposited on a solidly constructed cast-iron table. I-beams are laid on the jacket and clamped down by bolts to straighten it. This straightening may start some of the joints, but these leaks are re-rivetted and calked and finally tested by the use of compressed air. The cost of straightening and repairing will be between $50 and $60 where the pay of a machinist is $3.87 per 9-hour shift.

Bag-House Plant of the Mammoth Copper Mining Co., Kennett, Cal.[2]—Claude T. Rice, in a general article on the handling of blast-furnace fumes, describes the bag-house plant at Kennett, the general arrangement of which is given in Fig. 2.

The Plant is described in MINERAL INDUSTRY, XVII, 252.

The gases from the blast-furnaces and from the converters now unite in the main dust-flue where a good part of the flue dust settles and enters a chamber near the foot of the 18 by 200 ft. stack. Thence the gases pass through four 8-ft. pipes to the main flue 15 by 15 ft. in section. This climbs the hill to the fan house and neutralizers. Some air is admitted to the main flue to sufficiently cool the gases, which should not exceed 110° C. at the bag-house. There are two fans of the Sirocco type of the American Blower Co., each 11 ft. diameter running at 219 revolutions per minute and driven by a 400-hp. induction motor. These discharge through two 8-ft. pipes to the steel-fan discharge chamber 20 by 15 ft. in section and 160 ft. long in front of the cooling tubes. Below the fans there is a vacuum of 1.7 in. of water; above, on the discharge side, a pressure of 3 in. At the discharge opening of the fans is a machine for feeding zinc fume to the gases. This is done for a guard since the gases entering the fans show no free sulphur trioxide. This trioxide, formed at the blast-furnace, is neutralized by the zinc volatilized from the ore. Some trioxide is formed at the converters but this is neutralized by the immediate addition of zinc and calcium oxide added to and mixed

[1] *Eng. and Min. Journal*, XCI, 653
[2] *Eng. and Min. Journal*, XCI, 614; *Mining World*, XXXIV, 9.

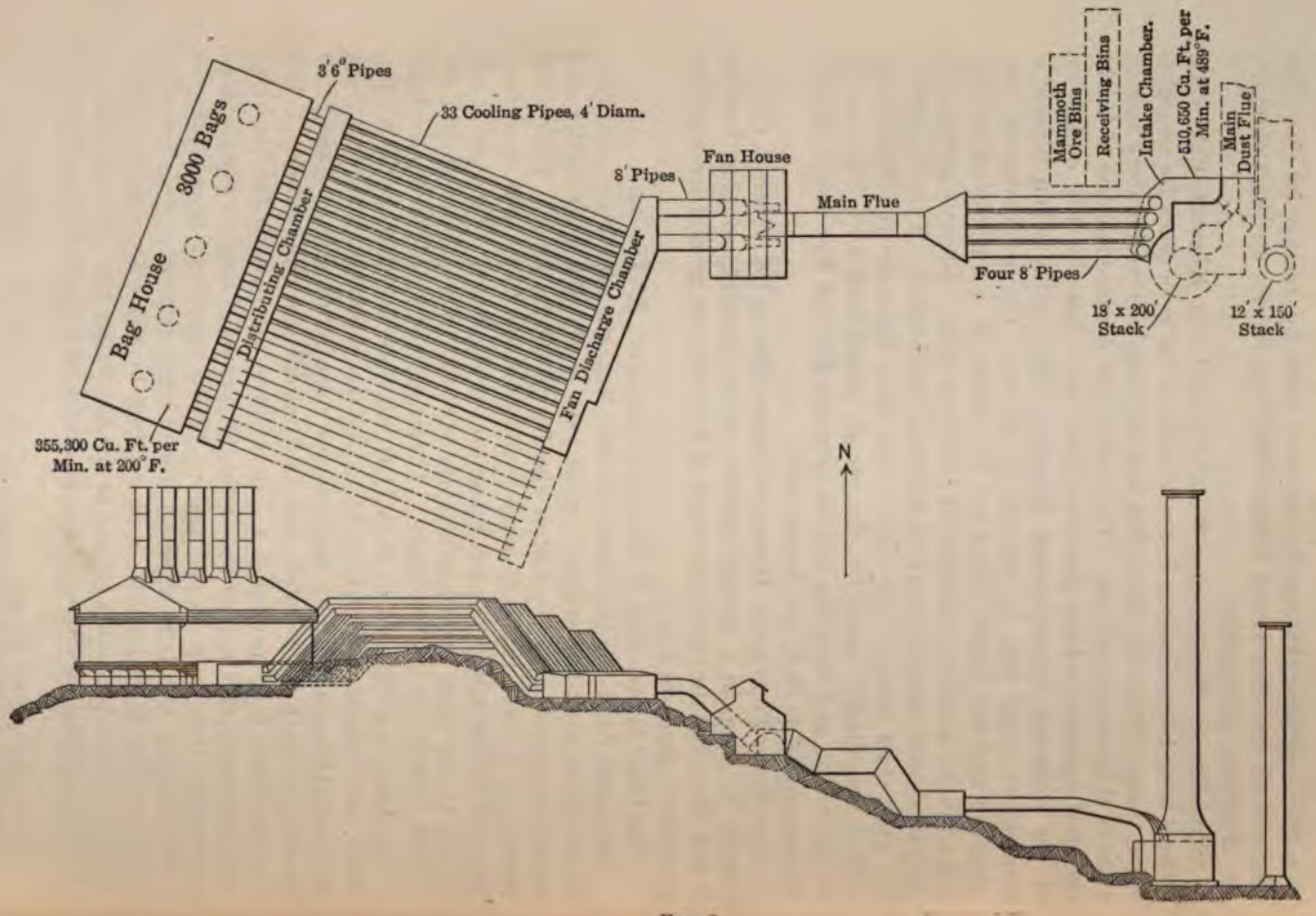

Bag House
3000 Bags
Distributing Chamber
3'6" Pipes
33 Cooling Pipes, 4' Diam.
Fan Discharge Chamber
355,300 Cu. Ft. per Min. at 200° F.
8' Pipes
Fan House
Main Flue
Four 8' Pipes
Mammoth Ore Bins
Receiving Bins
Intake Chamber.
510,650 Cu. Ft. per Min. at 489° F.
Main Dust Flue
18' x 200' Stack
12' x 150' Stack
N

FIG. 2

with the gases. If this were not done the trioxide would speedily destroy the fiber of the bags at the bag-house.

There are 39 cooling-pipes in two rows, each pipe 4 ft. diameter rising in the air to secure effective ventilation. Each pipe is furnished with butterfly valves, and when it is desired to clean them, a certain number are cut out; then the gases passing through the rest sweep out whatever dust may have settled in them. The cooling pipes discharge to a distributing chamber 13 1/2 by 14 ft. in section by 210 ft. long and thence by short pipes to the bag-house, these short pipes 3 ft. 6 in diameter having cut-off valves.

In place of the usual lower chamber there are 20 transverse hoppers, one for each short pipe already mentioned. Beneath the hoppers are circular gates at 3 ft. 6 in. centers for drawing off the dust into cars. The contents of these cars is later dropped through a sealed chute into a tank of water, and is an excellent binder for briquettes. The bag-house is 210 ft. long, 63 ft. wide and has the main chamber 40 ft. high. It has five bays or divisions, each of which contains 600 bags 34 ft. long and 18 in. diameter. Each bay is provided with a tower or stack 20 ft. square by 125 ft. high so as to give a good suction for drawing in air to dilute the newly-filtered gases. This air is sucked in by openings above the floor along the sides of the bag-house. The pressure at the bag-house is about 1.7 in. of water.

Both woolen and cotton bags are used, and for removing the flue dust from them the Benedict shaker is used. It has been found after 3 1/2 months use that no woolen bags have been lost while cotton bags have failed. The bags are inspected by a man who, furnished with a Draeger helmet, replaces defective or broken ones. Four men are needed per shift, one at the fan-house, two at the bag-house and one removing dust. There is one foreman for the three shifts.

In October, 1910, about 250,000 cu. ft. of gases were being treated per minute with a recovery of 10 tons of dust in 24 hours with two furnaces in operation. The fume, largely the sulphate and oxide of zinc, is high in silver, carries a little lead, some gold and about 0.5 per cent. copper.

The Great Cobar Smeltery.[1]—In addition to the description in MINERAL INDUSTRY, XVII, 265 we add the following: The blast furnaces have been increased to four. The forehearths between the furnaces are oval, 9 ft. 6 in. by 18 ft. while the end ones are 18 ft. diameter as already described. The cast-iron hearth base-plates are 3 in. thick, cooled below by water circulating through 1-in. coil pipes. A bottom is built on the base-plate consisting of 2 1/2 in. in depth of silica which serves to bed the bottom of 12-in. silica brick. The water-jackets are in two tiers, each 10 ft. high by

[1] *Aus. M. Ltd.*, Oct. 12, 12 & 26, 1911.

4 ft. wide, and having a 5-in. water space. The tuyère openings are 5 in. diameter. and 1 ft. 9 in. above the silica bottom. The furnace ends are vertical but the sides have a bosh of 7 in. in the length of the lower jackets. From tuyères to charge-floor is 21 ft. The ores analyze as follows:

	Cobar	Bourke.	Chesney.
Insoluble	16.20 per cent.	80.00 per cent.	68.00 per cent.
Alumina	6.50	2.80	8.00
Iron	40.40	9.00	12.00
Copper	2.60	1.30	3.50
Sulphur	11 20	....	....

For the ore-furnaces a typical charge would be Cobar 8000 to 6600 lb.; Chesney 300 to 700 lb.; Bourke 700 to 1700 lb., with 5 to 9 per cent. of coke. The matte produced runs from 16 to 22 per cent. Cu; the slag contains 30 to 40 per cent. SiO_2 and 50 to 53 per cent. FeO.

In the concentrating furnace the charge consists of Cobar 1800 to 2500 lb.; matte 3200 lb.; slag 1200 to 2000 lb., and a quantity of Bourke ore to make the required slag, this charge being run with 7 to 14 per cent. of coke. The matte from this furnace carries 30 to 38 per cent. copper and the slag assays about 40 per cent. SiO_2 and 54 per cent. FeO.

The matte is still reduced in acid-lined converters with the production of a slag containing 1 to 2 per cent. of copper.

Lake Superior Copper Smelting.[1]—Henry D. Conant, Supt., Lake Superior Iron Co., Dollar Boy, Mich., describes Lake Superior practice at the Michigan, the Quincy, the Lake Superior and the Calumet & Hecla smelteries.

Because of its simplicity the idea has arisen that there is no metallurgy in this practice, and on the other hand those who followed it were afraid to vary much from the regulation methods.

In the Lake district the copper occurs in native form disseminated throughout a vein rock of conglomerate or amygdaloid to the extent of 3/4 to 3 per cent. On concentration, products are recovered varying from table-concentrates of 20 to 30 per cent. copper up to jig-products of 70 or even 95 per cent. In addition, a good deal of the copper is found from fist-size to pieces sometimes of tons in weight. The latter have attached or included vein-matter so that they are but 70 to 75 per cent. pure.

When these products are melted in a reverberatory furnace there results metallic copper on which floats the slag resulting from the simultaneous melting of the gangue. From the amygdaloids there is produced an acid slag of about 40 per cent. silica, from the conglomerate comes a basic slag of but 25 per cent. The slag is skimmed as fast as it forms; the copper remaining, called copper of the first melt, is refined, while the slag,

[1] *School of Mines Quarterly;* MINERAL INDUSTRY, XIV, 162; XV, 206: XVIII, 225; private notes.

carrying still 20 to 35 per cent. Cu, is smelted in a blast-furnace to recover the contained copper.

Often the products of a given mine are kept separate in the treatment and whatever is recovered is returned to the shipper; at the Michigan Works, however, a large sample is taken which is credited with whatever belongs to him. As a result of the separate treatment system a small reverberatory has been and is yet largely used, as at the Calumet & Hecla and at the Quincy Works.

Calumet & Hecla Works, Lake Linden, Mich. There are 14 reverberatory furnaces treating principally the products of their mines, and thus distributed:

(*a*) Three melting and refining furnaces for melting and refining high-grade "mineral," etc., in one operation: They take "cover work" or stamp-copper, the higher-grade (over 70 per cent. Cu) mineral, scrap-copper and blister copper. The furnaces will take a charge of 42,000 lb. and produce 32,000 lb. of refined copper. The furnace having been completely emptied by ladle, is repaired or fettled by means of sand on the sloping edges of the bottom, and with a ganister of coarsely-ground brick and fire clay on the walls. A crane, actuated by compressed air, now removes the cover 5 ft. square from the roof-opening or hatch, and the furnace is charged by means of wheel-buggies holding each a ton and lifted by the crane. When charged, the roof-opening is covered, the doors closed and luted, and the furnace fired at an increasing heat until the night-shift comes on. They through the night proceed to melt down and skim the charge, making a coarse molten copper. This is rabbled to "set copper" by means of compressed air blown upon it through 3/4-in. gas pipes with a 1/4-in. nozzle. During this period small amounts of slag form and are skimmed off. The completion of the rabbling is indicated by the fracture of the test-button. This button taken with a small ladle is 2 in. diameter and 3/4 in. thick. It is placed in a vise, nicked and broken in two, and the progress of the rabbling judged from the crystallization of the fracture. When by repeated tests the copper is shown to be sufficiently purified, there remains some 6 per cent. of copper oxide dissolved in the molten bath. To reduce this back to copper, poling is begun and charcoal is thrown upon the bath. The poles are of hard wood their ends pressed down in the copper. When the fracture of the button is coarsely crystalline and sets level then a spike or sample is taken, rolled and tested for conductivity. At the C. & H. works the copper is now ladled by hand-ladles into the molds.

(*b*) Five melting and refining furnaces for the treatment of average grade (30 to 70 per cent. Cu) mineral. They have a capacity of 30,000

to 32,000 lb. and have charging platforms to which the mineral is delivered by platform elevator.

(*c*) Three melting furnaces of 40,000 to 70,000 lb. capacity, for the treatment of low-grade "mineral." One-third of the charge is put in at once, one-third each 8 hours thereafter. There is produced and tapped off a quantity of slag containing 15 to 25 per cent. copper and a little metallic copper. The slag is sent to the blast-furnace and the metallic copper or "bottoms" is charged with other copper into the furnace.

(*d*) Two slag-copper furnaces making a coarse copper from the black copper or cupola-blocks of the blast-furnace operation. This black copper contains 92 to 95 per cent. Cu; 7.5 to 4.5 Fe and 0.5 per cent. S. In blocks or ingots of 400 lb. they are charged through the roof-opening into the furnace, the furnace is luted and the blocks are subjected to the action of an oxidizing flame at such a temperature as will cause the blocks to slowly melt down. When melted, the bath becomes agitated by the escape of bubbles of gas arising from the reactions between the small amount of copper sulphide and cuprous oxide formed during the prolonged preliminary oxidizing operation and also to the attack of the ferrous oxide on the hearth and walls of the furnace. Upon cessation of this ebullition or boiling the slag is skimmed and the copper ladled into anode molds. This has the composition 99.5 per cent. Cu. 0.01 Fe and S, and 0.5 per cent. O, and is sent to Buffalo for electrolytic refining.

The slags from the various reverberatory furnaces go to a blast-furnace or cupola as it is locally called. This furnace, which resembles an ordinary copper matting furnace but with the addition of a crucible, is 42 in. by 144 in. at the tuyères and has 24 tuyères. The charge, consisting of slag, limestone, furnace cobbings, old brick containing copper and other furnace refuse, is smelted with about 15 per cent. of steamer-size anthracite coal. Coal is used rather than coke on account of its higher reducing action, though the latter fuel is occasionally used when it is desired to increase the smelting temperature to melt out accretions that form in the ordinary running. The furnace-top is fitted with a water-cooled thimble or hopper 5 ft. long 3 ft. wide and 2 1/2 ft. deep. The thimble is kept full. The furnace gases are accordingly drawn off by a down-take below the floor. They are high enough in carbon monoxide to permit their being burned at a waste-heat boiler at the foot of the downtake. The furnace is blown by a No. 6 Root blower. The cupola slag will average as follows: SiO_2, 38 per cent; FeO, 21; Al_2O_3, 10; MgO, 2; CaO, 26; Cu, 1 per cent. About 15 per cent. of flue dust is formed. Charge-cars, containing slag fluxes and fuel are raised 30 ft. from the ground-floor level by a monorail transfer system and delivered into

storage bins. From the bins the materials are withdrawn and weighed to the furnace.

The Quincy Smeltery: "Mineral" from the concentrating works is delivered in 60,000 lb. capacity steel-hopper cars by overhead trestle to inclined-bottom storage-bins. Subsequent treatment is as at the C. & H. works, furnaces (*a*). The dipping or ladling of the charge is done as follows: Ladles to hold 200 lb. are used which are swung from trolleys traveling on I-beams. These run from the furnace door to the molds The molds are laid on rails and when filled with copper are dumped into a water trough to cool them. On cooling the ingots are "fished" or taken out with tongs, loaded on trucks, weighed and piled for shipment.

The Lake Superior smeltery, Dollar Bay, Mich. has four large and three small reverberatory furnaces. The large furnaces, 16 by 35 ft. hearth-dimensions, will produce 200,000 lb. of copper per charge. The small furnaces, 14 by 22 ft., are used where small lots from a given mine are to be separately treated.

Three methods were tried and the estimated costs were (1) melting in a large furnace, tapping into and refining in a small furnace $4.12;(2) melting and refining in a small furnace $3.71 and (3) melting and refining in a large furnace $3.12. This includes the smelting of the slag and general expense.

Each large furnace is provided with endless-chain casting-machine, 28 ft. long of a capacity of 45,000 to 50,000 lb. per hour. The machine consists of a series of 35 cast-iron plates attached to the endless chains, which pass over sprocket wheels at each end carrying copper molds and bringing them in front of an hydraulically-operated ladle resembling the Walker casting-machine ladle. This is kept constantly filled with molten copper flowing from the furnace, the stream being regulated by inserting a pole more or less into the tap-hole. The ladle, in addition to a tilting has a reciprocating motion parallel to the molds, and by means of a lug which engages with one of two lines of the chain, is centered and carried along with the mold during the pouring. On being released by the operator it is returned to the initial position by means of a hydraulic cylinder. This permits continuous motion of the molds. As the molds pass over the sprocket wheels the copper ingots, now sufficiently cooled, drop from them into the water of the cooling tank and fall upon a submerged conveyor which gradually raises them to the loading table. The ingots are transferred to tracks of 4000 to 5000 lb. capacity pulled by motor, weighed by two scales in succession and thence go to the loading platform.

An electric overhead monorail crane, traversing the works and yard, serve to handle the cupola blocks, mass-copper, supplies and other

materials; and by means of a clam-shell bucket delivers coal to the furnaces, and hauls slag-pots from the reverberatory furnaces to the blast-furnace charging-floor.

The reverberatory furnaces have waste-heat boilers and by-passes for direct connection to the stack. They are provided with recording pyrometers and draft-gauges.

The blast-furnace, 40 by 120 in. tuyère dimensions, has a smelting height of 7 ft. and is run with a hot top. The copper is collected in the crucible and the slag, nearly free from copper, is run through a settler, then granulated. The furnace, formerly run at 12 to 16 oz. pressure, is now driven slowly with a pressure 3 to 4 oz. per square inch only. This causes better reduction and more thorough settling. By thus lowering the blast-pressure the quantity of copper in the slag has been cut to 0.2 to 0.3 per cent. The copper is periodically tapped from the crucible into ingot carts 300 lb. being the weight of an ingot. It is aimed by the addition of limestone and where needed of some iron ore to produce a slag of SiO_2, 33 per cent.; FeO, 33; and CaO, 20 per cent.

The Michigan Smeltery: This has been quite fully described in MINERAL INDUSTRY. Further particulars are as follows:

The "mineral" is received at the works in 100-ton hopper-bottom cars, a sample of approximately 20 kilogrm. being taken by shovel from the stream of ore as it is discharging into the storage bins. This sample is melted in a crucible, then poured into a mold. There results a bottom or "king" and the supernatant slag. The slag is pulverized, panned to remove the prills, then assayed. The copper, a bar $6 \times 2 \times 2$ in., is weighed with the addition of the prills, then from the drillings is assayed for copper and occasionally for arsenic, iron and sulphur. The results are calculated from the relative weights and assays of copper and slag. The duplicate results should agree to within 0.25 per cent. of copper.

Mass-copper is assayed by determining the specific gravity. A tank is prepared, filled with water till it just ceases to flow from an outlet pipe set a little below the top. The irregularly shaped mass is picked up by the traveling crane then immersed in the water in the tank. The water flows by the outlet pipe to a barrel balanced on a platform scale and the exact weight determined. From the original weight and the water displaced the specific gravity is known and compared with that of pure copper. Such mass-copper will vary from 70 to 75 per cent. Cu.

The copper from the blast-furnace is cast in sectional molds 4.5 ft. long, 2.1 ft. wide and 19 in. deep. The mold, when full, makes a 3-ton block, which can be satisfactorily charged through the furnace roof by traveling crane.

The melting reverberatories will take a charge of 100 tons in 24 hours

producing from it 55 tons of copper and 45 tons of slag. Hence two melting furnaces are needed to one refining furnace, the latter in addition melting other copper of high grade. The refining furnace has melted up to 400,000 lb. at a single charge. The melting furnace burns 25 tons in 24 hours, or with forced draft, 30 tons for a grate-area of 7 by 12 ft., being 30 lb. per square foot per hour. The fire-bed is 4 ft. thick. The works is now supplied with a new melting furnace not yet in use, of 18 by 39.5 ft. hearth area and having a grate 7 by 14 ft. in area.

The actual cost of smelting (including the blast-furnace treatment of the slag and general expense) may be given at $3.25 per ton.

The Quincy Smeltery.—Besides the output of the Quincy Mine, this works treats the product of other mines. Its plan is to treat the product of each mine in a separate reverberatory, to smelt the resultant slag by itself and to credit the owners with all the copper produced, charging them a stated price (formerly quoted at $7.25) for treatment. Thus a preliminary sampling of lots, as received, is not required. So irregular is the constitution of the product, composed as it is of irregular masses of all sizes and with coarse and fine concentrates that this plan has been followed everywhere from the first, but now at the Michigan smeltery, the lots are collectively treated and the resultant product correspondingly distributed to the shippers.

Instead of treating the low-grade "mineral" as specified under (*c*) this material, mixed with milk of lime, is briquetted in a White briquetting-press, then treated under steam pressure in large horizontal cylinders just as in the making of sand-lime bricks. These briquettes after treatment are hard and sufficiently strong to bear handling and smelting in the blast-furnace together with reverberatory slag.

Blast-furnaces of the Canadian Copper Co.[1]—This Company's plant is located at Copper Cliff, Ontario, Canada. They smelt copper-nickel ores to a copper-nickel matte containing 28 to 35 per cent. in combined copper and nickel, the ratio of the metals being as 3 to 5 respectively. As there smelted, the matte is particularly hot and corrosive. The furnaces are 50 by 204 in. at the tuyères and are 20 ft. from tapping-floor to charge-floor. The smelting column is commonly carried at 10 ft., and the furnaces are blown with a volume of 30,000 cu. ft. of free air per minute at a pressure of 50 oz. per square inch. The capacity of each furnace will average 450 to 500 tons per day. The ores on the charge will average 5 to 6 per cent. in combined copper and nickel. There is a resultant heavy matte-fall and a yield of a large percentage of a heavy irony slag, all very hot. About 8 per cent. of coke is used on the charge.

Six years ago smelting was begun at Copper Cliff with blast-furnaces

[1] *Mines and Minerals*, XXII, 55

furnished with steel-plate water-jackets and water-jacketted spouts. When the first furnace was blown in, upon pulling out the tapping bar, there was an explosion as soon as the hot matte touched the steel spout, and a series of explosions followed, lifting the steel spout from its place and hurling it 50 ft. or more. Other spouts were tried with but little success until one lined with chrome brick was put in. The spouts are now made of chrome brick cooled by cast-iron plates containing water-

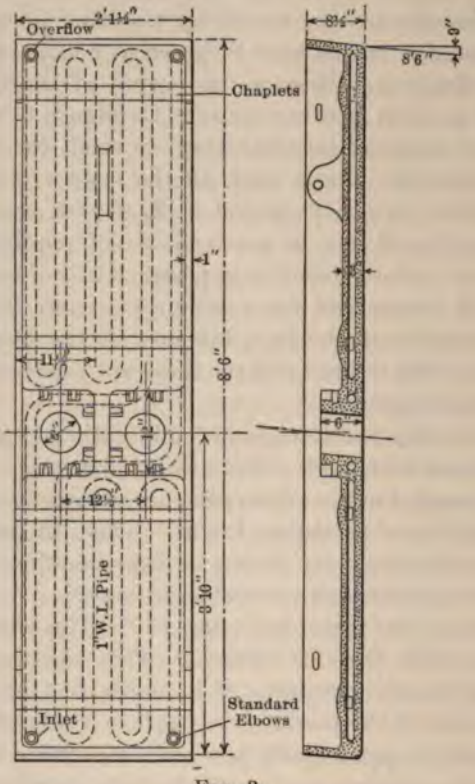


FIG. 3.

cooled pipe-coils. The channel of these spouts lasts a month, then the furnace must be shut down until the brick is renewed.

The greatest trouble was due, however, to the burning out of the lower jackets where attacked by the matte, so that explosions occurred and jackets had frequently to be replaced. The difficulty was finally overcome by using cast-iron jackets as shown in Fig. 3.

Wrought-iron pipe-coils are prepared and the iron is cast round them. They last as much as 9 months without replacement. The pipes are

blown out from time to time with compressed air to remove the sediment deposited by the circulating water. This style of jacket has been in use for 3 years.

Smeltery at Hayden, Gila Co. Arizona.[1]—This is a plant of the American Smelting & Refining Co., its construction having been taken over by that company from the Ray Consolidated Copper Co. for the treatment of the concentrates produced by the latter. The plant is to consist of two (perhaps three) 500-ton blast-furnaces and five or more reverberatory furnaces, each of a capacity of 300 tons daily, together with the necessary converters. Beside the concentrates from the Ray Consolidated the ores from other mines will be treated. Construction has been pushed so that smelting may begin on or soon after April, 1912.

Blast-Furnace Smelting at Rio Blanco, Chili.[2]—Attention is called to the blast-furnace smelting of ore at this works, because of the particularly clean slag of but 0.12 to 0.16 per cent. Cu claimed to have been produced.

A round furnace 44 in. diameter at the charge-door is blown with 12 tuyères and has a capacity of 20 metric tons and an output of 5 tons daily. The ore contains SiO_2, 45 to 50 per cent.; CaO, 8 to 12; Fe, 7 to 19; S, 4 to 6; Al_2O_3, 7 to 10; Pb, 1.5 to 3; Cu, 8.9 to 12 per cent., and Ag, 75 oz. per ton. This was smelted with a dolomite limestone containing 8 to 10 per cent. of heavy spar and with old reverberatory slag containing SiO_2, 45 per cent.; FeO, 31; CaO, 12; Al_2O_3, 8; Cu, 2.5, and Ag, 0.015 per cent. The furnace yielded matte, black copper bars and slag of the following composition: SiO_2, 50 to 52 per cent.; CaO, 26 to 30; FeO, 10 to 12; Al_2O_3, 6 to 9; Cu, 0.15 (*b*) per cent., with traces of mercury and lead.

In ten days were smelted:

54,530 kilos of ore of	12	per cent. Cu
12,910 kilos of ore of	7	per cent. Cu
38,200 kilos of ore of	2 1/2	per cent. Cu

or a total of 8401 kg. Cu and there was produced 2284 kg. of black-copper bars of 90 per cent. Cu and 8230 kgrm. of matte of 75 per cent. Cu, with a matte-fall of 5.7 per cent. The coke constitutes 20.7 per cent. of the charge. Some 90 per cent. of the sulphur, as estimated, was volatilized, and with this slow running, reduction was thorough. The clean slag here reported does not at all correspond with that normally obtained in such work where a slag of at least 2 per cent. was rather to be expected.

Tooele Plant of the International Smelting Co.[3]—This plant is located at

[1] *Eng. and Min. Journal*, XCII, 977.
[2] *Boletin de la Sociedad Mineria.*
[3] MINERAL INDUSTRY, XIX, 199; *Mines and Minerals*, XXII, 56.

International, near Tooele, Utah. The company owns outright 2000 acres, costing $150,000, and has secured options on the cultivated lands for 4 miles around. The plant with its railroad yards includes 10 miles of trackage with a double standard-gauge track running over the ore-bins. A 50-ton electric car handles the ore from the tramway to the steel storage-bins, and the Tooele Valley Railroad owned by the company delivers what comes in by rail. The 10,000-ton bins are built double, having sloping bottoms and in two sections; the conveyor-belts which deliver to the sampling-mill running laterally between them. The rails of the double track are supported by 15-in. I-beams, and the bins rest on 30-in. I-beams with 18-in. uprights set at 10-ft. centers, tied across the top with 24-in. I-beams.

One section receives coal, already-sampled ore and concentrates, the latter sampled by split-shovel. Discharge from these bins is by means of gates operated by compressed-air cylinders and the bin contents is discharged below to cars belonging to the industrial system.

The second section of bins is used to store ore intended for the sampling-mill. These bins discharge through hand-operated rack-and-pinion gates upon a short endless-belt feeder, set parallel and above a 30-in. horizontal belt-conveyor. This feeder can be placed at any desired spout of the bins. The conveyor discharges upon an inclined one having a speed of 170 ft. per minute and rising at an angle of 11 1/2 degrees to the second floor of the mill.

The sampling-mill of steel, 84 by 40 ft. is in duplicate, each unit being driven by a 175 h. p., 200-volt Westinghouse induction-motor.

On the second floor, the inclined conveying-belt just mentioned, discharges to a shaking screen having 2-in. punched holes. The oversize feeds to a 12×24 in. Blake crusher, the undersize discharges with the crusher product to a 20-in. vertical belt-elevator having a speed of 350 ft. per minute. The discharge of the elevator has a one-fifth sample cut out and chuted to a shaking feeder, which delivers into a 20 by 10-in. Blake crusher set to crush to 1 in. diameter. Passing this, a second cut of one-fifth is made and goes to 48 by 12-in. rolls set to 1/4 in. A third one-fifth cut is now made and sent to 26 by 15-in. rolls, then a final cut which is sent to a sample-safe. The sample thus secured is split-shovelled and further ground and quartered, using a laboratory crusher and a Braun disk-pulverizer.

All the ore is treated in reverberatory furnaces, and for roasting is crushed to 3/8 in. size. Oxidized or silicious ores, which do not need roasting, are also crushed to 3/8 in. size. The reject from this first sampling therefore goes to a 4×14-ft. revolving screen provided with manganese steel plates making 16 revolutions per minute. The first

two plates of the screen have 3/4-in. holes, the third 1 1/4-in. holes. The oversize of the screen is crushed by two 15×9-in. Blake crushers set to 1 in. The crushed product, with the undersize of both screens, goes now to a similar screen with 3/8-in. openings. The oversize goes to 48 by 12-in. rolls, which discharges with the undersize into the boot of a short 20-in. elevator and thence upon a belt-elevator. The rejects of the other sampling cuts are chuted to the revolving screens as their size determines.

Conveying system: This mill also sampled ore intended for the lead furnaces, and for this as well as for the contingency that may arise through the smelting of copper ore by blast-furnaces, provision is made both for the conveying of fine and coarse ore to the storage-bins. Both the coarse and fine ore discharge upon 20-in. horizontal and parallel belt-conveyors running in opposite directions. The coarse ore goes to double steel blast-furnace bins, having for copper ore, a capacity of 3500 tons. These bins deliver through compressed-air-operated gates to cars on an industrial system. The fine, or roasting-ore bins, are also double, 180 ft. long, and of a capacity of 5700 tons. The fine-ore conveyor, 300 ft. long, travels 375 ft. per minute, and discharges on either side of the bins by traveling tripper over a 1-in. grizzley carried on its discharge hopper.

From these bins to the roasting furnaces ore is delivered by a conveying-belt system as follows: On either side of the bins is a 20-in. horizontal belt-conveyor, these two discharging upon a third which runs at right angles at the end of the bins. The transverse conveyor is set 6 ft. below the ground-level, and above it is a hopper in one of the industrial system tracks. An ore car will take ore, not run through the sampling-mill, directly to this hopper. The transverse conveyor delivers to an inclined belt-conveyor set at an angle of 14 degrees. This fourth conveyor has a length of 340 ft., and travels at 350 ft. per minute. It carries the ore to the top of the roaster building, passing on its way over a Blake-Dennison weighing machine, so that the feed is automatically weighed and the weight recorded. This fourth conveyor has a tripper, so that delivery can be made to any of four belts running at right angles to the inclined conveyor and commanding a row of roasters. Each of these four belts has its own tripper, so that it can dump into any desired roaster of its row. By the above described system of belt-handling, ore is crushed and delivered to any desired destination with a minimum of labor, and continuous feeding is secured, while 35 hp. covers the requirements of the conveying system.

The roasting plant is contained in two steel-frame buildings, each 64 by 162 ft. In it are thirty-two 16-ft. McDougal roasters arranged in four rows. (In 1911 but fifteen were in use.) Each roaster has six

hearths, the shaft making a revolution in 55 seconds, and the ore taking 2 hours to pass through the roaster. The capacity per roaster will reach 45 tons per day. The principal ore now roasted is from the Highland Boy mine of the Utah Con. Mining Co. This contains 27 to 30 per cent. sulphur, and is roasted down to 6 to 7 per cent. sulphur. Silicious ore is added on the fifth hearth of the furnace in the proportion of 70 tons to 400 or 500 tons of sulphide ore, and hence the resultant calcine carries but 5 per cent. of sulphur. In the roasters treating Highland Boy ore, the third, fifth and sixth stirrer arms are water-cooled, but in treating fine concentrates from the South Utah mine the arms are not water-cooled. The hot water from the stirrer arms is piped to a cooling-tower, the cooled water pumped by a centrifugal pump to a circulating tank on top of the roaster building. The introduction of silicious ore on the fifth hearth tends to cool the lower hearths, and consequently to maintain the heat, a brick fire-box has been built at the third hearth and fired with coal. One roaster was used formerly for drying the silicious ore needed for convertor linings. Each unit of sixteen is driven by a 30-hp. motor.

The roasters have brick dust-chambers, each 162×14×12 ft. in size, the roasters being connected to these by 36-in. pipes. These chambers are provided with steel hoppers of 14×2-ft. dimensions, spaced 8 ft. apart. These smaller dust-chambers discharge into the main dust-chamber, which is 120 ft. wide, 140 ft. long and 40 ft. high, terminating in a flue 255 ft. long by 16 ft. wide and 16 ft. high, then entering one side of the main stack. Beneath the main flue-chamber run three tracks of the industrial system and the bottom of the chamber is provided with pyramidal steel hoppers having 12-in. gates by which the flue dust is withdrawn into cars set below. The ventilating of the dust-chamber posts (MINERAL INDUSTRY, XIX, 200) has not proved advisable. Each McDougal furnace has two discharge hoppers by which the calcine is withdrawn to cars holding 7300 lb.

An analysis of the roasted Highland Boy ore, or calcine, shows Cu, 2.3 per cent.; SiO_2, 26.7; FeO, 49; and CaO, 3.6 per cent.

The reverberatory furnaces have eight doors to the side. Between the doors the walls are braced with buck staves of 6-in. I-beams at 12-in. centers, tied with 1 1/2-in. rods across the roof. The furnace is tied lengthwise in the same manner. The calcine, arriving in cars from the roaster building, is dropped into two hoppers, one with two and one with three discharge openings. Two charges of 12 tons each are put in twice per 8-hour shift. Coal, brought from the storage-bins, is put into the furnace from a single hopper with five openings.

In operating the furnace, when the slag has accumulated to 3 or 4

in. above the skimming plate, it is skimmed, flowing by a steel launder to the 20-ton slag-cars of 225 ft. capacity. Each car has its own motor for independent dumping. The cars are hauled to the dump by a Baldwin-Westinghouse locomotive fitted with two 60-hp. electric motors, automatic couplings and air-brakes. The reverberatory matte can be tapped through a copper tapping-plate and steel launders direct to the converters. It is preferred, however, to tap to a ladle by which the amount of matte can be measured before pouring into the converter. Converter slag is returned to the reverberatory furnace by ladle, and is now poured into a side door, using a movable launder so as to discharge well within the furnace thus lessening the formations of skulls or chilled slag about the door opening.

An average reverberatory slag will analyze Cu, 0.35 per cent; SiO_2, 42 to 45; FeO, 42 to 45; CaO, 4.5 per cent., while an average of the matte shows Cu, 28 to 31 per cent.; Fe, 40; S, 25 per cent. Converter slag carries SiO_2 29 per cent. and FeO 58 per cent. The reverberatory flue enters the main stack on the side opposite where the roaster flue enters. This flue has clean-out doors, and both it and the roaster flue have heavy counterweighted steel dampers. The natural draft at the stack is 1.8 in. of water.

In the converter department the ladles into which the blister copper is poured are handled by the main traveling crane and carried to the casting floor. The blister copper, containing 98 to 99 per cent. Cu, is cast into ingots of 250 lb. These are shipped to the United Metals Refining Co., Raritan, N. J., for refining.

The converters are basic-lined. They are at present operated only one shift. When one is closed down it is sealed to retain the heat and by this means it is hot enough to resume operations next day. The converter hoods are raised or lowered by a small electric motor. The converter fumes go to a steel hopper-bottom flue 248 ft. long by 8 ft. wide and 12 ft. deep with hoppers having 12-in. discharge openings set at 4-ft. centers. This flue goes to another 181 ft. long with a cross-sectional area of 132 sq. ft., which connects to the flue serving the reverberatory furnaces.

The converter plant is equipped with five Morgan-Gardner electric cranes, one of 60 tons for handling converters, one of 30 tons for handling ladles on the casting floor, and two of 12 1/2 tons each for returning converter slag to the reverberatory furnaces.

At the power-house steam is carried in two 12-in. lines from the reverberatory waste-heat boilers, from one of three 250-hp. hand-fired boilers. The other two boilers are kept in reserve. The hand-fired boiler gases discharge into the main reverberatory flue, but the boilers are so by-

passed that the gases can discharge through a stack 15 ft. diameter by 40 ft. high. When thus arranged, the boilers are supplied with induced draft from a 12-ft. Sturdevant fan direct-connected to a vertical 8 by 13-in. piston-valve engine. In the basement of the power-house, the feed-water is filtered through coke and heated in an open heater, using the exhaust steam from the pumps and the steam-trap discharge.

At this plant the aim has been to reduce manual labor to a minimum especially in the handling of the various products. The operating force is less than 300 men. From the start operations have been carried out with few drawbacks due to design. The chief difficulty has been a shortage of ore. The contract with the Utah Consolidated Co. for the Highland Boy ore called for a tonnage of 1200 tons daily, and with this and other ores it was expected to keep five reverberatories running. The amount, has however, been so much less, that no more than three reverberatory furnaces have been kept running. As a result the stack-draft is less than was expected, and the general expense becomes proportionately more.

The water-supply is brought to the plant from a dam in Pine Canyon a distance of 5000 ft. in a 12-in. pipe and is distributed to the plant from a 50,000 gallon tank.

As a matter of precaution against "smoke farmers" the live stock and the vegetation of the surrounding country was examined by botanists and veterinarians before operations were begun and careful notes were collected which can be later used for comparison. Besides this options and abatement agreements were made.

Smeltery at Calama, Peru.[1]—The works are situated at Chorillos, near Calama, Peru. It belongs to the Compañia de Minas y Fundicion de Calama. Ores of 10 per cent. and concentrates of 17 per cent. are smelted with the production of matte and the conversion of it to blister copper.

There is one rectangular blast-furnace, 42×144 in., having two tiers of jackets, and with seven tuyères to the side. The shaft of the furnace is carried above the jackets in brick. There are two other blast-furnaces, one circular and one octagonal, the latter having one tier of jackets. The furnaces have forehearths 60 by 84 in. They have one blast-main. The blast air is furnished by electric power, and when that fails, by a suction gas-engine plant (using anthracite coal) of 32 hp. driving a 4-cycle engine, which runs at 160 revolutions per minute. The gas-power plant cost $95,266.81 (silver) and costs to operate it $12 per day.

The converters are placed on a lower level so that the matte is run to them by gravity. They are small, four of them 68 by 96 in., four of

[1] *Mines and Minerals*, XXXI, 605.

87 by 92 in., and are tipped or revolved by hand. The converters take their air from a Riedler two-cylinder compressor 22 by 45-in. stroke.

To drive the plant there has been put in a hydro-electric installation, and the current is transmitted 2 miles. Power is given by a 250-hp. horizontal turbine which drives a three-phase electric alternator of 2000 volts. At the receiving end is a motor of 180 hp. at 1950 volts. Labor operating-cost for this installation is $17 per day. This plant cost $102,015.40 or $400 per horse-power.

Limestone costs $2.80 per ton delivered at the smeltery. It carries 5 per cent. silica. Ten per cent. by weight of quicklime is also used for briquetting concentrates. Coke, delivered at the smeltery, costs £3 11s 4d, anthracite £3 7s 4d per ton.

In June, 1909, the furnaces put through 3162 metric tons of charge (including by-products returned) containing 990 tons or 32 per cent. of its weight of limestone. Of coke 598 tons were used or 18.8 per cent. of the charge. There was produced 2258 bars or 243 tons of blister copper, being a production of 8 per cent. of the charge. The copper loss is given at 16 per cent. Cost of smelting is figured as high as $24 (silver) per ton.

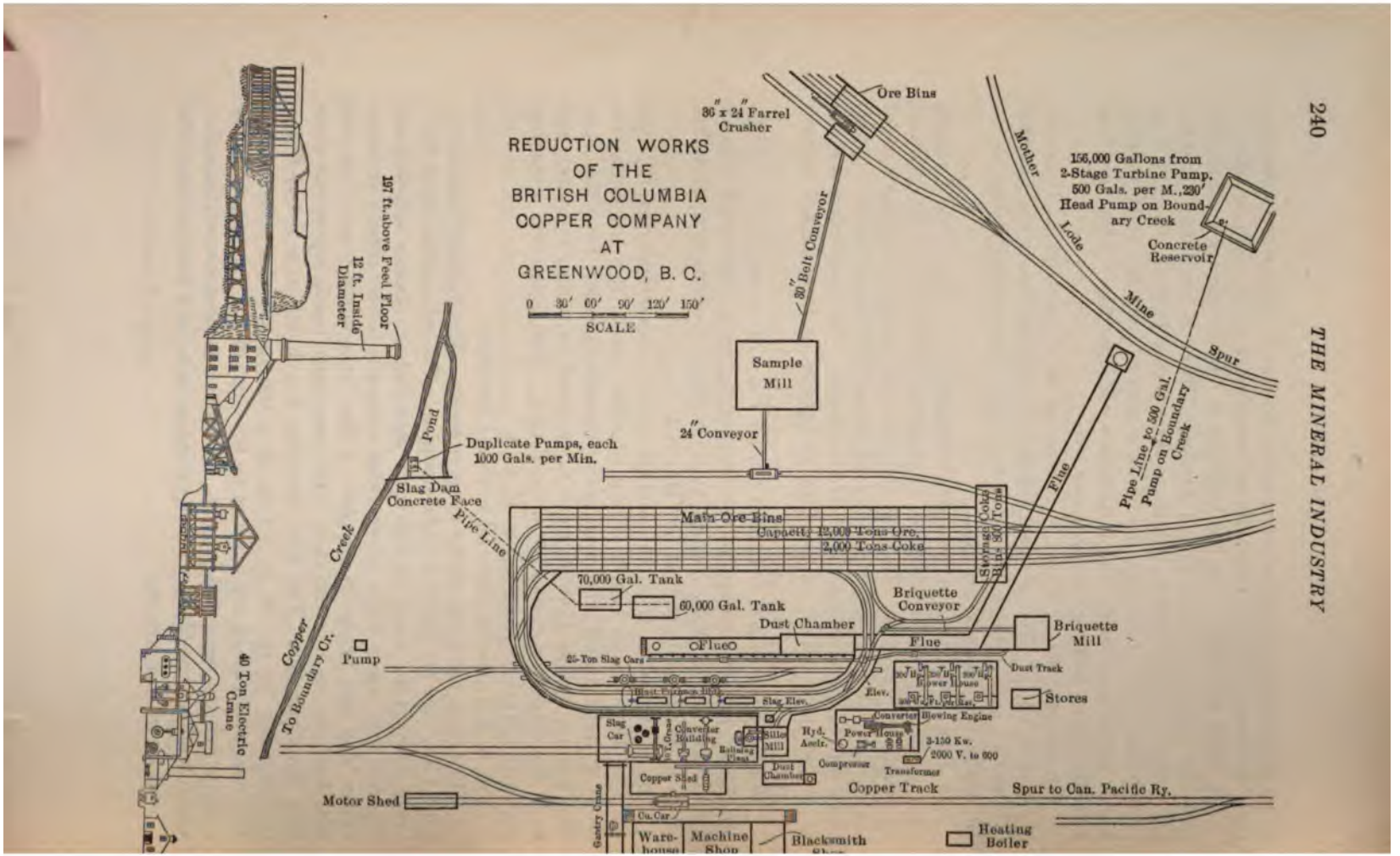
Works of the British Columbia Copper Co., Greenwood, B. C.[1]—J. E. McAllister, general manager of this company, gives a reliable and detailed description of this works, which may be read in conjunction with that given in MINERAL INDUSTRY, XVIII, 212.

Ore is furnished from the Mother Lode Group 4 miles distant, from the Oro Denoro and Emma Group 10 miles away, from the Jack Pot, Molly Pritchard and Athelstan group distant 18 miles, from the Lone Star Group, 9 miles (five of which are covered by aerial tramway), and from the Napoleon Group 60 miles distant. The latter two properties are in the State of Washington, U. S. A.

Fig. 4. is a general plan and sectional elevation.

The tracks enter the works at three levels. The *high level* delivers to "ore bins," the ore which is to be sampled. The *intermediate* level 48 ft. lower delivers to "main ore bins" the coke and the already sampled ore. The *low level*, again 56 ft. lower, takes away the blister copper and converter slag and brings in supplies to the "warehouse," etc. At the *high level* the ore bins can store 2000 tons; three tracks traversing these bins longitudinally. The bins discharge by chutes to a Farrel-Bacon rock-breaker 24 by 36 in. driven by a 100-hp. variable speed motor. The coarsely-crushed ore is taken to top floor of the "sample mill" by a 30-in. belt-conveyor 225 ft. long. It delivers to an automatic sampler, which removes one-fifth for further crushing at a No. 5 Gates crusher, the

[1] *Eng. and Min. Jrnl.*, XCI, 1011.


REDUCTION WORKS
OF THE
BRITISH COLUMBIA
COPPER COMPANY
AT
GREENWOOD, B. C.
0 30' 60' 90' 120' 150'
SCALE
Ore Bins
36" x 24" Farrel Crusher
30" Belt Conveyor
Mother Lode Mine
Spur
156,000 Gallons from 2-Stage Turbine Pump, 500 Gals. per M., 230' Head Pump on Boundary Creek
Concrete Reservoir
Pipe Line to 500 Gal. Pump on Boundary Creek
Sample Mill
24" Conveyor
197 ft. above Feed Floor
12 ft. Inside Diameter
Pond
Duplicate Pumps, each 1000 Gals. per Min.
Slag Dam Concrete Face
Pipe Line
Copper Creek
To Boundary Cr.
Main Ore Bins
Capacity, 12,000 Tons Ore, 2,000 Tons Coke
Storage Coke Bin, 500 Tons
Flue
70,000 Gal. Tank
60,000 Gal. Tank
Dust Chamber
Briquette Conveyor
Briquette Mill
Flue
Pump
25-Ton Slag Cars
Dust Track
Blast Furnace 100'
Slag Elev.
Elev.
Power House
Stores
Converter Blowing Engine
Slag Car
Converter Building
Ryd. Accelr.
Power House
3-150 Kw. 2000 V. to 600
Transformer
Compressor
Dust Chamber
Copper Sled
Copper Track
Spur to Can. Pacific Ry.
Motor Shed
Gantry Crane
Cu. Car
Ware-house
Machine Shop
Blacksmith
Heating Boiler
40 Ton Electric Crane

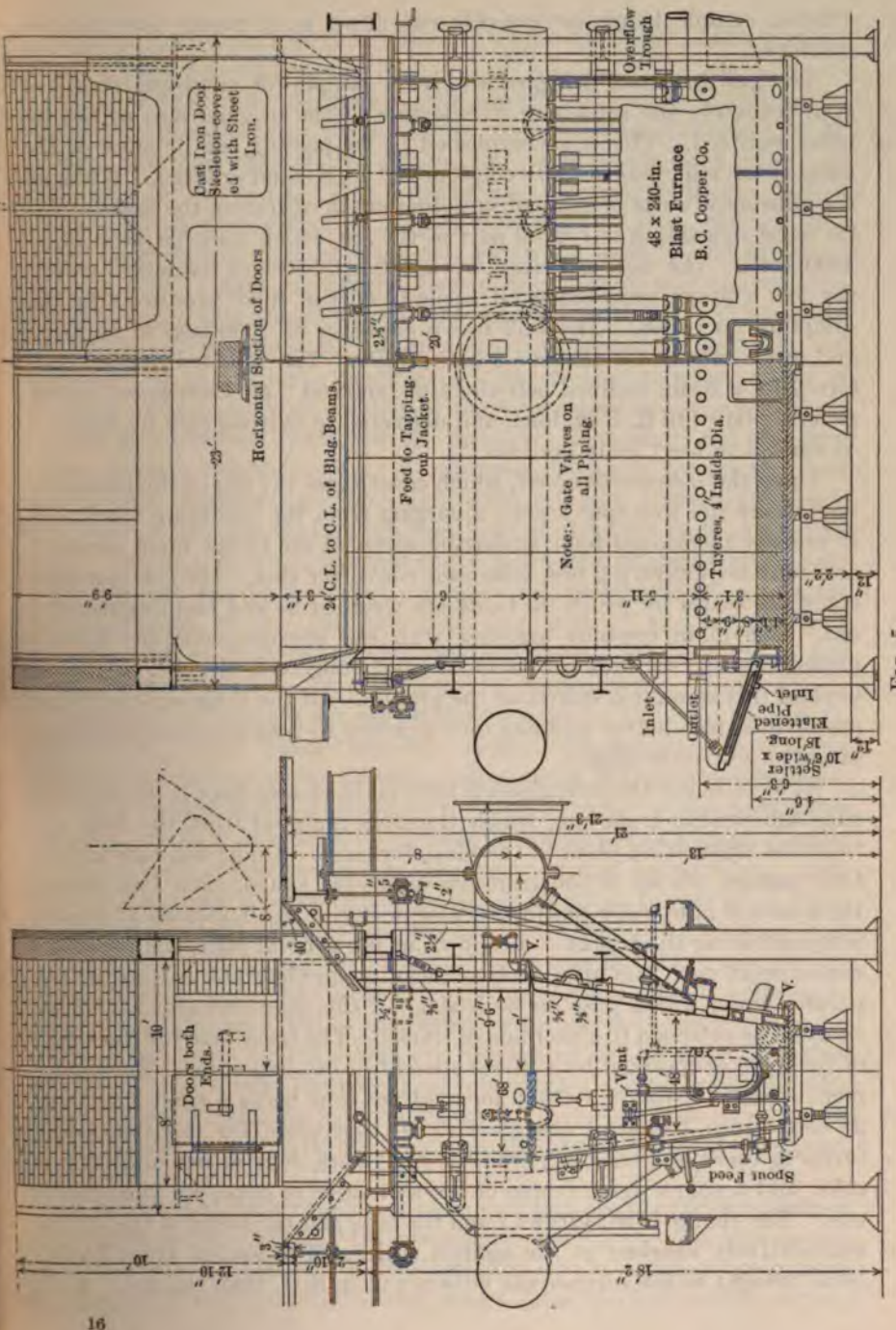

Overflow Trough
48 x 240-in. Blast Furnace
B.C. Copper Co.
Cast Iron Door Skeleton covered with Sheet Iron.
Horizontal Section of Doors
2(C.L. to C.L. of Bldg. Beams.
Feed to Tapping out Jacket.
Note:- Gate Valves on all Piping.
Tuyeres, 4 Inside Dia.
Inlet
Outlet
Flattened Pipe Inlet
Settler 10'6" Wide x 15' long
Doors both Ends.
Vent
Spout Feed

FIG. 5.

crushed sample being successively cut down as it passes through the machines.

The mill has been designed to give a sample for hand-quartering of approximately the same size irrespective of the original size of the lot being sampled. This is accomplished by by-passing one or more of the units of the mill, according to whether the lot is small or large. When all the automatic steps or units of sampling are in operation the amount left for hand quartering is 1/6250 part of the lot, which may be as high as 3500 tons. The mill is driven by a 100-hp. 550-volt induction motor, the low voltage used being on account of the dust produced in the sampling operation. The mill has a capacity of 2000 tons in 16 hours and needs 11 men for the two 8-hour shifts. The rejected ore is transferred by a 26-in. inclined belt-elevator (marked "24" Conveyor" on the general plan) 116 ft. 3 in. long, and delivered to railroad cars by which it is carried to the "main ore bins."

Upon the *intermediate level*, which constitutes the top of the main ore-bins, there are five spur tracks diverging from the incoming track, one as already mentioned for transferring sampled ore to the main ore-bins, and four for delivering ore, coke and converter slag. On the incoming track are scales by which all loads are weighed in and empties weighed out. The main ore-bins together with the storage coke-bins have a united capacity of 12,000 tons of ore and 3000 tons of coke. All coke beyond this amount is stored on the ground. Of the main ore-bins three are divided into halves and one into quarters to accommodate small lots of ore or converter slag.

At 31 ft. below the intermediate level is the charge-floor level on which runs the electric tram-line, which transfers material from the bins and from the briquetting plant to the blast-furnaces. Under the bins are the four parallel tracks of the tram-line of 36-in. gauge, each line having three sets of automatic platform scales or twelve in all, located at regular intervals along the tracks. As shown in the general plan, the four lines converge at each end to a semi-circle of a double track ending in two parallel lines, which pass on both sides of the blast-furnaces. Charge-cars can be switched to either line of tracks. The tram-line, for the safety of the men, is operated by a direct current of but 220 volts. The equipment consists of four 7 1/2-ton locomotives (one being a spare unit) and 26 side-dump V-shaped charge-cars. In operation for the two larger furnaces there is a charging train of eight cars, four of ore and four of coke, and a six-car train (three of ore and three of coke) for the smaller one. The charge is withdrawn from the bins and the items of the charge consecutively weighed at the nearest scales. The loaded train having been brought to the furnace the coke is put in first, then the ore. Each

car will hold 4000 to 6000 lb. of ore and hence each side of the furnace will take from 8 to 12 tons of ore.

BLAST FURNACES.—As indicated on the plan there are three furnaces arranged lengthwise with the building, the center furnace being 48 by 240 in., the end ones 48 by 360 in., the three having a capacity of 2400 tons in 24 hours. Fig. 5 gives two elevations of the center furnace. From charge-floor to tapping-floor is 21 ft. 3 in., and the converter floor which may be regarded as at the level of the low-level spur, is 4 ft. below the central bench which constitutes the tapping floor, while the slag-track, on the other side is 8 ft. below. The furnace has two tiers of jackets and a cast-iron charging or distributing plate set at an angle of 40 degrees. There are six jackets below and six above in the 48 by 240-in. furnace. Each of the long sides has two doors operated by air-cylinders under 100 lb. pressure. The doors are cast-iron frames covered with sheet-iron. To properly distribute the ore and fuel charges, the furnaces are provided with swinging baffle-plates hung from 4-in. pipes supported at the feed-floor level by the ends of the furnaces. The charge when dumped slides down the charging plate and strikes against the baffle-plates, which swing inward and affect a suitable distribution of the material. At each end of the furnace is a trapped-spout, so that either end may discharge. In case repairs are needed at one end of the furnace, it can be tapped at the other end.

Between the furnaces are shown the forehearths 18 ft. by 10 ft. 6 in. by 4 ft. 6 in. deep. From these the matte is tapped directly to 5-ton ladles handled by the 40-ton traveling crane of the converter department. The small matte-fall has a tendency to cause freezing of the forehearth tap-holes. This is overcome by burning the tap-hole out with an electric arc, using for the purpose a 1-in. diameter carbon about 30 in. long. The slag is handled by slag-cars described in MINERAL INDUSTRY, XVIII, 213, and XIX, 189. To tilt the slag-car in order to pour the slag, it is provided with a 15-hp. motor. This motor is operated from a controller mounted in the cab of the 35-ton locomotive, which handles the slag-cars. The current is transmitted to the tilting motors from the trolley wire by means of extension cables, which are coupled up at the same time that the cars are. During the interval of changing slag-cars a small ladle, supported on a swinging arm, intercepts and receives the slag-stream, this small ladle dumping its contents into the large bowl which is next set in place.

As shown on the sectional elevation of the works, the flue dust is removed from the furnaces by goose-neck pipes 7 ft. diameter to the main dust flue of 180 sq. ft. cross-section. This connects to a dust chamber of 550 sq. ft. cross-section where most of the flue dust settles,

then by a long flue rising on an incline to a stack 121 ft. high and 12 ft. inside diameter. The height from the feed-floor level to the top of the stack is 197 ft.

About 1.5 per cent. of flue dust is recovered from the flues. It is briquetted in a 3-plunger press driven by a 40-hp. motor. There is calcite in the charge so that when the flue dust is incorporated with water in a mixer there is a lime bond to compact the briquettes. These stand for 24 hours and are then fed to the blast-furnace.

The average contents of the charge would be from 1.1 to 1.3 per cent. Cu and $1.10 to $1.20 in gold and silver. To produce a matte of 45 to 50 per cent. copper this would mean a low matte-fall, and would call for rapid smelting in order to prevent incrustation in the furnace. The endeavor is to produce a slag of SiO_2, 42 per cent; FeO, 24; CaO, 2.2; Al_2O_3, 9 per cent. Some 12 to 14 per cent. of coke is used. A blast of 16 to 24 oz. is carried, but the blast-pressure and the percentage of coke is manipulated to keep down oxidation and to eliminate as little sulphur as possible. The smelting column is kept at 13 ft.

The converter plant has two stands and a capacity of 40 to 50 tons of 45 per cent. matte per 24 hours. The shells, 84 by 126 in., are lined with silicious gold-bearing ore from which a profit is derived in treatment. The blast-pressure for bessemerizing is from 7 to 12 lb. per square inch.

The plant is equipped with a complete water system, there being two sources of supply, one by gravity from Copper creek above, the other by means of two 500 gallon per minute turbine pumps from Boundary creek below the plant. The pumps are two-stage and direct-connected to induction motors. Their control is independent of the power-house switchboard in order to secure operation in case of fire. A concrete reservoir of 150,000 gallons is also provided. The jacket-water is sent to a cooling-pond and thence pumped back to the service tanks above through a 14-in. main by two direct-connected centrifugal pumps each of 1000 gallons capacity per minute operated by induction motors. The blast-furnaces need 40,000 gallons per minute.

Alternating current for all operations is purchased from an independent company on a watt-meter basis. It is generated 85 miles away and is conducted at 40,000 volts to a step-down transformer station near the works where the voltage is reduced to 220 volts.

The power-house is in two adjoining sections. Section A contains the switchboard, two motor generators, the converter blowing engine, high-pressure air compressor and hydraulic accumulator. The motor generators are of 75 kw. and 100 kw. respectively and furnish direct current to the traveling crane and the slag and charging railways. The converter blowing-engine consists of one side of a cross-compound

machine having an air cylinder 40 by 42 in. stroke and with a capacity of 5000 cu. ft. per minute. It is run by rope drives from a 300-hp. variable-speed induction motor. The air from the compound duplex compressor is carried at 100 lb. per square inch and is used for operating the blast-furnace charging doors and the pneumatic tools. The machine is of 300 cu. ft. per minute capacity and operates at 150 revolutions per minute. Section B of the power-house contains the blast-furnace blowers. Each blast-furnace is independently connected to a rotary blower of 300 cu. ft. displacement run at 85 revolutions per minute. They are designed to deliver at the rated capacity up to 40 oz. per square inch and are each driven by a 300-hp. induction motor by belts 40 in. wide.

Mr. McAllister gives the total cost of the plant at $900,000, a remarkably low figure for a tonnage of 2400 tons daily. This is equivalent to $375 per daily ton or $1.14 per yearly ton computed at 330 days per year. [1]

The Casapalca Smeltery, Peru.[2]—This works is on the Central Railroad of Peru, 95 miles from Callao at an elevation of 13, 600 ft. and is supplied chiefly by the Natividad and Carmen mines at Morochoca, 20 miles north on a branch line of the railroad. From 5200 to 5500 tons of crude ore and concentrates are treated monthly with custom ores, fine crushed and bedded with the concentrates.

In outline, the general process consisted in roasting this bedded mixture, briquetting it, then smelting it together with some raw ore for the production of matte. The process has been of late modified by sintering a portion of the mixture in pots, then feeding this together with raw ore, briquettes and limestone for the production of slag and of a shipping matte.

The coarse hand-jigged concentrate is mixed with custom ore and the mixture crushed to 4 mm. in a dry-grinding Krupp ball-mill. This, and the fine concentrates are bedded to form a roasting mixture of the following composition, viz., SiO_2, 25 per cent.; Fe, 15.6; CaO, 1.5; Zn, 10; Pb, 11; Al_2O_3, 4.4; S, 24; Cu, 5 per cent., and Ag, 45 oz. per ton. A loss of 10 per cent. occurs in roasting. The roasted product contains SiO_2, 27 per cent; Fe, 18; CaO, 3.5; Zn, 9.2; Pb, 10.3; Al_2O_3, 2.1; S, 7.6; Cu, 6.4 per cent., and Ag, 50 oz. per ton. These figures indicate a loss of 17 per cent. zinc, of 71 per cent. sulphur and of 7 per cent. lead, but there is no loss of silver.

The roasting plant consists of the Brown furnaces having single hearths 10 ft. wide by 180 and 240 ft. repectively, the discharge ends facing. Fuel-oil is used under a pressure of 12 oz. per square inch. The longer furnaces have eight rakes, the smaller six, each fitted with nine

[1] See MINERAL INDUSTRY, XIX, 214
[2] *The Mining Mag.*, V, 59.

plow-blades, all moving at 55 ft. per minute. To break up the crust, which tends to form on the hearth a special bladed rake is daily used. A 30-hp. motor drives the larger furnace, though 10-hp. is generally sufficient, the excess is useful in case the ore becomes sticky through cooling. The larger furnace outputs 68 tons daily consuming 3.3 tons of fuel-oil, the smaller has a capacity of 54.5 tons consuming 3.2 tons of oil. Delivered at the works oil costs $16.50 per ton. It comes from Lobitos in northern Peru.

For briquetting, the roasted ore is wheeled to a cooling floor, and is afterward fed, together with flue dust and quicklime, to a troughed helical-screw mixer, then elevated to a bin behind a briquetting machine of the White-Boyd-Chisholm type. Each stroke of the press yields six bricks, weighing each 8 3/4 lb. The capacity of the machine is 30,000 bricks daily. The bricks are allowed to stand to harden and to dry, and are then fed to the blast-furnace.

In place of roasting all the bedded mixture as above described, a portion is now directly sintered and in this form added to blast-furnace charge. The ore is crushed more coarsely than for roasting (3/4-in. mesh) and is mixed with additional limestone and with water to give a product containing 5 per cent. moisture. It is roasted in pots 8 ft. 8 in. diameter, with a shell 2 in. thick tapering to 1 1/8 in. thick at the bottom and weighing 11,000 lb. The pot is mounted on trunnions, and is revolved by worm-gear and worm to discharge its contents, the whole pot at the same time rolling forward on a slight downward grade. The inside surface of the shell has four shoulders or offsets of 7/8 in. each, regularly spaced, their object being to prevent the creep of the air blast up the sides of the pot. The grate forming the false bottom is 1 1/8 in. thick, arched and having three sections, the center one being the longest with side pieces of equal size. It stands 14 in. above the center of the bottom resting on a 1-in. shoulder. This false bottom is perforated with taper holes 3/8 in. in diameter spaced at 2 1/4 in. in centers. The hood, which covers the pot while in action, has eight openings or doors, one where the ore-spout enters and seven for poking the charge. The hood can be raised by a hand-winch when the pot is ready to dump. At present there is no flue-dust chamber, but this will be added when the full complement of 20 pots shall have been put in. During blowing the pot is held in position by a 3/8-in. wire rope controlled by a hand-winch. The capacity of the pot is 9 tons of crude ore. Its cost when made at Lima was $1500, including freight and erection ready to run, $2500.

Operation.—The pot is charged at 6.45 a.m. with some shavings and wood to the depth of 3 or 4 in. and this is set on fire by a rag dipped in oil, this being dragged over the surface of the fuel. A 1/2-oz. blast is

next turned on and after a few minutes 100 lb. of fine coke (3/4 in. and less) spread over the fire. After the coke is ignited 100 lb. of limestone is spread on it, this being added to prevent the bottom of the charge getting too hot and perhaps melting. When the limestone looks red 300 lb. of charge is added and spread, and the blast slightly increased. Yellowish sulphur fumes now begin to come off and the ore crackles. As the mass begins to get red, 300 lb. more of charge is spread on this. After a fourth charge has been added, making in all 1200 lb., the blast pressure is again increased and charges of 800 lb. run in by a chute from the storage bins above. The charge tends to pack and must be punched from time to time to keep it open and loose. At each addition of charge the pressure increases by 1 or 2 oz., but this falls as the ore is ignited. As charging proceeds volitilization proceeds more rapidly until at the last sulphur comes off at once when an addition is made. At 5.30 to 6 p.m., by which time the charging is complete, the pressure has increased to 5 1/2 oz. The surface is warm and the interior of the mass where punched shows red hot. There is no blowing away of flue dust. Following the yellowish sulphur fumes and the bluish-white sulphurous gases come the bluish and red flames till the surface of the mass resembles a forge fire. The heaviest part of the operation is now complete, and the pressure is raised every few hours, until by the time for discharging at 5.30 a. m., it has attained to 9 or 10 oz. At 6 a. m. the pot is dumped, the charge sliding out easily; the operation of discharging the two pots taking 15 to 20 minutes. The sinter yields but little fine, the texture is porous and mass tough. The charge loses 10 per cent. in weight, and desulphurization amounts to 75 per cent. As compared with roasting and briquetting the advantages of sintering are (1) less labor needed, (2) 5 per cent. better desulphurization, (3) a product that will smelt 40 to 50 per cent. faster, (4) less repairs, (5) more continuous running, (6) less room needed, (7) less flue dust, (8) lower fuel and less flux needed, (9) less cost of equipment.

The blast-furnace is of standard type 42 × 160 in. at the tuyères. There are 24 tuyères spaced at 12-in. centers. From tuyères to charging floor is 10 ft., the charge being generally carried to 7 or 8 ft. The side jackets have a 6-in. bosh and there is no end bosh. The matte is caught in a forehearth and the slag is granulated. The blast pressure varies from 11 to 15 oz. and 420 cu. ft. of free air per square foot of hearth area is furnished per minute. The furnace has a closed top. About 7 per cent. of flue dust is made.

The contents of an average charge may be thus given: raw ore 500 lb., slag 400, limestone 600, sinter 500, matte 400 and briquettes 3000 lb. This is run with 9 to 11 per cent. of its weight of coke.

The slag has the following analysis: SiO_2, 40.8 per cent.; Fe, 20.9; Al_2O_3, 5.5; CaO, 15.8; Zn, 10.3; Cu, 0.3, and silver, 1.1 oz. per ton. The matte contains Fe, 14 per cent.; Zn, 6; Pb, 6; S, 22; Cu, 50, and silver, 400 oz. per ton. It is shipped to Liverpool.

Properties of Copper and Its Alloys

Volume Changes as Related to Crystallization Changes in the Copper-tin Alloys.[1]—Prof. Turner of the University of Birmingham, England, in collaboration with J. L. Houghton, has investigated volume changes in a series of 35 bars of copper-tin ranging by steps or intervals of 5 per cent. Measurements were made with a specially contrived extensometer giving

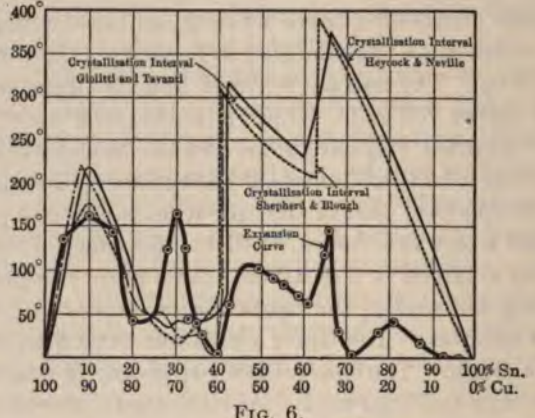


Fig. 6.

the lineal contraction in cooling of bars of the various alloys 3/4 in. square. Simultaneous observations were made of the contractions and fall in temperature of each bar of the series.

Fig. 6 represents on one diagram the expansion curve and the crystallization interval as given by Heycock and Neville, by Shepherd and Blough, and by Giolitti and Tavanti respectively.

It will be observed that the expansion curve corresponds with the crystallization interval of Shepherd and Blough from 0 per cent. to about 23 per cent. of tin and from 40 to 65 per cent. of tin, while it agrees with Giolitti and Tavanti from 15 to 25 per cent. of tin and from 35 to 60 per cent. of tin. The agreement with Heycock and Neville is only pronounced between 40 and 65 per cent. of tin.

The part of the expansion curve which exhibits a marked divergence from the crystallization interval curve is that lying between 25 and 35 per

[1] *Engineering* (British), Nov. 3, 1911.

cent. of tin. This expansion is a very remarkable one, rising rapidly to about 30 per cent. and falling away even more rapidly. Fig. 6 will show that this part of the expansion curve corresponds very well with the solidus-liquidus distance according to Heycock and Neville. In passing it may be noticed that the hardness curve exhibits a maximum at 30 per cent. of tin.

Tobin Bronze.[1]—Otherwise called by the government authorities rolled naval brass to avoid the specification of proprietary alloys. This alloy consists of Cu 59 to 63 per cent., Zn 40 to 36 per cent., Sn 0.5 to 1.5 per cent.; while of the impurities, lead must not exceed 0.2 per cent., iron 0.06 per cent., with small quantities of other ingredients that the manufacturer puts in as in the case of Tobin bronze, and which are trade secrets.

The mechanical tests are that the bars shall be capable of being hammered hot to a fine point and bent cold to an angle of 120 degrees. Their tensile strength shall exceed 60,000 lb. per square inch, the yield-point shall be one-half the ultimate strain and the elongation in 2 in. shall exceed 25 per cent.

Monel Metal.[2]—This natural alloy of copper and nickel is described in MINERAL INDUSTRY, XVII, 670. John F. Thompson gives these further particulars. The battleship North Dakota is equipped with Monel metal propellers, three-bladed wheels 13 ft. diameter and of 15,000 lb. each. These have already had 12 months satisfactory service. Numerous other propellers have been manufactured. Besides the Pennsylvania Station, the New York and the Chicago and the Northwestern in Chicago, several office buildings have been roofed with sheets of this metal. An interesting development has been the use of sheets and rods in the mining industry for punched screens and for bolts and pump rods.

For physical tests we have hot-rolled rods, elastic limit 58,873 lb. per square inch; breaking strength 86,899 lb., elongation in 2 in., 40 per cent.; reduction of area 60 per cent. Sand castings have an elastic limit 37,427 lb. per square inch; breaking strength 78,236 lb., elongation in 2 in. 38 per cent.; reduction of area 34 per cent. The modulus of elasticity is given at 22 to 23 million lb. As regards electrical properties, its resistivity (temperature coefficient 0.0011 per degree F.) is 256 ohms per mill-foot; conductivity 4 per cent, copper being 100 per cent. The heat conductivity is but one-fifth that of copper.

Torsional tests give 31,796 lb. at the elastic limit and 79,053 lb. at ultimate load. A remarkable result follows tests at 1000° F., in which

[1] Specifications of U. S. Navy, Dept. 46, B 6.
[2] *Eng. and Min. Jrnl.*, XCI, 223.

Monel metal retains over 80 per cent. of its tensile strength at the elastic limit, while at that temperature steel retains but 7 per cent., manganese 60 per cent., while copper had lost 60 per cent. of its strength at 750° F.

Conductivity of Lake Copper of the First Melt.[1]—From the Kearsage Lode: Wolverine Mine 90 per cent., Mohawk Mine 75 per cent., Tecumseh Mine 96 per cent., Michigan Mine 100 per cent., Mathiessen's standard. From the South-Range lodes: Baltic, Champion, and Trimountain Mines, 60 to 70 per cent., Calumet and Hecla properties 95 per cent., the Isle Royale Mine 60 to 70 per cent. All these are refined to 97 per cent. for hard-drawn or to 97 per cent. for annealed wire.

Hydrometallurgy of Copper

Chloridizing Roasting of Pyrite "Cinder."[2]—Von Kothny gives the results of the laboratory investigation of the chloridizing roasting of the well-roasted residue or cinder resulting from the chalcopyrite used in making sulphuric acid. Copper (and gold and silver if present) are extracted from the chloridized roasted material by leaching; hence it is so executed as to give a maximum extraction of the above enumerated metals.

A typical analysis of cinder from Spanish chalcopyrite, containing a total of 4.41 per cent. Cu, shows copper as sulphate ($CuSO_4$) 2.22 per cent.; as oxide (CuO) 0.71; as sulphide (Cu_2S) 1.24; and as unchanged chalcopyrite ($Cu_2SFe_2S_3$) 0.24 per cent. Besides this there was present undecomposed iron sulphide (FeS_2) 2.35 per cent. and ferric sulphate ($Fe_2(SO_4)_3$) 0.59 per cent.

In effecting a chloridizing roast the cinder is ground with perhaps 10 per cent. of salt. To this ground cinder is added raw pyrite, so that there shall be, for example, a sulphur content of 1 1/2 times that of the copper. The charge is roasted in a muffle furnace.

The following reactions take place:

(1) $FeS_2+7O_2=Fe_2(SO_4)_3+SO_2$.
(2) $Fe_2(SO_4)_3+6NaCl=3Na_2SO_4+Fe_2Cl_6$.
(3) $2Cu_2S+5O_2=2CuSO_4+2CuO$.
(4) $Fe_2Cl_6+3CuO=Fe_2O_3+3CuCl_2$.
(5) $CuSO_4+2NaCl=CuCl_2+Na_2SO_4$.

Care must be taken that there is enough pyrite to convert all the copper oxide into copper chloride according to reactions (1) and (2). For a 4 per cent. cinder 7.5 per cent. salt is the right proportion. The most suitable temperature is between 500° and 600° C., and prolonged roasting should be avoided.

[1] Private Notes.
[2] *Zeitschrift fur die Gesamte Huttenkunde* (1911), 389.

Leaching Copper Ores in Place.—The practicability of recovery of copper from ore that still stands in place in the mine has been recently discussed. There are conditions where this can be done to advantage. These are:

(1) The copper-bearing ore should exist, as sulphide; for example, as pyrite carrying chalcopyrite ($CuFeS_2$) covellite (CuS) or better chalcocite (Cu_2S).

(2) The vein, lode or ore body should be 25 or more feet in width.

(3) The property should have been well opened up by shafts and drifts, or, putting it in another way, where we have a mine which has already been exploited and where low-grade ore is left standing.

A good supply of water must be available. This is run into the workings, the water taking up copper sulphate which has formed by contact with air and moisture. This dilute solution is then run through tanks containing scrap iron and precipitated. If well managed no more than 1 1/4 lb. of scrap iron or pig iron is needed to deposit 1 lb. of copper.

Leaching Applied to Copper Ores.[1]—W. L. Austin in a series of articles discusses the principles and methods of the leaching of oxidized and sulphide copper ores.

Of the sulphides he regards chalcocite as being an ore capable of successful treatment. He proposes to grind the ore to 10 mesh, agitate by causing an upward flow of the lixiviant (ferric chloride) in a conical-bottom tank, to filter the solution, to precipitate the copper with pig iron mixed with coke, and to regenerate the lixiviant with chlorine. On the basis of the treatment of 100 tons of ore per day he gives the cost per pound as follows:

	Cost per pound copper
Milling (grinding to 10 mesh) charging and discharging tanks, pumping solutions 50 cents per ton of ore treated	$0.014
Chlorine at 2 cents per pound	0.015
Iron $30 per ton delivered	0.015
Melting precipitate at $8 per ton	0.005
Freight, refining and selling	0.014
Repairs, renewals, office expenses, etc	0.013
	$0.076
Mining, open cut at 50 cents per ton	0.014
Total operating expenses	$0.090

No item has been added for interest on cost of plant nor for depreciation, but on the other hand no credit is given for precious metals recovered.

The Hoeffner Process.—The costs as given in *Eng. and Min. Jrnl.*, Dec. 19, 1896, operating on 5 to 10 per cent. copper, was $0.164 per

[1] *Mines and Minerals*, II, 5, 31, 68, 79, 119, 211, 241, 257, 281.

pound of copper produced. The process was used from 1879 to 1895 at the Vitkovice Iron Works, Moravia, Austria, to extract copper, silver and cobalt from burned pyrites. The extraction was as high as 91 per cent.

Extraction of Copper from Ore in Place.[1]—W. L. Austin having taken out a patent for the leaching of ores in place in a mine, J. Park Channing and Joseph Irving have detailed their experience in such work. The former describes the experience of Mr. Stanton at the Eureka Mine, Ducktown, Tennessee. In this mine the black copper zone was not much over a foot thick and it did not pay to mine it. The mine had, however, a shaft and several hundred feet of drifts, and a method was devised of permitting the mine to fill with water, then to pump out the water once a month and to precipitate the dissolved copper on scrap iron. In this way the copper in the zone was finally pretty thoroughly dissolved. Twenty years ago Mr. Irving used the method in a mine in the south of Spain. The river water was run into the mine, which had already been honeycombed with drifts and which had four shafts. The ore body was perhaps 60 to 100 ft. wide consisting of pyrite with some chalcopyrite. The water was pumped to flow over pig iron and 1 1/4 lb. of iron was used to precipitate 1 lb. of copper. Naturally, the water had to be watched to prevent its getting acid, which would have entailed a further consumption of iron. The pump had to be of gun-metal and the pipes were lined with lignum vitæ to lessen corrosion. Still the repairs, because of this, were a considerable item.

Daniel Henry in a later communication thinks that in an ore body in place the rate of dissolution will be slow. He gives figures as follows: Mine water containing more than 0.5 lb. copper per ton is unusual in a mine percolating at the rate of 500 per gallon per minute; this would equal 1500 lb. in 24 hours, worth $150. Contrary to Mr. Irving's experience, he thinks 2.5 to 3 lb. of iron would be needed to precipitate 1 lb. copper and so would be more expensive.

Leaching of Copper Ores.—Wm. E. Greenawalt has patented a hydrometallurgical process with electrolytic regeneration. It is suited to oxidized silicious ore. The ore is crushed and mixed with some salt and is leached in large vats with dilute hydrochloric acid. Cupric chloride is formed, and in the presence of other chlorides it reacts on and dissolves silver, so that we can count on an extraction of 80 to 90 per cent. of it. The cupric solution flowing from the vats is next saturated with sulphur dioxide obtained by roasting sulphide ore. This converts the cupric into cuprous chloride. The solution goes to electrolytic vats where the copper is deposited at the cathode while the chlorine is liberated at the anode.

[1] *Eng. and Min. Jour.*, XCI, 601, 649.

The liberated chlorine is exhausted into a scrubbing tower filled with coke down through which the solution trickles. Sulphur dioxide is forced upward through the tower, and with the chlorine forms hydrochloric and sulphuric acids. This is the solution which is returned to the leaching vats. Both the copper sulphate and the free sulphuric acid react on the common salt of the charge so that cupric and hydrochloric acid result. This cycle of solution, precipitation and regeneration is repeated indefinitely. The inventor gives the cost of plant and equipment for a 100-ton plant at $120,000, and the cost of treatment $1.96 per ton of ore treated.

Costs and Prices

Costs at the Wallaroo and Moonta M. & S. Co., South Australia.[1]—The smelting plant treated during 1910 44,271 long tons of ore, and of cement copper 704 tons, recovering 4855 tons of fine copper, 2011 oz. of silver and 1296 oz. of gold, besides 210 tons of bluestone and 4486 tons of sulphuric acid. To produce the acid it cost $3.44 per ton.

The costs per ton of ore were: Mining, $4.93; sorting and ore-dressing, $1.05; loading and freight to the smeltery, $0.22; smelting, $2.18; shipping the product and general expense, $0.21; making a total of $8.59. The costs per pound of fine copper were: Mining and maintenance, $0.0855; smelting, $0.0300; other expense, $0.0031; total, $0.1186. For the year, $0.126 was realized per pound of fine copper.

Cost of Calcining at Cananea.[2]—The calcining expense in the McDougal roasting furnace includes all costs from the time the railway delivered the roast ore to the bins until the calcines were delivered at the reverberatory furnaces. The period is from February to July, 1911, inclusive.

The material calcined consisted of 32,929 dry tons concentrates, 9590 tons fine sulphide ores and 762 tons of limestone, or a total of 43,281 dry tons, which yielded of calcine 35,533 tons; this being a shrinkage of 7748 tons or 17.9 per cent. For the analysis of the charge and calcines see supra. Following are the costs:

Distribution of Costs	Amount.	Per Dry Ton.
Sampling	$ 222.62	$0.0051
Bedding	2,016.27	0.0466
Reclaiming	3,072.91	0.0710
Operating furnaces	7,680.58	0.1775
Hauling calcines	911.01	0.0210
General expense	1,639.99	0.0379
Total direct costs	$15,543.38	$0.3591
Flux (cost of)	1,097.85	0.0254
Total costs	$16,641.23	$0.3845

[1] *Eng. and Min. Jrnl.*, XCI, 1066.
[2] *Eng. and Min. Journal*, XCII, 693.

Analysis of Cost	Amount.	Per Dry Ton.
1. Operating:		
Labor	$7,398.56	$0.1709
Power	1,577.64	0.0365
Fuel	773.06	0.0179
Water	78.71	0.0018
Sundries	10.30	0.0002
Flux	1,097.85	0.0254
	$10,936.14	$0.2527
2. Repairs:		
Labor	$2,507.35	$0.0579
Shop expense	361.00	0.0083
Supplies	2,836.74	0.0656
Total	$16,641.23	$0.3845

Power Costs at the Copper Queen Smeltery.[1]—The central power plant at Bisbee used 8168.1 gallons of fuel-oil per day in 1910, which raised 705,969.2 lb. steam or 86.5 lb. of steam per pound of oil. At the Douglass power plant the costs per horse-power-year were: Labor, $9.23; tools and supplies, $1.49; repairs, $7.72; fuel $53.15 (at a cost of $1.70 per barrel); water, $0.61. This was on an average development of 3596 h. p. At the Douglass smeltery the power was distributed as follows: Blast-furnace blowing engines, 34.93 per cent.; converter blowing engines, 34.85; electrical power, 25.97, and compressed air, 4.25 per cent.

Reverberatory Costs at Cananea.[2]—In keeping the account of costs of reverberatory smelting, all expense of operating and maintaining the reverberatory boilers and the economizers was charged to the reverberatory department, while to it was credited the heat recovered as steam on the basis of the net evaporation obtained with a pound of oil burned under the power-house boilers, viz., 13.6 lb. of water. This credit includes the heat recovered at the economizers, from which a part only of the water is fed to the reverberatory boilers, the balance being delivered to the power-house boilers.

These costs from February to July, 1911, inclusive in 312 1/2 days, using of flue dust, 21,109 tons; of calcines, 35,533 tons; of ores, 3040 tons; of limestone, 479 tons, or a total of 60,071 tons, was thus distributed:

	Amount.	Per Dry Ton.
Operating furnaces	$111,687.17	$1.8593
Slag and matte expense	5,111.07	0.0851
Boiler house	11,468.77	0.1909
General expense	4,218.58	0.0702
Cost of flux	817.46	0.0136
	$133,303.05	$2.2191
Steam credit	48,861.86	0.8134
	$ 84,441.19	$1.4057

[1] Report of Copper Queen Smeltery Co., 1910.
[2] *Eng. and Min. Jrnl.*, XCII, 694.

	Analysis of Cost	
(1) Operating:		
Labor	$ 17,829.42	$0.2968
Power	529.36	0.0099
Fuel-oil	88,028.99	1.4654
Coal	243.61	0.0041
Water	91.68	0.0015
Transportation	380.45	0.0063
Sundries	315.64	0.0053
Flux	817.46	0.0136
	$108,299.61	$1.8029
(2) Repairs:		
Labor	$ 11,063.93	$0.1842
Supplies	12,425.30	0.2068
Shop expense	1,514.21	0.0252
	$133,303.05	$2.2191
Steam credit	48,861.86	0.8134
	$ 84,441.19	$1.4057

Smelting Costs of Tennessee Copper Co.[1]—These costs are thus given: Smelting, $1.227 per ton. This is based on a recovery of 29.3 lb. per ton of ore on an output of 405,463 tons mined, or $0.042 per lb. of copper.

Prices of Copper Ores in Utah in 1911.—The basis price for treatment was $2.50 to $3 with a charge of 10 cents per unit for silica and an allowance of 5 cents per unit for the iron. Copper was paid for in lead-bearing ores at 5 cents less than the New York quotations for electrolytic copper, while the copper in lead-free ore took a deduction of 2 1/2 cents per pound for all over 1 per cent. Cu. Gold was paid for at $19 per ounce and silver at 95 per cent. of the New York price for the same.

Smelter Prices for Copper Ores.[2]—The subjoined schedule of prices is quoted by western smelteries as paid for ores of copper:

In California, Smeltery A:

Gold, $19 per ounce, if under 0.5 oz. $19.50 per ounce, for all over 0.5 oz. per ton; silver 90 per cent. of the New York quotation. Copper, wet assay less 1.3 per cent. at New York quotation for electrolytic copper, less 4 cents per pound. Treatment charge $4 per dry ton with an additional briquetting charge of $1 per ton for fine concentrates and ores.

In California, Smeltery B:

Gold at $20 for all over 0.5 oz., no pay for gold less than 0.5 oz. per ton. Silver at 95 per cent. of New York quotation. Copper, wet assay less 0.75 per cent. at New York quotation for electrolytic, less 3 cents per pound. Treatment $4 per ton.

In California, Smeltery C:

Gold at $19 per ounce. Silver at 95 per cent. of New York quotation. Copper, wet assay less 1.3 per cent. at New York quotation, less 3 cents per pound. Treatment $4 per ton up to $50 value and 25 cents per ton for each increase in value of $5.

[1] Annual Report of the Tennessee Copper Co., 1911.
[2] *Eng. and Min. Jrnl.*, XCII, 316.

In British Columbia, Smeltery D:

Gold at $19 per ounce. Silver 90 per cent. of New York quotation. Copper, wet assay less 1.3 per cent. at New York quotation, less 3 cents per pound. Treatment $2 per ton.

Southwestern Plant, Smeltery E:

Gold $19 per ounce for all over 1 oz. per ton. Silver 95 per cent. of New York quotation. Copper 90 per cent. of wet assay for all over 3 per cent. at New York quotation for electrolytic, less 3 cents per pound.

Utah Plants, Smelteries G, H and I:

Gold at $19 per ounce for all over 0.02 oz. per ton. Silver at 95 per cent. of New York quotation. Copper at these rates: For Smeltery G, wet assay less 1.3 per cent. at New York quotation, less 3 cents if ore runs 1.4 to 5 per cent.; less 4 1/2 cents for ore 5 to 10 per cent.; less 4 cents for ore of 10 to 20 per cent.; less 3 cents for all ore over 30 per cent. For Smeltery H, wet assay less 1 per cent. New York quotation, less 1 per cent. less 2 1/2 cents per pound. For Smeltery I, wet assay less 0.85 per cent. at New York quotation for electrolytic cathodes less 2 1/2 per cent. The insoluble contents are penalized at 7 to 12 cents and iron paid for at 7 to 10 cents per unit. Treatment at Smeltery H, $2.30 per ton. At Smeltery I, $2.25 with the addition, of $2 per wet ton, a total of $4.25 for freight and treatment per dry ton.

COPPERAS

The sulphate of iron production in the United States in 1911 amounted to 35,501 short tons valued at $390,500 compared with 38,896 short tons in 1910 valued at $427,856 and 42,225 short tons valued at $464,475 in 1909. This chemical is obtained as a by-product in various industries such as the pickling plants of the sheet iron and steel and wire concerns where sulphuric acid is used preparatory to galvanizing or tinning. The largest producer, for this reason, is the United States Steel Corporation which supplies the greater part of the output. Other producers are the Pennsylvania Salt Company, Philadelphia, Penna., The Stauffer Chemical Company, San Francisco, Cal., S. P. Wetherill Company, Philadelphia, Penna., Wickwire Brothers, Cortland, N. Y., E. I. Du Pont Nemours Powder Company, Wilmington, Delaware and the Westmoreland Chemical Company, Philadelphia, Penna. The Du Pont Powder Company recovers copperas as a by-product in the manufacture of dynamite.

The table shows the production of copperas in the United States for the last 15 years. A large portion of the copperas produced by the manufacturers is converted by them at once into Venetian red which is used for paint makers and also to some extent in the production of rouge.

PRODUCTION OF COPPERAS IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Year.	Short Tons.	Value.	Year.	Short Tons.	Value.
1897.....	11,924	$56,565	1902.....	19,781	$118,474	1907.....	26,771	$294,481
1898.....	11,285	58,105	1903.....	20,240	121,440	1908.....	35,334	388,674
1899.....	13,770	108,508	1904.....	16,956	118,692	1909.....	42,225	464,475
1900.....	12,374	96,517	1905.....	21,103	147,721	1910.....	38,896	427,856
1901.....	23,586	112,336	1906.....	22,839	228,390	1911.....	35,501	390,500

The above table does not include this production of copperas. The average price during 1911 was lower than in 1910. In 1910, the New York quotations were 55 cents per 100 lb. of copperas in bulk, 65 to 75 cents per 100 lb. in barrels and 60 to 70 cents per 100 lb. sacked. The price in 1911 ranged from 40 to 55 cents per 100 lb. in bulk.

The following newer uses of copperas are given together with a list of those that are usual.

Water Purification.—For this purpose the sugar sulphate of iron is dissolved in water to a known strength solution which is injected into the

water to be treated by various types of mechanical devices at a uniform rate for the purpose of securing a uniform application, after which a lime water, milk of lime, or some other caustic solution is injected into the water in uniform quantities and at a uniform rate and at a point where there is sufficient agitation of the water with both chemicals to result in a complete mixture of the three. The result is the formation of a gelatinous coagulant which precipitates to the bottom of a settling basin, entangling the bacteria, or which passes onto the filter beds to form a coating through which the treated water must pass in the process of filtration, thus entangling the bacteria at that point and resulting in an almost perfect purification of the treated water.

Sewage Purification.—The treatment for this purpose is very much similar to that of water purification.

Weed Eradication.—This business is yet in its infancy, but the use of sugar sulphate of iron on the farm for this purpose has been demonstrated to be of practical value to the farmer. The process consists of dissolving the sugar sulphate of iron in water to make approximately a 20 per cent. solution, variable, however, with weather conditions, and applying the solutions to the weed-infested field in the form of a very fine mist or fog with a traction power spraying machine having a spray boom 12 to 20 ft. in length, applying approximately 52 gallons per acre. The process is applicable to a large variety of noxious weeds, but not to all weeds.

Manufacturing Fertilizer.—Meat packing companies convert waste products into fertilizer by using their tankage which contains solutions rich in animal juices and evaporating them. If evaporated without the addition of sulphate of iron, the resulting product absorbs moisture and becomes sticky.

Vanadium Industry.—A considerable quantity of sulphate of iron is consumed in the production of vanadium oxide.

Horticulture.—As a preventive and cure for chlorosis of grapes, raspberries, pineapples and fruit trees, it is made into varying strengths of solutions and the affected tree or shrub is sprayed or washed with this solution. It is also added to bordeaux mixture, the lime and sulphur wash, and other mixtures to act as a sticker, thus giving adhesive quality to balance the solution.

Gas Purification.—Sulphate of iron is dissolved in water and the solution is used in a washing tower to remove the cyanide in gas made from coal. It is converted into a spongy hydrated oxide of iron by being precipitated out of solution with a caustic.

Other Common Uses for Which Processes are Fully Known.—The preparation of rouil used as a mordant in fabric dyeing, manufacturing

of oxide of iron pigments, paper making, manufacturing of Prussian blue, inks, shoe blacking, photography, fuming sulphuric acid, manufacture of ferric sulphate, condimental stock foods, manufacture of rouge for plate glass polishing, manufacture of drugs and medicines, and leather tanning for color. In the manufacture of rouge and pigments, the copperas solution is simply precipitated by sodium carbonate. The resulting carbonate is filter pressed and then slowly dried, converting most of it into magnetic oxide Fe_3O_4.

CORUNDUM AND EMERY

These natural abrasives are still preferred, for certain purposes, to the artificial products alundum, carborundum, etc. The mineral corundum is the crystallized oxide of aluminum, having a hardness of 9; emery is a rock (sometimes disintegrated) consisting of grains of corundum, garnet, magnetite, etc., its abrasive power being in proportion to the percentage of corundum in its composition. In the United States, corundum has been mined on a small scale in Gallatin County, Mont., Rabun County, Ga., and Macon County, N. C., but production has ceased in recent years. Emery is regularly produced at Chester, Mass. and Peekskill, N. Y., but the output from these mines supplies only a small fraction of the total consumption in the United States, the remainder being imported from Greece and Asia Minor. Corundum is imported from Ontario. Both corundum and emery are now on the free list.

STATISTICS OF CORUNDUM AND EMERY IN THE UNITED STATES.

Year.	Production. (a)		Imports.				
			Grains.		Ore and Rock.		Other Mfrs.
	Short Tons.	Value. (b)	Pounds.	Value.	Long Tons.	Value.	Value.
1898	3,742	$ 207,430	577,655	$23,320	5,547	$106,269	$3,810
1899	3,970	228,570	728,229	29,124	7,435	116,493	11,514
1900	5,030	247,100	661,482	26,520	11,392	202,980	10,006
1901	4,305	146,040	1,086,729	43,217	12,441	240,856	10,926
1902	4,251	104,605	1,665,737	49,107	7,157	151,959	13,776
1903	2,542	64,102	3,595,239	109,272	10,884	194,468	17,829
1904	1,932	57,235	2,281,193	109,772	7,054	138,931	11,721
1905	(c)2,315	19,677	3,209,914	143,729	11,072	185,689	17,996
1906	(c)2,147	22,780	4,655,168	215,357	13,840	286,386	19,105
1907	(c)1,069	12,294	4,282,228	186,156	11,235	211,184	15,282
1908	(c) 669	8,745	1,735,366	89,702	8,084	146,105	12,592
1909	(c)1,580	18,185	2,696,960	132,264	9,836	186,930	19,803
1910	(c)1,028	15,077	1,189,664	53,709	16,321	344,421	.....
1911	(c) 769	8,810	712,134	35,384	10,232	176,890	.....

(a) Statistics of the United States Geological Survey for 1901–1903. (b) Values have not much significance owing to the wide variation in the quality of the materials combined in the totals. (c) Emery only.

Canada.—The principal output of corundum comes from the Craigmont district, Renfrew County, Ontario. Of the total 1911 production, 742 tons, or about one-half, was exported. New developments in the Bancroft area of Ontario are described by F. D. Adams and A. E. Barlow in *Memoir* No. 6 of the Canadian Geological Survey. The principal occurrences lie in a belt of syenitic rocks extending from Lutterworth to Clear

Lake; a smaller exposure is in Methuen Township. The principal development has occurred in the northeast corner of Carlow Township, where the Ashland Emery and Corundum Co. is actively engaged in exploratory work, and has already erected a mill. The Canada Corundum Co. is similarly engaged in lot 5, concession I, of Monteagle Township. On lot 13, of the same concession, the National Corundum Wheel Co., of Buffalo, has also undertaken to prospect a deposit.

PRODUCTION OF CORUNDUM IN CANADA.

Year.	Short Tons.	Value.	Year.	Short Tons.	Value.	Year.	Short Tons.	Value.
1900....	60	$ 6,000	1905..	1,644	$149,153	1910..	1,870	$198,680
1901....	534	53,115	1906..	2,274	204,973	1911..	1,472	161,873
1902....	1,137	83,871	1907..	1,892	177,922			
1903....	1,119	87,600	1908..	1,089	100,398			
1904....	1,665	150,645	1909..	1,491	157,398			

Greece.—The production of emery on the island of Naxos is under Government control, official inspectors being stationed at Syra, the only export harbor, to classify and weigh the output. The production during 1909 was 10,965 metric tons, of which only 8193 tons were sold; in 1910, the mines yielded only 8000 tons, while stocks were drawn upon to afford shipments amounting to 12,939 tons. The average price in 1910 was $19.71 per metric ton.

EXPORTS OF EMERY FROM NAXOS, GREECE.

Year.	Metric Tons.	Value.	Year.	Metric Tons.	Value.	Year.	Metric Tons.	Value.
1897....	3,125	$ 65,683	1903..	5,813	$120,348	1909..	8,193	$164,762
1898....	4,500	93,166	1904..	6,353	131,531	1910..	12,939	255,053
1899....	5,139	106,181	1905..	6,395	132,090	1911..	.....	.....
1900....	6,023	124,503	1906..	8,030	166,251			
1901....	6,080	125,582	1907..	10,982	221,154			
1902....	4,315	88,841	1908..	7,471	164,520			

The emery deposits of the Grecian archipelago, according to M. Papavasiliou, are associated with massive deposits of limestone and dolomite, which constitute the principal islands. Three workable zones are distinguishable, of which the lowermost is the most productive, being extensively operated at Naxos. The emery of Naxos is very pure, and occurs in compact granulated mass, in which the crystallized alumina is mingled with magnetite and a little hematite.

India.—The production of corundum in 1910 amounted to 137 short tons, valued at £634, of which 52.7 tons came from Rewa and 84.3 tons from Madras. In 1909, most of the output, 14 tons, came from Rewa. The occurrence of the mineral at those recently developed producing centers is described in Vol. XVII of THE MINERAL INDUSTRY.

CRYOLITE

At the Oresund Chemical Works in Copenhagen, a separation of cryolite from the associated minerals is carried out on jigs and magnetic separators. There is only one occurrence of this mineral which has been developed commercially. This is at Ivigtut, Greenland, latitude 61, where the mineral occurs associated with quartz, siderite, blende, galena and chalcopyrite. Fluorspar when present causes a pink tint instead of the pure white of the cryolite. Of this fluorspar radio-activity is characteristic; it also contains some yttrium and the pink coloration disappears on heating. The deposit at Ivigtut is a large one and extends 600 by 200 ft. on the surface, widening at depth which, in some places where open cut mining is carried on, is vertically 150 ft. Inclusions of granite are found imbedded in the ore.

Cryolite was first successfully used by Julius Thomson in 1852. The plant was at Copenhagen where soda was manufactured by the heating of this mineral with lime and then passing carbonic acid through the solution. The by-product calcium-fluoride was sold to the glass makers. The alumina precipitated in the process was ignited to oxide but later converted into alum. The Solvay process replaced this as a soda process although the Pennsylvania Salt Mfg. Co. acquired the United States rights and built a plant at Natrona, Pa. To this day, the Pennsylvania Salt Company is the sole importer of cryolite. The process certainly gives the purest alumina whether it be used for concentrated alum or for metallic aluminum. The imports of cryolite into the United States for 1910 are reported as 36 long tons valued at $2343 as compared with 1278 long tons valued at $18427, in 1909. In 1911 the imports were 2007 long tons, valued at $47,093. Cryolite enters free of duty.

After the abandonment of the soda process, especially in Europe, the uses found for cryolite were much as they are to-day, namely, milk-glass, for the enameling of iron ware and in the metallurgy of aluminium. The ingredients that are commonly used for opaque glass are calcium phosphate, fluorspar and tin oxide. It has been found that in using cryolite, only 10 per cent. of the weight of the glass was required to produce the same effect. The action is by the separating out of minute crystals in the body of the glass. The composition of these mixtures that were used for enameling cast iron and pressed steel articles consisted principally of a mixture of feldspar, soda, quartz, borax, etc. This gave a more or

less transparent covering like the ordinary granite ware. The addition of 5 to 10 per cent. of cryolite makes this non-transparent. One advantage of the cryolite enamels is that they are not poisonous and therefore particularly suited for kitchen work. Fluorspar appears to produce a similar result but renders the mixture less fusible. A substitute for cryolite that is being tried in Germany is sodium fluo-silicate, a by-product obtained in the phosphate works from the gases, as these gases are injurious to the health of the workmen. A law compelled manufacturers to neutralize the gases and they were therefore absorbed in water and mixed with bauxite giving the sodium fluo-silicate. This material, however, loses 30 to 40 per cent. of its weight on fusion. In the production of aluminium in the original Héroult method, the bauxite is added to a fused bath of cryolite and it is said that ten parts of aluminium require one of cryolite, in this process. Further, the mineral must be pure and nearly free from silicate which if introduced would allow the formation of silica and therefore produce a brittle alloy with the aluminium. The United States Geological Survey report of 1909 states that cryolite is used for ornamental purposes being employed as a cement. In this white cement, the silica and the alumina must be properly proportioned and only 2/10 per cent. of iron oxide is allowable to prevent staining. From 2 to 5 per cent. of cryolite is used in this cement.

The separation of the minerals in preparing cryolite for market consists in hand picking of the pure lumps and the screening of the remainder so that the larger impure lumps can be cleaned by means of mechanically driven chisels. The smaller impure pieces from the outside layers are added to the smaller screenings and together subjected to a jigging process. The size of the smallest material jigged in this way is from 1 mm. down to the finest dust and between 14 and 1 mm. for the larger size. The two classes produced in each case are pure cryolite and nearly pure gangue, that is, considering galena and heavier minerals as gangue. The products are dried and then separated successively on magnetic concentrators, the iron-bearing minerals being most attracted and therefore first obtained. The various minerals, in fact, are separated as by-products. The galena and chalcopyrite containing gold and silver are sent to a smelter. Siderite is shipped to the steel plants.

The chemical composition of cryolite is a fluoride of sodium and aluminium. It has a fusing point of about 1000° C. In the final preparation for market the purified cryolite obtained by a second treatment on the magnetic machines is ground to a fine powder allowing some of the quartz still remaining to be eliminated as it does not grind as readily as the softer cryolite. We find cryolite sells in large lots at about 6 1/2 to 7 cents per pound.

DIATOMACEOUS EARTH

This material which is commonly or erroneously called "infusorial earth" was formerly used to a great extent as an absorbent substance in the manufacture of dynamite which use for this purpose at the present time is rather limited. Probably the largest deposit in the world is at Lompoc, Cal. These deposits contain no iron or other impurities which for some purposes is essential. A deposit of diatomite, as it is also technically known, was discovered in Nebraska, but it has not been developed. As reported by Consul W. L. Lowrie, the Carlsbad district in Austria has a number of deposits of this siliceous material in the bogs where the mud is obtained for the famous "mud baths" of the Bohemian health resorts. In the vicinity of Soos and at Franzensbad, deposits are owned by the Government. They are, however, of no commercial value as no prices are quoted and the quality has not been determined by analysis. It is used in small quantities, being molded, burned or otherwise treated for the manufacture of pottery and statuary, gold edges, papier maché, dynamite, putty, adulterants for soap, caoutchouc and carbolic acid preparations as it is a non-conductor of heat and electricity, it is employed as a filling in the walls of iron chests and ice houses and other insulating purposes. This Kieselguhr as it is known in Germany is also used for water purification in the Bergfeld filters. In America, the chief use at the present time is probably for polishing powder as in silver polishes or as an abrasive. It is also used to some extent for filtering water and as a non-conductor in compounds, as on boilers, steam pipes, etc. It is being tried in records for talking machines.

For most uses the material is dried at a high temperature and ground.

The distribution of the material is almost universal as the diatoms or minute living organisms that leave the siliceous skeletons live in fresh or salt water, hot or cold. They act as water purifiers with the aid of the action of sunlight giving off oxygen obtained from the carbonic acid of the air.

The production of this substance together with tripoli, valued at $147,500 in 1911. On the Isle of Skye, the only deposit in the United Kindom was worked by the British Diatomite Co., in 1909, producing only 240 tons valued at about $600. The deposit is extensive and about 40 ft. in thickness.

California.—The production in the last 2 years was

in 1911.....2,194 tons valued at $19,670
in 1910.....1,843 tons valued at $17,617.

This is from Monterey and Santa Barbara Counties.

FELDSPAR

The production of feldspar in the United States in 1910 amounted to 80,822 tons valued at $501,451 and in 1911 it was 89,552 tons valued at $560,851. The principal producers are Maine, Pennsylvania, New York, Connecticut and Maryland.

FELDSPAR IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Crude.		Ground.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1901	9,960	$ 21,699	24,781	$198,753	34,741	$220,422
1902	21,870	55,501	23,417	194,923	45,287	250,424
1903	13,432	51,036	28,459	205,697	41,891	256,733
1904	19,413	66,714	25,775	199,612	45,188	266,326
1905	14,517	57,976	20,902	168,181	35,419	226,157
1906	39,976	132,643	32,680	268,888	72,656	401,531
1907	31,080	101,816	60,719	457,128	91,799	558,944
1908	18,840	65,780	51,634	362,773	70,474	428,553
1909	31,037	85,421	45,502	316,367	78,707	415,131
1910	24,655	81,965	56,167	419,486	80,822	501,451
1911	28,131	88,394	64,569	490,614	92,700	579,008

(a) Statistics reported by the U. S. Geological Survey for 1911; other years the same, except for New York.

PRODUCTION OF FELDSPAR BY STATES.
(Crude and Ground. In tons of 2000 lb.)

	1909 (a)		1910 (a)		1911 (a)	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Connecticut	13,644	$ 43,013	18,739	$ 89,974	16,497	$ 73,557
Maine	18,137	166,031	20,762	196,463	25,976	246,305
Maryland	6,564	35,399	9,977	47,340	12,694	56,265
New York (b)	13,871	46,444	12,132	46,863	15,652	61,769
Pennsylvania	18,573	111,466	15,091	104,751	13,284	101,090
Other States	8,018	12,778	4,121	16,060	5,449	21,865
Total	78,707	415,131	80,822	501,451	89,552	560,851

(a) U. S. Geological Survey, except for New York. (b) New York State Geological Survey.

In the State of New York, as reported by D. H. Newland, Assistant State Geologist, there was a slight increase in the production although the market for pottery and roofing grades followed about the same course as in the preceding year. The total production of the state amounted to 15,652 short tons, valued at $61,769 for 1911, as compared with 12,132 short tons, valued at $46,863 and 13,871 short tons, valued at $46,444, in the years 1910 and 1909 respectively. The New York material is mainly crushed waste from the feldspar industry proper and is sold under this head and used as a roofing material. This has

become more or less a staple article and shippers near Crown Point continue in this industry. There are several producers of pottery feldspar namely at Bedford, Westchester County, and at Batchellerville, Saratoga County.

At Chester, Vermont, feldspar has recently been opened by A. L. Stone and mining is progressing. In the State of California, feldspar is a new industry, the output being from Monterey and Tulare Counties. Most of this potash feldspar is made use of in the pottery industry and in 1911, 740 tons valued at $4560 were produced. The output in 1910 was 760 tons valued at $5720. In Connecticut the chief producers kept up a steady production; the average value was about $3.20. In Maine the chief producer, the Maine Feldspar Company at Littlefield, near Auburn, increased production and grade. In Maryland the production, which all goes to the pottery centers—Trenton, N. J., and Wilmington, Del.—for grinding, was small. From Pennsylvania, usually a large producer, there are no advices to show prosperity in this industry. In 1910 a quarry was opened and in 1911 a mill of 6000 tons yearly capacity was erected near Edgmont, by the Edgmont Feldspar Co. of Philadelphia. In Virginia there are only three or four quarries worked for feldspar; the operators are the Bedford Spar Co., Bells; Dominion State Mines Corporation, Prospect; J. M. Patterson, Amelia Co.; and Schlegal, Jetersville.

Canada.—Canada is one of the most important producers of feldspar. The quantity produced in 1911 was 17,718 tons valued at $51,924. The selling price of this material at Trenton, N. J., was $5 per long ton. It contains nearly three-fourths in the form of potash feldspar. The ground feldspar of the same sort brings $10.50 per short ton and the No. 2 Canadian crude sells at $5 to $5.25, or when ground from $9 to $9.50 per short ton. The American grades do not sell at so high a figure.

Sweden.—The *Bulletin* of the Geological Institute of Upsala, Vol. IX, 1910, gives a good review of the feldspar industry in Sweden since 1783. The total yield of these mines from 1864 to 1908 was 25,453 tons of potash feldspar and 3013 tons of soda feldspar.

United Kingdom.—China clay, a semi-decomposed feldspar, now has an output of 800,000 tons, an increase of nearly 25,000 tons over 1910, and this output, would have been greater if sailing vessels had not been prevented by storms from sailing during the last quarter. Prices have also ruled higher, and the pay of the laborers increased. The china stone output remains stationary at about the 70,000-ton mark, as the Norwegian feldspar is a keen competitor and much cheaper.

Uses.—The uses of feldspar are primarily for pottery, enamel ware, enamel bricks and electrical supplies. Carefully selected feldspar is

used in the manufacture of artificial teeth and for scouring soaps and window washes. The use for roofing sand is confined to a low-grade material that is not suitable for other uses. Another use is for abrasives. The addition of feldspar to china clays within the range used in pottery causes decreased porosity, the vitrification being complete and more rapid with fine grained spar. If coarser than 150-mesh, no amount gives a vitrified body for the usual temperatures used and contraction is irregular. Twenty per cent. of 200-mesh spar causes the same vitrification as 40 per cent. of 40-mesh.[1] The limits of pottery body compositions, according to R. C. Purdy[2] are 12 parts of clay, 9 of flint, and 4 of feldspar. Sufficient clay must be present in order to keep plasticity, but not too much for excessive shrinkage; also feldspar sufficient to give vitrification but not so high as to cause warping. The flint and china clay should be as high as possible to give surface texture and white color. Artificial teeth in the United States are colored only on the surface. In England and Germany, the practice to use homogeneous mass, white throughout is favored. A mixture of 20 per cent. quartz and 80 per cent. feldspar previously at temperature of cone nine, then finely ground, to which is added up to 5 per cent. bone ash to cause natural appearance, is used. The stain made of various oxides previously prepared such as TiO_2, NiO, CaO, Fe, etc., are added to the mixture in small quantities and it is then fired at a somewhat higher temperature.

Of about nine species of this mineral, which constitutes probably 58 per cent. in the composition of the earth's surface, only a few are of commercial value. The principal ones are the potash feldspars, orthoclase and microcline, which are the kinds commonly used in pottery. These potash feldspars are frequently associated with those containing soda, such as albite. Commercial feldspar usually is associated with quartz and mica in pegmatite dikes, a coarse rock occurring in veins and which is the common source of this mineral either alone or as a by-product. There is an abundance of such deposits and those suitable are naturally mined where the transportation is good to the large pottery centers. The potash feldspar is sufficiently high in potassium and also sufficiently abundant to stimulate search for a cheap method of extracting the potash, but as yet this has not been successful.

The United States Department of Agriculture, *Bul.* 30, Bureau of Soils, gives a reference list showing the use of feldspar as fertilizer. Bureau of Plant Industry, in *Bul.* 104, entitled "The Use of Feldspathic Rocks as Fertilizers" gives a description of experiments made here and abroad. For other particulars on the use of feldspar for potash see potassium salts. E. S. Bastier,[3] gives geological, chemistry, etc.

[1] *Trans.* Am. Ceram. Soc., 1911, 13, 731.
[2] T. A. Cer. *Ibid.*, p. Soc., 1911, 13, 86.
[3] Bulletins 420 and 445, U. S. Geol. Survey, 1912 and 1911.

FLUORSPAR

There were no new developments in the production of fluorspar in the United States in 1911 and the production did not increase materially. The largest production, as before, came from Southern Illinois and Kentucky. Nearly all of the fluorspar from this district is used as a flux in the open-hearth steel furnaces of the Pittsburgh and Birmingham districts. In the West, fluorspar was mined near Jamestown, Colorado and near Deming, New Mexico. Most of this production is consumed by the Colorado Fuel and Iron Company's plant at Pueblo. Other western works have been enlarged so that a larger field can be supplied if it is merely a question of transportation.

Fluorspar is chemically a calcium fluoride containing 48.9 per cent. fluorine and is used principally as a flux in the basic steel processes and has also partly replaced cryolite in the manufacture of aluminum. If smelted with tellurium gold ores, it is stated that it prevents volatilization. Other uses are as a bond in the manufacture of emery wheels and in carbon electrodes in which latter it increases the lighting efficiency and decreases the current. The mineral fuses and flows at the ordinary temperature of the assay furnace but the lime is not decomposed to form a silicate. In open hearth work the fluorspar serves as a desulphurizing agent and should be free from barite which is much more commonly present and more objectionable than silicates or carbonates. Up to 10 per cent. barite may be present with only 1.7 per cent. sulphur. Methods of analysis are given by H. G. Martin.[1]

An idea of the importance of fluorspar is obtained from the United States imports for the fiscal year to June 30, 1910, when 16,561 tons worth $60,725, and to June 30, 1911, when 44,004 tons worth $148,548, were imported. The duty is $3 per ton and is therefore between 80 and 90 per cent. advalorem. The value of the imported product per unit fell from $3.67 to $3.37. Before 1909, when the duty was imposed, fluorspar from foreign countries, primarily England, was able to compete with that used in the Pittsburgh and Birmingham districts, and the imports are sufficient at present to show that, to a considerable extent, this competition is still taking place.

The fluorspar imported from England is derived from waste dumps and is obtained at very little expense. A few years ago this did not exceed a cost of $2.31 per ton at Liverpool. As the material was generally carried as ballast, the freight rate to American ports was very low and

[1] *Journal of Industrial and Engineering Chemistry*, Vol. 1, page 463.

even with the addition of railroad freight and the tariff, was able to compete with the American product at eastern points. Many Fluorspar mines in South Durham and Derbyshire were idle and the production was only about half that of 1910. Few orders from the United States were given. The fluorspar mining in the United States must be carried on very efficiently under these circumstances in order to pay. In the West the veins are comparatively narrow and the workings scattered. The imports prior to the tariff were estimated at 30,000 to 100,000 tons.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES.
(In short tons.)

Year.	Mined.	Value.	Per Ton.	Year.	Mined.	Marketed.	Value.	Per Ton.
1901 (*a*)	19,586	$113,803	$5.81	1906 (*b*)	31,592	37,034	$211,231	$5.70
1902 (*a*)	48,018	271,832	5.19	1907 (*b*)	39,777	38,922	223,308	5.76
1903 (*a*)	42,523	213,617	4.28	1908 (*b*)	46,677	38,795	225,998	5.83
1904 (*a*)	36,452	234,755	6.44	1909 (*b*)	45,801	50,843	301,704	5.94
1905 (*b*)	38,918	228,543	6.32	1910 (*a*)	.....	69,427	430,196	6.20
				1911 (*c*)	47,000	.....	.....	.....

(*a*) Statistics of U. S. Geological Survey. (*b*) Statistics collected by F. J. Fohs. The figures for 1905 to 1908 inclusive differ slightly from those obtained independently and reported by THE MINERAL INDUSTRY for these years. (*c*) Estimated.

FLUORSPAR OUTPUT OF THE PRINCIPAL PRODUCING COUNTRIES.
(In metric tons.)

Year.	Austria-Hungary.	France.	Germany.	Spain.	United Kingdom.	United States.
1898	4,169	3,077	23,787	5	57	11,021
1899	4,959	5,140	24,306	310	796	21,806
1900	5,649	3,430	30,310	4	1,471	19,646
1901	5,774	3,970	28,741	*Nil.*	4,232	17,768
1902	5,902	2,650	(*a*)14,177	93	6,388	47,190
1903	5,445	2,447	(*a*)13,028	4,000	12,102	38,577
1904	7,061	2,047	(*a*)13,540	(*b*)	18,451	33,069
1905	7,601	2,434	(*a*)15,019	(*b*)	40,079	(*c*)35,299
1906	7,795	4,218	(*a*)15,493	70	42,521	(*c*)28,657
1907	8,779	4,795	(*a*)16,624	270	50,257	(*c*)32,969
1908	7,359	5,456	(*a*)14,925	253	35,257	(*c*)35,738
1909	8,106	5,725	(*a*)14,545	246	43,165	(*d*)41,550
1910	(*e*)8,000	8,264	(*a*)17,988	180	62,607	63,000
1911	.....	.....	(*a*)23,073	.....	32,100	(*e*)42,300

(*a*) Exports. German statistics no longer report production. (*b*) Not reported. (*c*) These statistics collected by THE MINERAL INDUSTRY differ slightly from those reported for the same year by F. J. Fohs in the preceding table. (*d*) Amount mined. (*e*) Estimated.

Illinois.—The principal producers of fluorspar are the Fairview Flourspar and Lead Company at Golconda, Illinois. The market price of the output in Illinois in 1911 was an average of $6.80 as compared with an average of $5.50 in 1910. The various grades were quoted relatively at $7.50 for gravel, $9 for lump and from $12 to $15 for ground. The proportion of gravel produced in 1911 was less than in 1910 when 80 per cent. of the product was gravel. In the Fairview and Rosiclare shafts the vein is proved to a depth of 500 ft. and 335 ft. respectively. The Rosiclare mine is the property of J. R. McLean, Cincinnati, and has been operated for 70 years. For 3 years the product has been distributed

by Rogers-Brown and Company of Cincinnati. The cleaned spar contains calcium fluoride from 83 1/2 per cent. to 93 2/10 per cent., generally 85 per cent.; calcium carbonate from 1/4 to 10 per cent., generally about 6 per cent., and silica from 1/2 to 8 per cent. In the mining large bodies of calcite are encountered and overhand stoping is used. The veins range from a few inches to 22 ft. in thickness. A new steel mill built at the Rosiclare mine will handle 500 tons of crude rock per day. A description of the treatment is given in an article by E. F. Burchard[1] from which the following notes are taken. The mill consists of three buildings, namely the sorting and sizing building, the power-house and grinding department and the jig-house. From the mine the spar is hoisted to the top of the mill and dumped directly from the cars on steep grizzlies having 2 1/2-in. spaces. From the oversize, No. 1 lump is picked and thrown on conveyors. The material passing the grizzlies and the crushers feeds into the two 150-ton steel bins and in turn two shaking screens. These screens have steel frames 17 ft. by 2 ft. 9 in. and consist of heavy wire, the opening being 1/2 in. by 1 1/8 in. The picked lump spar, No. 2, is dried in a rotary kiln, 3 ft. in diameter and 25 ft. in length supplied with hot air from the top of the boilers. The product is either shipped directly or recrushed. The No. 1 spar passes through a crusher and is then ground in a mill which discharges through a 30-mesh screen into storage-bins. These storage-bins feed directly into barrels. The barrels of ground spar weigh from 550 to 610 lb. The capacity of the packing room is about 30 tons per day. From the shaking screens the undersize is carried in a 9-in. pipe to the jig-house and the undersize falls on a picking belt where a separation is made by 9 to 15 men who pick out lead, zinc, calcite, waste and fluorspar. The lead and zinc ores pass to separate 13-in. disk crushers and the product is carried from each to the jig-house in pipes. The over run of fluorspar from the picking belt passes to a 24-in. disk crusher from which the mixture is fed into a bin supplying the same pipe that takes the undersize from the shaking screens. The treatment of this material is accomplished by five-compartment roughers and then by six-compartment cleaners. Between the jigs are crushers and pumps. The lead and zinc ores from the 13-in. disk crushers are treated by two four-compartment pulsator jigs. Ninety per cent. of the material treated is in the form of concentrates of fluorspar, galena and blende, the galena being about 1 per cent. and the blende less. A high efficiency must be attained in this process even though a small percentage of tailings results. The lead and zinc ores are by-products but cover part of the expense. The shipping product is carried to the Ohio River where it is loaded on barges for Shawneetown, Ill., Evansville,

[1] Mineral Resources, U. S. Geol. Surv., 1910; and *Iron Trade Review*, December 14, 1911.

Ind. and Golconda, Ill. In case of interrupted traffic, large stocks are kept at Shawneetown.

The Rosiclare or Fairview spar[1] contains no barite or gypsum. Carbonates of lead, zinc, and iron are frequently present, as is also calcium carbonate. The method of analysis is to dissolve in acetic acid, leaving calcium fluoride and silica to be estimated. Calcium fluoride is somewhat soluble but allowance is made for this. This fluorspar is free from boron and contains very little aluminum. For this latter reason, it cannot contain any laumontite or stilbite. As there are no alkalies, it cannot contain any mineral like apophylite or natrolite, these being mentioned for Kentucky spar by F. J. Fohs.[2]

New Mexico.—The fluorspar obtained 10 miles northeast of Deming, Luna County, N. M. was opened in 1909 in a number of veins by the American Firemen's Mining Company of Kansas City and promises to supply the western markets. In 1910, G. M. Sadtler took over the mines. The product is shipped from a siding on the Atchison, Topeka & Santa Fe Railway at Mirage, 5 1/2 miles southeast of the mines. Fluorite Ridge consists of a central mass of monzonite intruded in strata of limestone and sandstone, these being often folded into the igneous rock. The base of the ridge is surrounded by an agglomerate of angular masses of andesite imbedded in tuff. The veins of fluorspar occur in the decomposed monzonite. The veins consist of some fluorite and some quartz and are generally 2 to 5 ft. thick. The spar is light green but sometimes purple, especially near the margin of the vein. None of the veins extends into the limestone. At the prospects 1000 ft. west of the principal shafts, the main vein appears to lie on a fault plane. The footwall here is a cherty conglomerate and the hanging wall has not been determined. In mining a layer of clay, the hanging is protected by a shell of spar to prevent caving.

The fluorspar of the West is said to be less pure than that from the East although analyses by the principal consumer, namely the Colorado Fuel and Iron Company, do not show this. The material is not treated mechanically and probably there is a great deal of low-grade waste from cobbing, which cannot be marketed. The cost of mining near Deming is probably not greater than $2 per ton as it is done under contract with Mexican labor. This would make the total cost, after a 5 1/2-mile haul to the railroad, $3.50 f. o. b. cars. In 1910, 90 per cent. calcium fluoride was selling at $5.25 f. o. b. cars. A premium is paid at 20 cents per unit per ton when the fluoride exceeds 95 per cent. and the same penalty is deducted when the assay is less than 90 per cent.

[1] E. Bidtel, Laboratory of Fairview Fluorspar and Lead Company. *Journal Ind. and Eng. Chemistry*, March, 1912.
[2] *Bul.* IX, Kentucky Geological Survey.

FULLER'S EARTH

Most of the production of fuller's earth in the United States comes from Florida but the quality of the material is not such that it can entirely supply the domestic uses. During 1911 the production was 34,668 short tons valued at $333,350; in 1910, it was 30,857 short tons valued at $277,293 and in 1909, it was 29,561 short tons. Fuller's earth is produced also in Texas and Arkansas but on account of the location, the price of the product delivered is too high to compete with the Florida production in most cases. The principal producers, in the order named, are the Floridin Company, Warren, Pa., with mines and mills in Florida, Atlantic Refining Company, at Philadelphia, Pa., the Lester Clay Company, Jacksonville, Fla., The Fuller's Earth Company, Somerville, Texas, the Arkansas Fuller's Earth Company, Chicago, Ill., and the Little Rock Refineries Clay Co. at Little Rock, Ark.

Markets.—The article being indispensible to oil refining industries, it is firmly established in the markets. The price has not fluctuated very much and the supply is usually mined and shipped during the dry seasons as most of the appliances are on a small scale at the mines. The price in small quantities has been constant for some time at 80 to 85 cents per 100 lb. The average price per ton given by the producers for the year 1910 was about $9 and for the year 1911, the price averaged $9.50 per ton at the mines.

The Production of Fuller's Earth in Florida During 1911

By E. H. Sellards, State Geologist

The Floridin Fuller's Earth Company continued the operation of two plants in Florida during 1911. One of these is located at Quincy and was formerly under the ownership of the Owl Commercial Company. The second plant is located at Jamieson, also in Gadsden County. A new plant was built and began operations at Midway in Gadsden County by the Fuller's Earth Company. In Manatee County the Atlantic Refining Company continued the operation of its plant at Ellenton. The Florida Fuller's Earth Company operated a plant at Ellenton during a part of the year. The total production of fuller's earth in Florida during 1911 approximated 25,000 short tons. A detailed description of a number of fuller's earth deposits was given in Vol. XVIII of The

MINERAL INDUSTRY. Methods of mining in Florida are given in the Second Annual Report of the Florida State Geological Survey. The over-burden is from 2 to 4 ft. and recently steam shovels are used for its removal. Blasting is sometimes necessary. A feature of the deposits is the number of masses of impure limestone in such places where the overburden is thick. At Quincy the pit was abandoned when the overburden exceeded 12 to 14 ft. The material was sorted, crushed and passed through a drying cylinder which was 30 to 40 ft. long and 6 ft. in diameter and having inclined blades to convey the material. Each cylinder dried 1 ton per hour thus making its time in the kiln 15 minutes. This product is stored in bins and ground as needed. There are two grades of fineness, the usual size being 30 to 60 mesh.

Fuller's Earth in Texas.—The expansion of the petroleum and cotton-seed-oil industry in Texas and the demand for high-grade fuller's earth elsewhere has given a new impetus to the development of this material in the State. Fuller's earth, of excellent quality, has been found in Ellis, Burleson, Shelby, Smith, Fayette and Washington Counties. The best known deposits are in Burleson County, near Somerville; in Washington County, near Burton and in Fayette County, near O'Quinn and Muldoon. So far as known, at present, the largest deposits are in Fayette County and more is being done there for their development than elsewhere in the State. In 1910, Texas produced 864 tons, valued at $9.93 per ton, according to William B. Phillips, of the Texas Bureau of Economic Geology and Technology.

In 1911 the production was about 800 tons, as some of the companies had not gotten to the producing stage. The Somerville Fuller's Earth Co.'s plant in Burleson County, is 1 mile west of the Gulf, Colorado & Santa Fe Railway. The deposit covers 50 acres with an overburden of 6 to 10 ft., in some places this overburden being sandstone. The bed of brown fuller's earth is 12 ft. thick and is of fine texture, compact and hard. It contains iron and occasionally nodules of iron pyrites. The Texas Fuller's Earth Company in Washington County, is 7 miles north of Burton. The deposit consists of soft brown fuller's earth and covers 250 acres. There are two strata separated by a bed of lignite shale about 6 in. thick. The upper stratum is 4 ft. thick and the lower 5 ft. thick. The overburden is about 6 ft. and the upper seam is the one that is being worked.

Other properties in Texas as described in a contribution taken from the United States Geological Survey for 1910, are Melcher's, 5 miles south of West Point in Fayette County; J. C. Murray's, 3 miles north of Lyons in Burleson County and Paul Taylor's, 1 1/2 miles northwest of Somerville in Burleson County. The latter two deposits are shale

which has been classed as fuller's earth. Other deposits of the same nature also exist on Red Gully, 4 miles north of Burton in Washington County and 2 miles east of Ledbetter on the Burkhart land on Turkey Creek, Washington County.

Fuller's Earth in Foreign Countries

Fuller's earth was produced in the United Kingdom in Bedford and Surrey where the quarries are in Lower Greensand, and also from Somerset where they are in beds of Jurrasic age. The amount produced in 1910 was 28,293 short tons valued at £33,123. The imports of English earth into the United States are increasing in spite of the duty, but the domestic product is still used chiefly for petroleum. The other oils are mostly clarified with the imported article, although one large concern uses domestic earth to some extent.

Fuller's earth, according to the United States Consul at Berlin, Germany, is extensively imported there from the United States. It is duty free and its uses in Germany are to dissolve fats and oils, to cleanse cloth, in the manufacture of paper and rugs and in the production of ultramarine, and also for an article to clean out spots.

In the United States domestic earth could be produced at a profit at a lower figure than the present cost for foreign earth. Large foreign imports of this, however, are still necessary to supply the exacting conditions of some of the American oil companies. It requires considerable skill to prepare the earth for market, although some earths are entirely unsuitable for some particular oils. It also requires the right kind of handling to prevent any earthy taste in the edible oils. In 1910 there were imported into the United States, 1578 tons of manufactured fuller's earth valued at $12,253 and also 11,614 tons of the unmanufactured product valued at $106,706. In 1911 the imports of the first class were 2049 tons and of the second class 13,764 tons, valued at $14,121 and $125,899 respectively. These figures refer to the fiscal years ending June 30th. The duty is $1.50 and $3 per ton.

Properties and Uses.

The peculiarity of fuller's earth for clarifying oils by absorbing coloring matter does not apparently depend on the chemical composition. Fuller's earth is not a plastic substance and the porosity must be of the right degree, as many substances, like diatomite, are porous but cannot absorb coloring matter. Fuller's earth is a clay that contains an excessive amount of hydrated silica, the dry substance adhering to the tongue.

Actual tests alone will tell its other properties. The test applied for petroleum or lubricating oils is not as exacting as for vegetable oils. A sample of oil is heated to redness, cooled and placed in a five-inch funnel. Oil is then dropped on gradually and the amount that comes through clarified to the required degree, is estimated by comparing with a sample of the desired color. For vegetable oils, a sample is generally treated by mixing and stirring after grinding and drying, with a large bulk of oil that has been previously heated. The temperature is as high as the oil will stand. If the clarifying effect is satisfactory, there is no way of testing the regenerative qualities except on a large scale in practice. It was reported that fuller's earth, used successfully in California, is not porous and the same is reported of some of the Texas fuller's earth deposits. In Florida also, a brown colored earth that is not porous, has been developed and it decolorizes oil very well. On account of the high price of fuller's earth, the large consumers are particular as to its qualities, according to which the market is adjusted and probably no two firms have the same requirements. The decolorizing properties of the oil are held by some to be due to the chemical, and by others, to the physical nature of the substance.[1] After fuller's earth is used as a decolorizing material for oils, according to French patent 409,915, March 1, 1909, the material mixed with salt water and heated to 85° C. with sulphuric acid, then treated with sodium carbonate and mechanically agitating liberates carbon dioxide, carrying to the surface the oil particles. This regenerates the material for new use. A steam coil and mechanical agitation are used in this process.

STATISTICS OF FULLER'S EARTH IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Years.	Production.		Imports.	
	Sh. Tons.	Value.	Sh. Tons.	Value.		Sh. Tons.	Value.	Sh. Tons.	Value.
1901....	14,112	$ 96,835	12,061	$ 80,697	1907...	34,039	323,275	14,648	$122,221
1902 ...	14,100	109,980	15,135	102,580	1908...	30,517	270,685	12,279	92,413
1903....	20,693	190,277	17,100	120,670	1909...	29,561	289,000	12,752	101,151
1904....	29,480	168,500	10,221	74,000	1910...	30,857	277,293	16,857	132,545
1905....	25,745	157,776	15,181	105,997	1911...	34,668	335,350	18,224	143,594
1906....	28,000	237,950	14,827	108,696					

Fuller's earth or other mineral substances that have been used in deodorizing and decolorizing oils or fats may be rendered again commercially available by saponifying the absorbed fat, so as to yield a soapy compound. If desired, the water may be evaporated and the product obtained in the form of a dry powdered detergent. (N. Sulzberger, New York. U. S. Pat. 968,092, Aug. 23, 1910.)

[1] Abstract of paper presented before the American Institute of Chemical Engineers by David Wesson, Dec. 8, 1910; J. T. Porter's paper "Preparation and Tests of Fuller's Earth."

GARNET

The industry, located chiefly in the Adirondacks, like other abrasive industries, was on the decline in 1911. In addition to New York State, there is a small production from North Carolina and New Hampshire. Competition from other abrasives is increasing, especially as the manufactured abrasives are found more and more suitable. The quality of these materials also can be controlled to a certain extent, whereas the natural materials vary considerably.

As long as garnet can be cheaply produced, however, it will hold a place in the abrasive field. Crystal garnet particularly has some properties for abrasive paper use that make the industry permanent, though small. A small amount of massive garnet, generally of paler color than the crystalline, is used for the same purpose, but is not so suitable as the crystalline on account of the rough fracture. The cheapness of manufacture of such materials as alundum and carborundum and the price of their raw materials can hardly proceed to a point where these materials will altogether replace garnet. Some of the natural abrasives are, undoubtedly, better adapted for some particular purposes than are the manufactured products; this applies more to the harder abrasives such as corundum, than it does to garnet.

The accompanying table shows the production of garnet in the United States for a number of years.

PRODUCTION OF GARNET IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Value per Ton.	Year.	Short Tons.	Value.	Value per Ton.
1902	3,772	$122,826	$33.00	1907	6,723	$209,895	$31.22
1903	4,413	146,955	33.30	1908	2,530	78,090	30.86
1904	2,952	89,636	30.36	1909	3,802	121,700	32.01
1905	3,694	114,625	31.01	1910	5,297	151,700	28.64
1906	5,404	179,548	33.22	1911	4,285	121,759	28.44

New York (By D. H. Newland).—The Adirondack garnet mines reported an output last year of 4285 short tons with a valuation of $121,759. This was less than in 1910 when the total amounted to 5297 short tons valued at $151,700, but may be considered as about the average out-turn. The production has varied from year to year according to the activity of the market but at no time has taxed the full capacity of the mines. In fact the demand for abrasive garnet has shown very little

tendency to increase and there would seem to be little opportunity at present for the development of new sources of supply.

No important changes in the industry have taken place during the past year. The principal producers, as heretofore, were the mines in the vicinity of North River. The largest factor in the industry has been for some time the North River Garnet Co., with mines and milling plant on Thirteenth Lake, Warren County. The other active mines in that section included those on Gore mountain owned by the H. H. Barton & Son Co., and those of the American Glue Co., a little farther north in Essex County. At Riparius, the Warren County Garnet Mills have operated in a small way. In northern Essex County near Keeseville the American mine shipped some material.

The conditions surrounding the occurrence of garnet in the Adirondacks have been described in several papers and in previous issues of this report. The mineral is fairly widespread as a constituent of the metamorphosed igneous and sedimentary rocks, but only in a few places is it found in sufficient abundance and with the requisite characters to be worked for commercial abrasive purposes.

The value of abrasive garnet depends, of course, primarily upon its hardness. This is a variable character and on the usual mineral scale garnet is classed as having a hardness of from 6.5 to 7.5. The limits as given are only approximate, as it is difficult and even impossible to estimate hardness with precision. Chemical composition is undoubtedly a factor in determining the hardness of the common kinds of garnet found in the metamorphosed rocks, like gneisses, schists and crystalline limestones. The iron-alumina variety (almandite) is generally harder than the lime-alumina (grossularite), or the lime-iron variety (andradite). Well-crystallized garnet is tougher and probably also harder than the granular or massive garnet of similar chemical composition. The property of toughness or tenacity is very important in an abrasive which has to withstand considerable pressure, as when used in polishing machines. Another factor which has a bearing upon the value of abrasive garnet is the size of the product which can be secured in the ordinary practice of mining and separation. If the crystals are small or have been badly shattered by compression after crystallization, the product may be too fine to yield the necessary assortment of commercial sizes. It is an advantage, however, that the garnet should possess an imperfect cleavage or parting, so that on crushing the grains show one or more smooth surfaces. These surfaces permit firm attachment to the cloth or paper and also provide a sharp cutting edge. Color is no criterion of quality in ordinary garnet, but abrasive users seem to prefer the darker shades of red which approach the distinctive garnet color.

The local industry has very little competition from other mines in this country. Mines have been worked at different times in New Hampshire, Massachusetts, Pennsylvania and North Carolina but have not proved permanent factors in the trade. The importation of Spanish garnet, first noted in 1907, has assumed some importance as a substitute for the finer sizes of the Adirondack mineral. This garnet is said to be obtained by concentration of alluvial sands and can be produced cheaper than the domestic garnet, but comes only in the finer sizes. It pays no import duty. The importations in 1911 were 693 short tons, with an invoice value of $10,526. In 1900, they amounted to 775 short tons valued at $14,830. The principal ports of entry are New York, Boston and Chicago, and the shipments are made from both Spanish and British ports.

The production in Spain in 1910, the last figure available, was 200 metric tons for the calendar year valued at 4000 pesetas at the mines.

Georgia (By Louis M. Richard).[1]—Near Dahlonega, Lumpkin County, Georgia, about a mile south of Porter Springs, occur garnet deposits of commercial importance. The garnet crystal of the garnetiferous schists does not attain a large size, and when found is nearly always covered with a coating, due to weathering, rendering it valueless. Several analyses on this type of garnet show them to be of the almandite variety and of no commercial importance.

Owing to extreme weathering, the hills for the most part, are covered with a residual mantle of decomposed rock mass called by Dr. Becker saprolite.[2] In this saprolite, the garnets are found ranging in weight from 2 oz. to several pounds and of dodecahedral crystallization.

Several pits not over 15 ft. deep have been dug on promising exposures. The garnets are much fractured and by breaking them small, clear specimens are obtained. These are very hard, show sharp fractures and are of blood-red color, making excellent gems when cut. Occasionally specimens are obtained showing a purplish hue. Selected fragments yielded gems when cut weighing from 1 to 4 carats. Analyses show it to grade toward a magnesia-iron-lime-alumina garnet, and it may be termed a pyrope. Workable deposits for abrasives may possibly be discovered in the district.

Labrador.—At St. Michael's Bay, 35 miles north of Belle Isle, there exists an island about a mile long and a half mile wide, on which occurs garnet composed of crystals exposed in a ledge 330 ft. long and 11 ft. wide. There is also associated some quartz and mica. This garnet has

[1] *Mining World*, June 3, 1911.
[2] Becker, G. F., Gold Fields of the Southern Appalachians; Sixteenth An. Rept. U. S. Geol. Surv., Part 3, 1895.

been tested for abrasive work and found to be suitable. There are good shipping facilities.

New Hampshire.—Garnet milling is described by Earl S. Bardwell.[1] The rock, a granite, outcrops over an acre, near the top of Currier Hill in North Wilmot, Merrimac County, and contains 40 to 50 per cent. of garnet crystals. These are rounded, about 1/4 in. in diameter, and contain quartz and biotite. The rock is quarried and crushed to 1 1/4 in. and then lowered to the mill by a two-bucket rope tram. Six tons per hour are crushed to 3/8 in. in Sturtevant balanced rolls 26×15 in., 821 r. p. m. The ore is then dried in a cylinder (American Process Company) 21 ft. long making 17 r. p. m. This enters a dust chamber of 10 in. concrete walls and 12×12×12 ft. outside, and supporting a 50 ft. × 24 in. stack. Practically no garnet is found in the dust which is wasted.

The treatment is then as follows: Through a No. 3 Newaygo screen (90 r. p. m., 14-mesh No. 14 wire at 45 degree inclination) and also through one Sturtevant 26×15 rolls (100 r. p. m.). The material goes to another No. 3 screen, followed by a 6-mesh, 16-wire screen. This yields a size from 2 mm. to 1 4/10 mm. that is treated on a Sutton, Steele & Sutton dry table (390 r. p. m.) which makes tailings that are wasted a final product which is sacked, and middlings which pass to the following screen. This is a 10-mesh, 20-wire screen producing two sizes, the larger one being 1.4 mm. to 1.5 mm. and the other smaller than 3/4 mm. The smaller size is again screened on a 24-mesh, 29-wire screen producing one size which is less than a .36 mm. and is wasted and another size from .36 to .75 mm. This latter and also the 1.4 to .75 mm. size are treated alternately on a Sutton, Steele & Sutton table making 425 r. p. m.

The garnet is shipped in canvas sacks of 100 lb. each. The plant produced at a rate of 12 tons in 10 hours. The shipping point is at South Danbury, 3 miles from the mill.

[1] *E. and M. J.*, June 17, 1911, p. 1209.

GLASS SAND

It is not possible to give details of the glass industry in this volume except as to the chief raw material, glass sand. Other sands as far as total values and tonnage go, can be summarized by giving the production figures collected by the United States Geological Survey as follows:

In 1909 building sand, 23,116,199 short tons, valued at $7,238,814; fire sand, 151,239 tons, valued at $126,178; engine sand, 724,409 tons, valued at $266,150; furnace sand, 431,203 tons, valued at $227,581; other sands, 7,632,340 tons, worth $1,448,186; gravel used for concrete paving and ballast, 23,282,904 tons, valued at $5,719,886. The total amount of sands of all grades and classes for the year 1909 amounted to 59,565,551 short tons, valued at $18,336,990.

In 1910 building sand, 25,743,072 short tons, valued at $7,832,638; fire sand, 175,081 tons, valued at $136,361; engine sand, 878, 411 tons, valued at $349,145; furnace sand, 493,566 tons, valued at $279,046; other sands, 4,438,428, valued at $1,240,365; gravel, 32,584,622, valued at $7,252,110. This makes the total for 1910 of all sands and gravel as reported 69,410,436, worth $21,037,630. Of the classes of most value per ton are glass sand and molding sand, the average price of the former being $1.05 per ton in 1909, and $1.04 per ton in 1910. That of molding sand was 70 cents in 1909 and 1910. Fire sand had an average value of 77 cents per ton of 2000 lb. The table on page 281 gives the production by state.

The total production of glass sand in the United States in 1902 was 943,135 tons, valued at $807,797.

Sand valued at $114,844 was imported into the United States in 1910, as compared with imports valued at $106,234 in 1909 and at $77,574 in 1908. In 1911 the value of sand imported was $139,784.

Glass; plate-glass and other glass.—There were 5,313,564 sq. ft. of polished plate glass imported into the United States in 1910, valued at $1,184,622. In 1911 the figures were 2,272,851 sq. ft. and $526,859. For window glass in 1910, 31,814,389 lb. and in 1911 26,520,858, pound were imported, valued at $943,762 and $902,916 respectively. There is a duty on these kinds of glass. The average import price for window glass in 1911 was 3.43 cents per pound. Other glass imported in 1911 was only about two-thirds the value of the same class imported in 1910, when the largest imports took place.

PRODUCTION OF GLASS SAND AND MOLDING SAND. (a)
(Short tons.)

State.	1909				1910			
	Glass Sand.		Molding Sand.		Glass Sand.		Molding Sand.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Alabama			119,999	$57,236			56,085	$22,069
Arkansas	(b)4,487	$2,575	(c)				600	515
California	(d)		(e)28,002	32,068	(b)	(b)	c 10,133	c 9,342
Colorado			6,544	3,840			3,857	2,627
Connecticut			1,069	1,117			295	779
Delaware			50	50			1,000	600
Florida			559	717			795	785
Georgia	(d)		(e)18,776	10,067	(d)	(d)	e 42,355	c 19,341
Illinois	224,381	153,226	88,518	143,922	268,654	$216,531	407,232	215,742
Indiana	(d)		(e)142,864	93,021	(d)	(d)	e 165,357	e 85,204
Iowa			26,463	35,340			13,320	9,775
Kansas								
Kentucky	(d)		(e)42,298	33,587	(d)	(d)	e 57,318	e 43,788
Louisiana								
Maryland	(b)14,483	15,730	(c)		b 25,392	b 22,996	(c)	(c)
Massachusetts	(d)		(e)24,190	29,572	(d)	(d)	e 13,166	e 23,630
Michigan	65,000	79,000	53,226	20,756	(d)	(d)	e 110,024	e 49,679
Minnesota			33,657	36,374			23,099	26,422
Mississippi			226	213			256	166
Missouri	98,480	73,082	70,136	38,578	186,342	130,686	79,977	42,787
New Jersey	85,696	62,830	499,291	310,910	87,680	61,078	679,191	436,460
New York	(d)		(e)477,121	423,172	(d)	(d)	e 536,232	e 453,362
North Carolina			409	282			439	396
North Dakota								
Ohio	88,054	95,331	650,336	481,125	139,122	104,295	724,313	533,343
Oklahoma							90	24
Oregon								
Pennsylvania	281,120	399,707	540,007	350,975	434,147	617,200	605,340	413,741
South Carolina	(b)850	739	(c)		b 2,144	b 1,993	(c)	(c)
South Dakota								
Tennessee	1,550	1,250	47,006	24,691	1,320	1,344	48,129	24,439
Texas			4,774	3,178			4,186	2,649
Vermont			1,820	1,207			2,268	1,736
Virginia			25,480	17,241	(d)	(d)	e 36,217	e 25,404
Washington			3,935	6,200			4,070	7,577
West Virginia	169,731	205,102	2,248	1,669	235,784	282,267	3,541	2,461
Wisconsin	(d)		(e)84,381	63,915	(d)	(d)	e 87,786	e 54,732
Total	1,033,832	1,088,572	3,193,425	2,221,023	g1,380,585	g1,438,390	3,716 671	2,509,575

(a) From U. S. Geol. Surv. (b) Includes molding sand. (c) Included in glass sand. (d) Included in molding sand. (e) Includes glass sand. (g) Complete output 1,461,089 tons, $1,516,711.

In the United States window and plate glass are in demand, the consumption being equal to the production, although the markets were weakened by the great capacity for production not utilized. Importations, as stated above were materially reduced in 1911. The manufacture of window glass in Europe has been recently described.[1] The value of all the glass products made in the U. S. probably exceeds $70,000,000 per year. Compared with this the clay products are at least 2 1/2 times as great.

In the United States glass manufacture for optical purposes is very limited. In fact, most of the glass used for such purposes is imported from France, Germany or England. This glass is generally a crown glass

[1] *Genie Civil*, Nov. 25, 1911.

and requires a constant or definite index of refraction. Much research work is being done in avoiding the striae and other imperfections, but thus far a good glass has not been manufactured in this country. The glass industry therefore confines itself chiefly to window and plate glass. The optical glass may not be a crown glass at all times as an occasional index of 165 may be required, in which case it might be a mixture between the two. For camera work both crown and flint glass are required. The flint glass index is from 1.6 to 2. This glass is manufactured in this country and probably its principal use is in cut glass. One of the few manufacturers of crown glass for optical purposes in this country is the Kanas Universal Lens Co. This firm manufactures bifocal lenses made of crown glass.

In Germany glassware for scientific purposes is manufactured on a very large scale. Only a small proportion goes to the United States.

The volume of Germany's business of this character with all countries is large in the aggregate, and it seems probable that most of the apparatus is manufactured at Jena and Beuel-Bonn. The total value of exportations of "glass apparatus and instruments for industrial or scientific purposes" was $1,939,462 in 1911 and $1,627,444 in 1910, the shipments to the United States amounting to $362,959 and $339,579, respectively.

By weight these exports amounted to 1737.7 and 1434.4 tons, respectively, and were distributed in 1911 as follows, in metric tons of 2204.6 pounds: United States, 325.2; Great Britain, 353.4; Russia, 155; Italy, 134.7; Switzerland, 128.1; Austria-Hungary, 120.3; France, 99.3; Belgium, 60.3; Netherlands, 44; Argentina, 40.4; Brazil, 27.1; Sweden, 24.7; Japan, 14.1; all other countries, 211.1.

There is advertised in Germany for glass manufacture ground 'phonolith' which has the following composition. It is furnished of any fineness.

Silica	50 per cent.
Alumina	24 per cent.
Potassa	9 per cent.
Soda	8 per cent.
Ferrous oxide	2 per cent.
Lime	1½ per cent.

L. O. Armstrong, of the industrial department of the Canadian Pacific railway, states that extensive deposits of white silica sand suitable for the manufacture of high-grade glass, have been discovered on the Canadian shore of Lake Huron, 28 miles east of Sault Ste. Marie. Samples of the sand, containing from 96.82 to 99.75 per cent. silica, have been shipped to glass manufacturing firms in the United States and Europe.

GOLD AND SILVER

The gold production of the world in 1911 again exceeded that of any previous year, though the increase over 1910 was only two per cent. This was due to the increase of over $15,750,000 in the Transvaal. Fair gains were also made in Rhodesia, Canada and West Africa. The United States production was stationary. The important decreases were in Australia, the causes of which are explained elsewhere; in Russia, where unusual drought interfered with operations in the Siberian placers during the first half of the year; in Mexico, where revolutionary disturbances affected mining and mine labor; and in India, where two of the important Kolar mines are still in a zone of lean ore. These decreases, however, were not sufficient to overcome the large gain in the Transvaal and the other increases.

GOLD PRODUCTION OF THE WORLD. (a)

Countries.	1909.			1910.			1911.		
	Oz. Fine.	Kilograms	Value.	Oz. Fine.	Kilograms	Value.	Oz. Fine.	Kilograms	Value.
America, North:									
United States	(b)4,822,129	149,986	$99,673,400	(b)4,657,017	144,853	$96,269,100	(b)4,655,297	144,794	$96,233,528
Canada	(a) 453,865	14,117	9,382,230	(a) 493,751	15,357	10,205,835	472,280	14,690	9,762,096
Newfoundland	(c) 2,438	76	50,330	(c) *Nil.*	.....	.....	*Nil.*	.....	.....
Mexico	(a)1,089,113	33,875	22,511,966	(b)1,093,053	33,998	24,484,372	(g)1,412,483	43,933	29,196,026
Central America	(b) 127,242	3,958	2,630,100	(b) 131,287	4,083	2,713,700	165,457	5,146	3,420,000
America, South:									
Argentina	(c) 10,352	322	213,982	(b) 8,372	260	173,061	(c) 8,841	275	182,750
Brazil	(g) 108,988	3,390	2,252,900	(g) 94,566	2,972	1,954,674	(g) 137,927	4,290	2,850,940
Chile (f)	40,766	1,268	842,649	(b) 22,429	698	463,600	(c) 45,235	1,407	935,000
Colombia	150,000	4,660	3,100,500	163,038	5,071	3,370,000	(c) 165,699	5,154	3,425,000
Ecuador	17,000	529	351,390	(g) 12,054	375	249,179	(c) 15,820	492	327,000
Guiana, British	(g) 64,977	2,021	1,343,075	(g) 57,402	1,773	1,178,645	(a) 52,453	1,632	1,084,180
Guiana, Dutch	39,000	1,213	806,130	(b) 38,744	1,193	792,600	(a) 36,500	1,140	754,450
Guiana, French	144,675	4,500	2,990,700	149,975	4,822	3,100,000	(c) 142,719	4,589	2,950,000
Peru	(c) 23,920	774	514,354	22,762	708	470,505	(c) 31,205	971	645,000
Uruguay	4,019	125	83,075	(b) 4,433	138	91,642	(c) 4,433	138	91,642
Venezuela	1,200	37	24,804	(g) 16,472	512	340,512	(a) 16,238	522	335,650
Europe:									
Austria	(a) 4,755	148	98,295	5,739	178	118,895	(c) 4,840	150	100,000
France	(a) 55,510	1,727	1,147,400	(a) 70,392	2,190	1,455,027	(c) 76,840	2,390	1,588,287
Hungary	106,095	3,300	2,193,180	(a) 97,824	3,043	2,022,032	(c) 100,387	3,122	2,075,000
Germany	(a) 162,807	5,064	3,365,281	(a) 148,697	4,625	3,073,568	159,692	4,967	3,300,845
Italy	(b) 317	10	6,562	(a) 485	15	10,047	(c) 484	15	10,000
Russia	(a)1,542,330	47,987	31,889,956	(c)1,716,722	53,396	35,566,045	1,202,952	37,329	24,865,018
Portugal	1,500	47	31,005	(a) 129	4	2,658	(a) 1,441	44	29,850
Spain	(b) 158	5	3,781	*Nil.*	.....	.....	(c) 142	4	2,950
Sweden	(a) 453	14.1	9,370	96	3	1,990	(a) 787	24	16,250
Turkey	(b) 106	3.3	2,200	(b) 290	9	6,000	1,447	45	29,905
United Kingdom	(a) 1,209	37.6	24,990	(a) 2,427	76	50,166	(a) 273	9	5,650

(a) Official statistics of the country. (b) United States Mint Report. (c) Six States and New-Zealand. (d) Exclusive of Formosa. (e) Estimated. (f) Includes Bolivia. (g) Exports.

and requires a constant or definite index of refraction. Much research work is being done in avoiding the striae and other imperfections, but thus far a good glass has not been manufactured in this country. The glass industry therefore confines itself chiefly to window and plate glass. The optical glass may not be a crown glass at all times as an occasional index of 165 may be required, in which case it might be a mixture between the two. For camera work both crown and flint glass are required. The flint glass index is from 1.6 to 2. This glass is manufactured in this country and probably its principal use is in cut glass. One of the few manufacturers of crown glass for optical purposes in this country is the Kanas Universal Lens Co. This firm manufactures bifocal lenses made of crown glass.

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Canada	(a) 453,865	14,117	9,382,230	(a) 493,751	15,357	10,[illegible]
Newfoundland	(e) 2,438	76	50,330	(e) *Nil.*		
Mexico	(a)1,089,113	33,875	22,511,966	(b)1,093,053	33,998	24,[illegible]
Central America	(b) 127,242	3,958	2,630,100	(b) 131,287	4,083	[illegible]
America, South:						
Argentina	(e) 10,352	322	213,982	(b) 8,372	260	
Brazil	(g) 108,988	3,390	2,252,900	(g) 94,566	2,972	
Chile (f)	40,766	1,268	842,649	(b) 22,429	698	
Colombia	150,000	4,660	3,100,500	163,038	5,071	
Ecuador	17,000	529	351,390	(g) 12,054	375	
Guiana, British	(g) 64,977	2,021	1,343,075	(g) 57,002	1,773	
Guiana, Dutch	39,000	1,213	806,130	(b) 38,744	1,205	
Guiana, French	144,675	4,500	2,990,700	149,975	4,665	
Peru	(e) 23,920	774	514,354	22,762	708	
Uruguay	4,019	125	83,075	(b) 4,433	138	
Venezuela	1,200	37	24,804	(g) 16,472	427	
Europe:						
Austria	(a) 4,755	148	98,295	5,739	178	
France	(a) 55,510	1,727	1,147,400	(a) 70,392	2,189	
Hungary	106,095	3,300	2,193,180	(a) 97,823	3,042	
Germany	(a) 162,807	5,064	3,365,281	(a) 148,667	4,625	
Italy	(b) 317	10	6,562	(a) 457	14	
Russia	(a)1,542,330	47,987	31,889,956	(e)1,718,[illegible]	[illegible]	
Portugal	1,500	47	31,005	(a) 19		
Spain	(b) 188		3,[illegible]			
Sweden	(a)					
Turkey	(b)					
United Kingdom	(a) 1					

(a) Official s[illegible] (d) Ex[illegible]

[illegible] .se-
[illegible] ord-
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[illegible] er proper.

PRODUCTION OF GOLD IN THE UNITED STATES. (a)

States.	1908.		1909.		1910.		1911.	
	Fine Ounces.	Value. (b)	Fine Ounces.	Value. (b)	Fine Ounces.	Value. (b)	Fine Ounces.	V
Alabama	1,993	$41,200	1,413	$29,200	1,593	$32,900	887	
Alaska	959,755	19,858,800	984,015	20,339,600	787,148	16,271,800	774,144	16
Arizona	120,948	2,500,000	127,083	2,626,800	165,113	3,413,200	142,938	2
California	935,157	19,329,700	1,001,625	20,703,600	988,854	20,441,400	982,544	20
Colorado	1,106,483	22,871,000	1,056,923	21,846,600	992,967	20,526,500	926,568	19
Georgia	2,719	56,200	2,100	43,400	1,161	24,000	1,477	
Idaho	69,835	1,443,500	65,031	1,344,200	50,113	1,035,900	56,563	1
Montana	152,879	3,160,000	181,427	3,750,100	179,974	3,720,400	153,341	3
Nevada	565,525	11,689,400	792,753	16,386,200	913,015	18,873,700	917,605	18
New Mexico	14,819	306,300	12,230	252,800	23,084	477,200	30,955	
North Carolina	4,717	97,500	1,519	31,400	3,122	64,500	3,710	
Oregon	43,827	905,900	40,106	829,000	32,960	681,400	28,988	
South Carolina	2,598	53,700	358	7,400	1,827	37,800	650	
South Dakota	374,562	7,742,200	318,026	6,573,600	260,266	5,380,200	359,444	7
Tennessee	179	3,700	212	4,300	136	2,800	684	
Texas	24	500	19	400	18	400	57	
Utah	190,938	3,946,700	203,836	4,213,300	208,627	4,312,700	227,834	4
Virginia	174	3,600	193	4,000	44	900	208	
Washington	12,274	253,700	20,755	711,359	38,992	806,000	24,407	
Wyoming	368	7,600	189	3,900	199	4,100	909	
Other States	179	3,700	309	6,400	283	5,900	14,965	
Total	4,560,953	$94,274,900	4,810,122	$99,425,200	4,649,496	$96,113,700	4,648,878	$96
Porto Rico	29	600	29	600	50	1,000	106	
Philippine Islands	13,764	284,500	11,978	274,600	7,471	154,400	6,313	
Total	4,574,746	$94,560,000	4,822,129	$99,673,400	4,657,017	$96,269,100	4,655,297	$96

(a) The statistics in this table are reported by the Director of the Mint, those for 1911 be preliminary figures (subject to revision). (b) At $20.67 per oz.

PRODUCTION OF SILVER IN THE UNITED STATES. (a)

States.	1908.		1909.		1910.		1911.	
	Fine Ounces.	Commercial Value. (d)	Fine Ounces.	Commercial Value. (d)	Fine Ounces.	Commercial Value. (d)	Fine Ounces.	Co cial
Alabama	400	$211	200	$103	300	$160	174	
Alaska	204,600	108,160	198,600	102,283	153,900	82,336	275,691	
Arizona	2,900,000	1,533,056	2,523,600	1,299,700	2,655,700	1,420,800	1,594,428	
California	1,703,700	900,644	2,304,900	1,187,070	1,791,600	958,506	2,727,336	1
Colorado	10,150,200	5,365,802	8,846,300	4,556,020	8,523,000	4,559,805	7,530,940	4
Georgia	200	106	200	103	300	160	225	
Idaho	7,558,300	3,995,620	6,755,900	3,479,422	7,027,000	3,759,445	7,507,802	4
Illinois	2,000	1,057	900	464	2,000	1,070	4,648	
Michigan	294,100	155,473	217,600	112,068	262,200	140,277	507,234	
Missouri	49,400	26,115	15,200	7,828	32,200	17,227	56,228	
Montana	10,356,200	5,474,702	12,034,500	6,198,004	12,282,900	6,571,351	11,116,778	5
Nevada	9,508,500	5,026,573	10,119,200	5,211,606	12,366,000	6,615,810	10,651,571	5
New Mexico	400,900	211,932	324,200	166,969	779,000	416,765	1,142,335	
North Carolina	1,300	687	400	206	8,300	4,440	2,227	
Oregon	56,100	29,657	69,600	35,845	43,800	23,433	69,116	
South Dakota	197,300	104,301	196,300	101,098	120,600	64,521	206,188	
Tennessee	60,900	32,494	65,300	33,631	69,800	37,343	126,683	
Texas	447,000	236,302	408,100	210,180	364,400	194,954	442,486	
Utah	8,451,300	4,467,695	10,551,100	5,434,024	10,445,900	5,588,556	12,679,633	6
Virginia	300	158	6,400	3,296	200	107	45	
Washington	86,800	45,886	75,200	38,730	204,900	109,621	142,196	
Wyoming	3,500	1,850	1,800	927	1,300	695	1,009	
Other States	6,500	3,436	3,000	1,545	800	428	1,007,761	
Total	52,439,500	$27,721,617	54,718,500	$28,181,122	57,136,100	$30,567,810	57,792,683	$30
Philippine Isla	1,300	687	3,000	1,545	1,800	963	3,383	
Total	52,440,800	$27,722,304	54,721,500	$28,181,681	57,137,900	$30,568,773	57,796,117	$30

(a) The statistics in this table are reported by the Director of the Mint, those for 1911 be preliminary figures (subject to revision). (c) Included in other States. (d) Based on the a value for the year at New York, as follows: 1908, 52.864 cents; 1909, 51.502 cents; 1910, 53.48 1911, 53.304 cents per oz.

Alaska (By A. H. Brooks[1]).—Although most of the gold still comes from placers, much progress was made during 1911 in paving the way for an increased output from auriferous lodes. This work was carried on in most of the gold-bearing areas of Alaska, but the most notable advances were in the Juneau, Valdez, Kenai Peninsula, Willow Creek and Fairbanks districts. Dredge mining also made great progress, in the Nome region. In the entire territory, 22 dredges were operated for the whole or a part of the open season of 1911. In addition to those operated, at least six were in process of construction.

Lode Mining.—About 15 lode mines produced in 1911. Custom mills at Valdez, Chena, Fairbanks, Nome and Skagway were operated and milled ore from many prospects which can not yet be classed as productive mines. Development was done during 1911 on several hundred prospects widely distributed over the territory. The output from auriferous lodes in 1911 was somewhat larger than that of 1910, which was valued at $4,105,459.

In production the Juneau district continues to be the center of the gold lode industry of the territory. In 1911 the Alaska-Juneau Co. began a 6000-ft. tunnel from lower Gold creek, which will form an outlet for the mine to tidewater and permit operations throughout the year. The Treadwell group and the Perseverance mine continued their normal activity. Work was continued at the Eagle River mine, and much systematic prospecting was accomplished in the region lying between the Juneau and Berners Bay districts. In the latter district, both the Kensington and Jualin mines will soon be again on a productive basis. The Chichagof and Golden Gate mines, in the Sitka district, each supplied ore to 10-stamp mills. The success of these two properties has stimulated the development of a number of prospects in the adjacent portions of Chichagof Island.

Gold lode development was active in the Prince William Sound region during 1911. Interest centered in the Port Valdez district, though much was accomplished in the McKinley Lake region near Cordova, and some promising prospects are said to have been found near Port Wells and Gravina Bay. The gold-bearing region of Port Valdez stretches from Columbia glacier on the west to the east side of the Valdez glacier on the east, a distance of about 20 miles. Most of the important development work in the region has been confined to the vicinity adjacent to Shoup Bay, to Mineral creek, and to the west side of Valdez Glacier. The Cliff is the only productive mine yet developed in the district. A depth of 100 ft. below tide-water has been reached in the Cliff mine, with, as reported, no marked changes in grade.

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

Much activity occurred in the development of gold lodes in Kenai Peninsula.

In 1911 a 5-stamp mill was installed on the property of the Kenai Alaska Gold Co. and a 3-stamp mill on the property of the California Alaska Mining Co.

In the Willow Creek district of the Susitna Valley there were two productive properties, the Gold Bullion and the Alaska Gold Quartz. Five 1000-lb. stamps were added to the 2-stamp mill in use at the Gold Bullion mine. A 3-stamp mill with 500-lb. stamps, driven by a Pelton waterwheel, was erected by the Alaska Gold Quartz Mining Co. In southwestern Alaska the Apollo mine, on Unga Island, was operated during the summer.

Much was done on lode gold properties in the Fairbanks district. Many of these shipped ore from development to the custom mills at Chena and Fairbanks, and $50,000 worth of gold was recovered from this source. Late in the summer small stamp mills were installed on the Free Gold, Gold Bullion, and Tolovana properties, in the upper Cleary Creek basin, on the Newsboy, situated near the divide between Eldorado and Cleary creeks.

The report of the Alaska Treadwell Gold Mining Company, for the year ended May 15, 1911, shows that development work to the extent of 9150 ft. was done during the year. The total ore reserves are estimated at 6,637,370 tons, divided as follows: 5,408,960 tons ore in place, and 1,228,410 tons broken ore in stopes.

The record of the 240-stamp mill was: Ore crushed, 395,332 tons; crushed per stamp in 24 hours, 4.71 tons; sulphurets saved, 7339 tons; per cent. sulphurets saved per ton of ore crushed, 1.86. The record of the 300-stamp mill was: Ore crushed, 374,752 tons; crushed per stamp in 24 hours, 5.50 tons; sulphurets saved, 6607 tons; sulphurets saved per ton of ore crushed, 1.76 per cent. The amount of ore crushed in both mills was 770,084 tons, an increase over the tonnage of 1910 of 25,858 tons. The cost of mining the ore was $1.1138 per ton; the cost of milling, $0.2042; sulphuret and general expenses, $0.2107; total operating costs, $1.5287 per ton. The average assay value of the ore was $3.07, which yielded in free gold, including base bars, $1.4049 per ton, and the sulphurets yielded $0.9485 per ton of ore milled, making average returns for the year $2.3534 per ton of ore milled.

Placer Mining.—The value of the output of the gold placers of Alaska in 1911 was about $12,700,000; the corresponding value in 1910 was $11,985,000. This increase in 1911 is to be credited to the Iditarod-Innoko region, for the output from both the Fairbanks and Seward

Peninsula districts was less than in 1910. Placer mining is a relatively unimportant industry in the Pacific coast region of Alaska.

No noteworthy advances were made in the mining of Kenai Peninsula placers. In 1911 a small dredge was installed below Copper creek, but it appears to have given unsatisfactory results. The installation of a large dredge, started during the summer, was not completed at the end of the season. Hydraulic plants were in operation on Crow, Resurrection, Canyon, Gulch, Quartz, Cooper, Mills and Sixmile creeks. Two dredges were successfully operated in the Fortymile region. One on Walkers Fork worked the entire open season; the other, situated on the South Fork, 3 miles below Franklin, began operations about July 1. The latter had been moved from Walkers Fork to its present site during the winter. It is estimated that in the Fortymile district proper about 115 men were working on 35 claims during the winter and 90 men, including those engaged in dead work, on 15 or 20 claims in the summer.

Regarding the season's operations in the Iditarod-Innoko region, the total gold production for 1911 was reported worth $3,500,000, compared with $800,000 for the previous year. Much of the gold was from the Flat, Otter, Willow, Happy and Chicken creek basins, in the region tributary to the town of Iditarod, and in this field some work was also done on Black, Little and other creeks.

The gold production of Seward Peninsula for 1911 was worth $3,100,000; that of 1910 had a value of $3,500,000. In view of the fact that the winter operations of 1910 produced almost $1,000,000 worth of gold, while those of 1911 probably had a value of less than $200,000, the output of the summer was far greater than in the preceding year. It seems probable, therefore, that the peninsula may have reached its minimum gold production for some time to come. As in 1910, the important feature was the building and operating of dredges. Nineteen dredges were operated for a part or the whole of the season, of which five were built during the summer of 1911. Of these, seven were in the Nome region, seven in the Solomon River basin and five in the Council district. In addition to these, five more dredges were in process of construction, of which three are in the Nome region and one each in the Council and Kougarok districts. While dredging was the most important industry, other phases of mining were not neglected. A number of hydraulic plants were operated in the Kougarok, Nome and Fairhaven precincts. An important feature of the years' mining was the fact that a large number of the dredges continued operations until the first week in November.

Arizona.—Gold is obtained in about equal proportions from gold-quartz mines, and from copper mines. The three most prominent mines of the first class are the Gold Road and the Tom Reed, both in Mojave

County, and the Golden Star, in Yuma County. Among the copper mines those most productive of gold are the Copper Queen, the United Verde and the Congress Consolidated. The bullion from the gold-quartz mines averages nearly 30 per cent. silver, but the big production of this metal comes from the above-mentioned copper mines, to which should be added the Calumet & Arizona and the Superior & Pittsburg, as among the prominent silver producers. In 1911, as compared with 1910, the output of gold decreased by $458,770 and of silver by 1,060,000 oz., due largely to diminished scale of operation at the Tombstone and the Commonwealth mines.

California (By L. H. Eddy).—The total production of gold in 1911 was about the same as that of 1910. While the dredges made an increased output, the quartz mines did not, because they were directing greater effort to the development of new ore bodies in old mines and to the exploiting of prospective mines. Advances in quartz mining were particularly notable in Nevada and Sierra Counties and in several counties of the Mother Lode district. In Nevada County, the North Star Mines Co. took over the old Champion group and enlarged the Champion shaft, which is to be used as the central working and pumping shaft. The working force was increased to 175 men. It is expected that the North Star mine output will exceed $1,000,000 during 1912. This company also devoted some attention to other districts, among them being the Nevada City and Alleghany districts. The Empire mine installed a 100-ton cyanide plant, took over on lease the Pennsylvania and W. Y. O. D. mines, and began work of extensive development and the installation of machinery. Other advances in the year include the Sultana, Cassidy, Cedar Hill, Mountaineer, Coan and others. The development of the Tightner and Red Star in Sierra County will be undertaken by J. M. O'Brien of San Francisco. The Tightner, of the Alleghany district, said to have over $3,000,000 of ore developed. In the course of development, $1,000,000 was produced. A new electric plant and compressor and 10 stamps were installed. The Plumbago, one of the largest producers, increased its production and started a new shaft. At the Independence, new machinery was installed. The Caesar began operations in November.

Notable features in the Mother Lode district were the installation of electric power, new hoists, and added milling capacity. The Bunker Hill, South Eureka, Central Eureka and Argonaut completed such installations. The Kennedy and Zeila continued hoisting with steam although the mill of the latter is run by electric power. Among the old mines reopened are the Plymouth Consolidated where preparations were made in December for the installation of modern machinery by Bewick,

Moreing & Co.; the Georgia Slide was reopened by Sacramento men, who did development work. The Lightner mine in Calaveras County did extensive development work and paid a large amount of the purchase price. The Lincoln and Lucky Baldwin are among the mines which were reopened. Activity in development was not confined to the Nevada-Sierra region and the Mother Lode, but extended to nearly all the gold-producing counties of the State. Plumas drew capital along the line of the Western Pacific; Butte and Yuba developed new prospects and Mariposa attracted considerable attention by improvements on the old Fremont grant and the opening of several small mines. A notable feature of the year in Trinity County was the installation of stamp mills and cyanide plants. In the last half of the year, the bonding of a number of old mines in Weaverville basin gave new interest to the section.

Colorado (By George E. Collins).—No bonanza discovery was recorded in 1911. Cripple Creek retains its position of undisputed supremacy in the production of gold and is likely to do so for many years to come. Golden Cycle, Portland, Vindicator, Elkton and El Paso upheld the lead for a number of years. The Golden Cycle has displaced the Portland as the premier producer and it is understood that it has, by far, the largest ore reserves. A special dividend of $1,500,000 was recently declared and may have some relation to the suit that has been brought against the Golden Cycle by the Christmas Mining Co. It is maintained that the Christmas Mining Co. contains the apex of one of the most extensive ore bodies in the Golden Cycle. The plant has been closed presumably on account of the difficulty in obtaining ore supplies. Raw cyaniding, preceded by concentration is a proved success at the Independence and Portland mills. At the Ajax, a new mill, employing the Clancy method, is being completed.

In San Juan district, the Liberty Bell, Smuggler-Union and Tomboy maintain large production. The Smuggler-Union has, however, seen its best days. The vein contains considerable zinc, for which reason the mill is being remodeled. The life of the Tomboy will probably be greatly extended by the recent purchase of the Montana claims, which will be worked through the Ophir tunnel. At Ouray the largest production during recent years has been furnished almost entirely by Camp Bird, now practically exhausted. The only large producer in the Silverton district is the Sunnyside. Continuation of the large ore bodies formerly worked in the Gold King, which were cut off by faulting, has not been found. In the Bear Creek district, in the southeastern part of San Juan County, tellurides have been found and experiments indicate that these will be treated by modifications of the cyanide

process, preceded in most cases, by wet concentration. The main drawback is that the ore bodies are small; difficulties in transportation have not made it advisable to build the necessary custom mills before now. In the San Juan district, the difficulties confronting the industry are the treatment of two classes of ores, those requiring cyanidation and those requiring separation of zinc sulphides. The production of San Juan County fell off in 1911 on account of floods of the Animas River. This blockade lasted from Oct. 4 to Dec. 10.

The Creede and Aspen districts continue to make a large production of silver. In the Aspen district, the Consolidated Smuggler and A. J. properties have been unwatered through the Free Silver shaft. An expected revival in Gilpin County has been delayed largely by the lack of resources. The Gregory-Bobtail is idle. The Newhouse tunnel has drained the Gunnell and partially all the other mines north of Nevada Gulch. The Quartz Hill mines are, however, uneffected. The largest producer has been the Frontenac, the new concentrating plant of this company having commenced operations early in the year. In Clear Creek County, the bulk of the production has been made by lessees, the largest producers being the Gem and Bellman. These are worked through the Argo and Central tunnels. In the Cripple Creek district the Roosevelt tunnel has drained the Portland mine, although a little delayed. A large production was not obtained until November and December. The El Paso mine dried the latter part of the year at the 1000-ft. level. Toward the end of the year, the flow from the portal of the deep drainage tunnel was 9200 gal. per minute. The level of the water had been lowered 85 ft. during the last 10 months of the year. The tunnel is still being driven, headed for the Gold Dollar watercourse.

The Cresson Consolidated Gold Mining and Milling Co., of Cripple Creek, has been a dividend payer for 6 years. For 5 years the shipments were 2500 to 3000 tons of medium-grade ore per month. During 1911 the ore on the lowest level increased in value so that only 1600 tons per month will be mined but with an increased dividend.

The report of the Elkton Consolidated Mining and Milling Co. for the year 1911 shows that the value of ore mined and shipped was $340,111. Dividends paid during the year were $225,000, and to date $2,929,460. Toward the end of the year, the sinking of the main shaft and the opening of the 1000-ft. level was possible with the receding water. The grade of ore mined during 1911 was lower than the preceding year by about $4.50 per ton. This is because the bulk of it was taken from depleted ore bodies above the ninth level. Shipments of ore, including that of lessees, amounted to 21,026 tons of an average value of $16.28

per ton. It is stated that the company sorted as waste 42 per cent. of the total quantity mined.

Idaho (By Robert N. Bell).—Gold production in 1911 showed a marked improvement over the previous year, owing to successful operation of two new dredging enterprises. (See under Dredging.)

Discoveries were made near the head waters of Big creek, a tributary of the middle fork of Salmon River, in Idaho County, 100 miles by wagon road east of Meadows, the nearest railroad shipping point. This is one of the best watered and timbered parts of the state which may prove an offset for the present isolation. Here an immense zone of gold-bearing mineral strikes north and south across the high mountain spurs. The elevations range from 6000 to 8000 ft. The lode can be traced along its strike for 10 miles. The ores contain antimony and have proved difficult to treat, but the problem of producing bullion is believed to have been solved in 1911. At the Moore property, the total width of this zone is 250 ft. Here 1500 tons of ore were mined and milled in a crude stamp mill, with a 4-mesh screen, yielding $5 per ton to plate amalgamation. A sample of clean concentrates from the sulphide ore below the tunnel level assayed $32 per ton in gold and silver, of which $6 was silver, and showed 94 per cent. extraction by fine grinding and agitation in a cyanide test.

At intervals of about 2 miles to the north from this mine three other properties show similar great ore-shoots. On the Laufer & Davis property, the croppings indicate a shoot of ore over 100 ft. in width. Following north on the strike, there is another similar shoot of ore on the Golden King group.

At Smith Creek, 2 miles farther north, the Independence mine has a body of white quartz 200 ft. thick, impregnated with iron sulphide assaying $4 to $5 per ton in gold and silver, of which two-thirds is gold and one-third silver.

At Edwardsburg, a 40-ton concentrating and cyanide plant has been completed at the Eagle Mines. In Elmore County, the Homestake at Neal, with the best equipped cyanide plant in the state, continues to treat 60 tons per day. The ore runs $10 to $15 per ton. The Yellow Jacket mine at the head of Blackbird creek was experimenting throughout the year on the treatment of its ores. This property is equipped with 60-stamp mill. The Gold Hill Iowa mine at Quartzburg is credited with an output of $3,000,000 in its history. The enterprise has been embarrassed by lack of capital, but present owners have put in a 20-stamp mill. The ore is difficult to treat.

The yearly production of the Delamar mine amounts to several hundred thousand dollars in gold and silver. The Rich Gulch Mining

Co. started the erection of a large mill to try its ores. The Banner mine, also in Owyhe County, promises to develop. It is next to the old Trade Dollar that produced $20,000,000 but is now closed. The mines at Flint were transferred to an English investor who is employing a large force of men in their development. At the South Mountain mines, 12 miles south of Flint, the gold and silver are combined with copper and lead sulphides. The first smelting plant in Idaho existed on this property.

At the Kitty Burton mine on Indian creek, Lemhi County, the continuation of the faulted vein has been found after driving a 1000-ft. cross-cut tunnel. The ore is 40 ft. thick in places and is worth about $12 a ton in gold. Experiments are being made with cyaniding in the 15-stamp mill on the property. The completion of the Gilmore & Pittsburgh Railway is stimulating resumption at a number of properties in the neighborhood.

Montana (By F. L. Fisher).—Mining activity was renewed in a number of districts, which have been practically deserted since the slump in silver about 18 years ago. In the Georgetown district in Deer Lodge County a number of old properties were rejuvenated and work started on several new ones. The Hidden Lake group was worked until Oct. 1 by a leaser, and since then by the owners. The George Hill Mining Co. was incorporated by Anaconda men, and some work was done on the George Hill claim. The Silver Lake Mining Co. operated the George mine and made shipments of ore to the Washoe smeltery. Other properties where mining was conducted in this district were the Venezuela, Old Bonanza, Roundup, Oro Fino, Holdfast, Montana, Twilight and the Cochrane.

The Radersburg district in Broadwater County was also the scene of considerable activity. The Keating Gold Mining Co. operated steadily and made regular shipments of ore. At the Black Friday mine development work was done on the 600- and 700-ft. levels. The Calumet & Corbin company sank a shaft and developed its property to some extent. The Black Diamond group was operated by a company of that name composed of Butte men. The Ohio Keating company also operated and shipped ore. The Combination Keating company installed an electric hoist and developed some ore. The De Lome Gold Mines Co. was incorporated and operated the De Lome and Sunlight claims after installing an electric hoist for sinking purposes.

In the Marysville district in Lewis and Clark County, mining was done at a number of properties. The St. Louis Mining Co. resumed operations after winning its suit over disputed ground against the Montana Mining Co., and treated its ore in the 60-stamp mill of the

latter company. The Bald Mountain and Belmont claims were operated by Col. Cruse, of Helena.

Madison County took the lead in the production of gold, due principally to the working of the four gold dredges of the Conrey Placer Co., at Ruby, which produced steadily from the first of the season. A mill was erected at the Gold Coin mine in the Rochester district, Madison County. The Easton mine mined about 25 to 30 tons of ore daily, which was treated at the mill at the property. The Alder Gulch Consolidated Mining Co. operated its placer ground near Virginia City from the middle of April.

The Elkhorn silver mine at Elkhorn was worked steadily during the year. In Lincoln County several companies operated placer mines on Libby Creek, near Libby. At the Libby Placer Mining Co.'s property work was begun early in the season after a shutdown of 7 years. The Comet Placer Mining Co. operated its property until forced to suspend operations in October, owing to lack of water. The Snowshoe mine, in Lincoln County, was operated from early spring, and the ore run through the concentrator on the ground. Shipments of the concentrates were begun late in the fall.

In Gallatin County placer gold was discovered in the West Gallatin Basin near Bozeman. The property of the Bi-Metallic company in Granite County was worked in a small way by leasers, who made regular shipments. The Snow Creek Mining & Milling Co. operated the Snow Creek Mine in Cascade County, the ore being treated in a cyanide mill erected there over a year ago. The Johannesburg mine was considered worked out and closed down in March. A gold dredge was constructed by the Kennedy Creek Gold Mining Co., owning 400 acres of placer ground in the Nine-Mile district near Missoula, and operations started late in the year.

Nevada.—Two important tendencies were noted during 1911: (1) concentration; (2) deep prospecting. There was a tendency throughout the year for larger companies to absorb smaller ones, or to take over properties strategically situated. This was evident in Pioche and Tonopah. The tendency toward deep prospecting was made plain at Goldfield in the formation of the Goldfield Deep Mines Co. Noteworthy progress was made at the Nevada Wonder, Nevada Hills and Mason Valley properties. Jarbidge and several other districts received attention from prospectors and investors, but no important developments occurred. With a few exceptions, work was confined to sound, legitimate development of the older properties.

Goldfield.—The two large producers in 1911 were the Goldfield Con-

solidated and the Florence-Goldfield; of these the former maintains its position as the largest gold producer in the world.

At the Florence-Goldfield production continued at the rate of from 100 to 170 tons per day, a satisfactory recovery being maintained from low-grade ore. New mill equipment was added during the year, which resulted in a reduction of costs and an increase in capacity. In December a fire occurred which destroyed the mill and its machinery, entailing a loss estimated at about $400,000. Slight damage was done to the shaft collar. The company has not yet announced its plans in connection with the rebuilding of the plant.

Leasing was continued at the Goldfield-Belmont, but during the year it was reported that no more leasing would be done; that a deep shaft was contemplated, to open the property at depth and that exploration would be carried on from this shaft. A change of control occurred in the Silver Pick Mining & Milling Co., and it was announced that a deep shaft would be started in January, 1912.

An event of interest was the formation of the Goldfield Deep Mines Co., about the middle of the year. The company was formed to operate at depth in the Goldfield district, and controls the Goldfield Merger Mines Co. In October an announcement was made that operations would start soon on the property of the latter company, and that a 4-compartment shaft, from 1500 to 2000 ft. deep was planned. The St. Ives was one of the properties included in the merger.

According to its annual report for the year ending Oct. 31, 1911, the Goldfield Consolidated Mines Co. produced 330,549 tons of ore of an average value of $32.55 per ton, a total of $10,760,198. Metallurgical losses amounted to $1.81 per ton, a total of $597,071, leaving $10,163,127 realized from the year's production, an average of $30.74 per ton. Gross expenses, including mining, milling, transportation, general, all construction, local, state and Federal taxation, amounted to $2,636,281, leaving a net operating realization of $7,526,846, ($22.77 per ton), an increase of $179,154 over the previous year, notwithstanding that the metallic contents realized were $110,806 less than the previous year. Additional milling facilities are being provided at an estimated cost of $70,000, whereby large economies are anticipated in the treatment of the concentrate residues over the past cost of shipment. The results of this construction will narrow the company's output to bullion alone, all by-products being handled locally at maximum economy.

The company purchased the Bulldog Fraction for $13,000, the Jumbo Fraction for $2500, and a one-half interest in the Vinegerone claim for $195,000. The stoping cost for the year was $2.29 per ton of ore mined; development cost, $1.06; total mining cost, $3.35 per ton. Development

work, amounting to 46,739 ft. was performed at a cost of $7.51 per ft. The total cost of mining and milling, including concentrate treatment, was $5.62 per ton, or 66 cents per ton less than the previous year. Other expenses, including marketing, income and bullion taxes, but not including construction, bring the cost to $7.55 per ton. Reserves in all the company's mines were estimated at 600,000 tons, a 2-years' supply at full capacity.

The Tonopah Mining Co., during the year maintained its position as the first producer of the camp. During November, 14,799 tons of ore were milled with an extraction of 91.81 per cent. The ore averaged gross $20.75 per ton and net profits for the month were $172,316. The Tonopah-Belmont company did much development work and started the construction of a new mill. This plant was originally planned to include 60 stamps, eight tube mills, concentrators and a complete cyanide plant; the whole was designed to treat 500 tons of ore per day. The company maintained a good production throughout the year and in November sent 7807 tons of ore to the mill and 2730 tons to the smeltery. Net profits for that month were $156,459. In February a fire in the East Belmont shaft did considerable damage and caused the death of 17 men.

Mining and development were conducted at the Tonopah Extension and in December it was announced that the company had paid the final installment of its bonded indebtedness and had a surplus. In April the MacNamara company announced its intention of building a mill of 2000 tons monthly capacity. On November 14 the Montana-Tonopah company made the first payment on the Commonwealth mine at Pearce, Cochise County, Ariz., which it purchased on a working option. The West End company purchased the Midway mill at Tonopah, with the intention of reconstructing it, its original capacity being 120 tons per day.

The Comstock.—Revival of operations at the Comstock mines continued throughout the year and while assessments were still the rule, dividends were paid by the Mexican, Ophir and Comstock-Phoenix companies. Underground developments were generally satisfactory. The following mines were producing during the latter part of the year: Ophir, Mexican, Union Consolidated, Crown Point, Yellow Jacket, Chollar, Potosi, Gould & Curry, Justice and Comstock-Phoenix. Of these the Ophir, Mexican and Comstock-Phoenix produced high-grade ore.

The Nevada Hills company at Fairview continued the construction of its 20-stamp mill and finished it during the latter part of the year, when it was placed in commission. In August, the mill of the Nevada Wonder company at Wonder, was started. During 1911 the Mason Valley company continued underground development work and the

construction of the smeltery and electric power line, the latter being 69 miles long. Mining in the Eureka camp was retarded by lack of adequate transportation. Activity continued in the Jarbidge and Manhattan camps, chief interest in the former having been aroused through an option secured by George Wingfield on the Bluster group.

Pioche District (By A. L. Scott).—On Dec. 15, 1911, the Prince Consolidated made its first shipment of four cars of tailings from the old dump at Bullionville, over the new spur just completed by the Salt Lake Route. Bullionville is near Panaca and about 14 miles from Pioche. The tailings are from an old pan-amalgam mill which handled the ores from the Raymond & Ely in the early days of the camp. The first dump to be shipped, at the rate of 200 tons per day, contains about 15,000 tons wihch is expected to net about $5 per ton. It is being shipped to the new International smeltery at Tooele.

At the Prince mine about 5000 tons of ore, already extracted, are awaiting completion of a railroad to that point which was contracted to be finished in May, 1912, and blocked out in the mine are about 1,000,000 tons of maganese ore, occasionally high in lead and silver. The projected road will pass in the immediate vicinity of the Pioche mines, Ely Valley, Pioche Metals, Pioche Demijohn, Abe Lincoln, Pioche Coalition and the Pioche King; the mines in the Highland district, such as the Mendha and the Highland Mary, will also be able to save $1.25 per ton on shipments.

The reorganization of the Nevada-Utah means early shipments from the famous black vein on the hill just back of Pioche town, and the financing of the Day-Bristol tunnel in the Jack Rabbit and Bristol districts about 15 miles from town.

South Dakota.—The output of gold in 1911 was $2,050,000 more than in 1910, which was the only importnat increase recorded in any State. The leading contributors are the Homestake, Mogul, Golden Reward and the Wasp No. 2.

The latest report of the Homestake Mining Co. covers two periods, the year ended June 1, 1911, and an iterim period from June 1 to Dec. 31, 1911. Hereafter the fiscal year will be the calendar year. During the first period 1,468,263 tons of ore were milled and 888,057 in the second. On the assumptions that 5 per cent. of the ore milled is reground that 60 per cent. is leached as sand and 40 per cent. treated as slime, regrinding costs 28 cents, cyaniding sand 22 cents, and slime treatment 27 cents per ton of material actually treated.

The broken ore in the stopes on Jan. 1, 1912, was 1,883,277 tons, and it is estimated that enough ore is blocked out to supply the 1000 stamps for 20 years. Water was turned into the hydro-electric plant at Spear-

fish early in 1912 and on Mar. 15, the full available head was turned into the ditch and through the tunnel. The generating station, pipe lines, etc., were given a thorough test.

COSTS AT THE HOMESTAKE MINE.

	June 1, 1910 to June 1, 1911.		June 1, 1911 to Dec. 31, 1911.	
	Amount.	Per Ton.	Amount.	Per Ton.
Assaying	$32,164	$0.0219	$12,237	$0.0137
Surveying	24,364	0.0165	16,441	0.0185
Mining	2,726,622	1.8570	1,529,939	1.7219
Milling	410,651	0.2797	244,857	0.2756
Regrinding	20,884	0.0142	12,273	0.0138
Cyaniding sand	204,099	0.1390	108,463	0.1219
Cyaniding slime	155,098	0.1056	95,359	0.1073
Net sundry expense	371,621	0.2534	231,788	0.2606
Total costs	$3,945,503	$2.6873	$2,251,357	$2.5335
Spearfish hydroelectric power plant	393,088	.....	231,134	.....
Dividends	1,310,400	.....	764,400	.....
Surplus for period	−397,537	.....	414,261	.....
Bullion return	$5,251,454	$3.5766	$3,661,152	$4.1205

The Wasp No. 2 treats profitably $2.50 ore. Handling 500 tons per day, the mine produces $30,000 per month, $10,000 of which is being disbursed as dividends. At the present time, a slight change in mining methods is being perfected which will cheapen the cost of mining. The skip tracks in the open cut are being dropped below the level of the bottom of the quartzite ore body, and 18-in. mine tracks are being laid at right angles to them. On these tracks 1-ton ore cars will be used. The duty per man will then be about 17 tons per day instead of 14 tons, as previously. A steam shovel was installed late in 1911 for the removal of overburden.

The Bismarck Consolidated Company, adjoining the Wasp No. 2, built a 300-ton mill in 1911, designed for dry crushing and cyanidation. The ores are identical with those at Wasp No. 2.

In the Bald Mountain district, gold ores occur in shoots smaller in size and more irregular in shape than in the Black Hills. Two classes of ore are mined. The oxidized has the composition: SiO_2, 84; Al_2O_3, 4; Fe_2O_3, 7; SO_3, 4; CaO and MgO, 1 per cent; Au, 0.57 oz.; Ag, 2.80 oz. per ton. The sulphide or "blue" ore analyzes: SiO_2, 69; Al_2O_3, 3; Fe, 14; S, 12; P_2O_5, $CaSO_4$ and $CaFl_2$, 2.5 per cent.; Au, 0.32 oz.; Ag, 10.55 oz. per ton. The precious metals are thought to be in the form of tellurides. The Golden Reward has recently discovered a body of ore consisting mainly of clay and pyrite, which is not only unusual in character but

was found well outside the supposed boundaries of the productive area. The oxidized ores are easily cyanided, and the sulphide ores also, after they have been roasted.

Five cyanide plants are now operating on Bald Mountain ore, viz., Mogul, Golden Reward, Reliance, Trojan, and Lundberg, Dorr & Wilson. These plants are treating 1025 tons per day, and with the second unit of the Trojan mill this total will be increased 150 tons per day. The Golden Reward is having plans drawn for a new mill of 500 tons capacity, which will give that company a daily output of nearly 800 tons. In addition to this, the Imperial company, now shipping to Lundberg, Dorr & Wilson, is figuring on starting its own mill.

Erection of a smelter was begun at Galena. This plant will use the pyritic process, securing the pyrite from the Gilt Edge-Maid mine, the latter company having a concentrating plant well under construction on a site adjoining the smelter. Limestone will be secured from a carboniferous bed on the hill overlooking the smelter; siliceous ores will be obtained from Galena, Bald Mountain and adjacent camps, and the coke will be the only material hauled any distance.

In the Bald Mountain mines blue ore worth $8 to $12 per ton, will be handled by the new plant, and ores from the Galena district, now being shipped to the smelteries at Denver and Omaha, will find a market right at home.

The Gilt Edge-Maid mine shows a body ore almost 100 ft. wide that will concentrate into a good grade of clean iron pyrite. Another big producer in the Galena camp is the Sitting Bull, the output of which has been shipped to Denver.

Utah.—The Consolidated Mercur was the chief producer of gold from silicious ores. This company treated from 600 to 700 tons of ore daily. The February cleanup brought $45,000. The annual report for the year ended June 30 shows that 239,190 tons were treated, averaging $3.21, the tailings averaging 88 cents. This is lower than in former years because the tonnage included 12 per cent. of the old tailings retreated with the ore. Mining costs were $1.29; milling, $1.04; total, $2.33 per ton. The gold produced for the year was 26,675 oz., which cost $20.92 per ounce to produce, making a loss of 25 cents per ounce. There was a small gain for the year from receipts from other sources.

The greater part of the silver produced came from the Tintic district in Juab and Utah Counties, and from Park City in Summit and Wasatch Counties. The silver in the copper and lead ores from Salt Lake County also added appreciably to the output. Bingham produced an increased tonnage of argentiferous lead ore from the Utah Apex, Highland Boy, Yampa, and other properties.

Gold and Silver in Foreign Countries

Australasia (By F. S. Mance).—The production of gold in the Commonwealth of Australia and New Zealand is given in the accompanying table.

PRODUCTION OF GOLD IN AUSTRALASIA.
(In fine ounces.)

State.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
West'n Australia	1,983,230	1,955,316	1,794,547	1,697,554	1,647,911	1,595,269	1,470,632	1,370,867
Victoria	765,596	747,166	772,290	695,576	670,910	654,204	569,644	503,541
Queensland	639,151	592,620	544,636	465,882	465,085	455,577	441,400	381,845
New South Wales	269,817	274,267	253,987	247,363	224,792	204,709	188,857	181,121
Tasmania	65,821	73,541	60,023	65,355	57,085	44,777	37,048	37,992
South Australia (a)	29,177	20,330	24,439	10,651	9,162	7,111	6,603	10,700
Total Commonwealth	3,752,792	3,663,240	3,449,922	3,182,381	3,074,945	2,961,647	2,714,184	2,481,066
New Zealand	467,898	492,954	534,616	477,312	471,967	472,451	446,421	411,390
Total Australasia	4,220,690	4,156,194	3,984,538	3,659,693	3,546,912	3,434,098	3,160,605	2,892,456

(a) Northern Territory is included with South Australia.

PRODUCTION OF SILVER IN AUSTRALASIA.

State.	1909.		1910.		1911.	
	Ounces.	Commercial Value.	Ounces.	Commercial Value.	Ounces.	Commercial Value.
New South Wales (a)	10,584,791	$5,451,167	12,804,659	$6,826,050	14,529,145	$7,745,810
Queensland	1,001,383	515,712	861,202	460,743	549,015	273,642
South Australia	1,660	855	6,250	3,344	1,400	680
Tasmania	1,534,780	790,400	656,793	351,380	(e)700,000	373,100
Victoria	21,655	11,152	18,800	10,157	(e)19,000	10,127
Commonwealth	13,144,269	$6,769,286	14,347,704	$7,651,674	15,798,560	$8,403,359
New Zealand	1,813,831	879,038	1,677,484	897,454	1,310,943	698,733
Total	14,958,100	$7,648,324	16,025,188	$8,549,128	17,109,503	$9,102,092

(a) Metal produced in Australia, plus silver contents of concentrates exported. (e) Estimated.

Western Australia continues to furnish the largest part of the gold, but in comparison with 9 years ago, there was a drop of over £2,750,000 in value. The mines which gave such good yields from the upper levels have failed to maintain their grade with increasing depth. Thus the value of the ore mined in this state in 1903 was 77s. per ton; in 1910 it had fallen to 42s. a ton, and to a still lower figure in 1911. There have been some recent promising developments but not of such extent as to arrest the downward tendency of the yield.

Victoria ranks next in importance for its gold production, but in this state, also, developments at the deeper levels have shown declining grade, and the yields from the old established fields, such as Bendigo and Ballarat, fall far short of the average for the past 6 years. At Bendigo

a revival on the parallel saddle reefs was an offset to the flooding of the Victoria quartz and adjacent mines.

In Queensland the decrease is likewise noticeable. The diminishing yields from fields such as Charters Towers and Gympie have had a depressing effect on prospecting, attention being given to the search for other minerals, and any immediate improvement in the gold yield cannot be hoped for under existing conditions. The Mount Morgan mine for the year ending May, 1911, estimates reserves at 1,543,000 tons, containing 3 1/2 per cent. copper and $8.00 gold, and 2,070,500 tons carrying 3 per cent. copper and $2.50 gold. The chlorination works treated 109,371 tons of silicious sulphide ore, averaging 0.516 per cent. of copper and 9.36 dwt. of gold, recovering therefrom 565 tons of copper precipitate and 51,210 oz. of gold; an additional yield of 1287 oz. of gold was obtained from a cleanup of oxidized ore treated in the preceding year.

In New South Wales as in Queensland, mining for silver, copper and tin commands most attention. The Cobar district is the chief center of gold-mining operations, and now that the Great Cobar, Ltd., has acquired the Cobar gold mines, a good increase may be looked for from the treatment of auriferous-copper ores. The yields from the other fields and from the dredges were disappointing.

The amount of silver produced in New South Wales in 1911 was 5,731,468 oz., to this should be added the silver contents of the silver-lead concentrates shipped to Europe, which was 8,797,677 oz., making a total of 14,529,145 oz. of silver, worth £1,593,788, derived from the various lead-silver-zinc mines of the state. The mines at Broken Hill, except Block 10 and the Junction, worked uninterruptedly throughout 1911. The Proprietary resumed after a year's idleness. The output of the district in 1911 was 1,484,397 long tons of ore, almost all sulphides, having an estimated value of £3,832,430, out of which dividends to the amount of £651,125 were distributed.

The decrease in New Zealand was mainly due to the restriction of the output from the Waihi mine. Recent developments in the lower levels of the Martha Lode have gone to confirm the previous evidence of decreasing assays as depth is attained. This mine has now produced gold to the value of about £10,000,000, and returned £4,300,000 to shareholders. Other important contributors are the Talisman and Waihi Grand Junction mines, while there were over 100 dredging plants which supplied a good quota of the production. The Talisman mine works a number of reefs containing some sulphides of copper and iron and has a depth of 1300 ft. from the outcrop. The bullion carries from 10 to 30 oz. of silver to one of gold.

The bulk of the gold mined in Tasmania comes from Beaconsfield,

on the Tamar River, near Launceston. This is an exceedingly rich mine, but on account of the great cost of pumping, the net profits in recent years have been small, and in 1911 there was a slight loss.

Belgian Congo (By Sydney H. Ball).—The State budget for 1911 for prospecting and mining was about $310,000. The State operates in the northeastern part of the colony, its chief placer being at Kilo, to the west of Lake Albert. The product of the State mines in 1910 was 876 kg. of gold, valued at about $525,000; the product for 1911 was about 700 kg. or perhaps $420,000. This falling-off was due to the fact that some of the richest streams have been exhausted, and to labor and food shortages. No exploitable vein has yet been found. The country rock is granite gneiss and granite. Hydraulic mining is being done in some of the more favorable streams, but the greater recovery is from sluices. A dredge, ordered some years ago, is still in Uganda, due to the difficulties of transport of the heavier parts. Thirty to 40 whites are employed, but the black personnel, once 3000, has fallen off markedly. Supplies arrive by way of Mombasa, whence they are shipped by the Uganda railroad to Port Florence, thence by boat across Lake Victoria Nyanza to Entebbe, and from there on are carried some 200 miles on the backs of natives. Considering this method of transport, the reported cost of sluicing (40 cents per yard) does not seem excessive. The importance of the alluvial deposit at Nebula, about 110 miles N. N. E. of Stanleyville, was apparently overestimated. The State worked this mine in 1911, but presumably it will be abandoned in 1912, with perhaps a total output of $20,000 to $50,000 gold. Operations were started by the State in 1911 on the glod placer at Moto, about 325 miles northeast of Stanleyville.

The Société Internationale Forestière et Minière du Congo (a Belgian-American company) during 1911 bent its efforts largely to prospecting, with satisfactory results. About 30 American and English engineers and prospectors and 700 blacks are employed. Toward the middle of 1911 it began to operate a rich, though rather small, placer at Kanwa, 3 miles from Nebula. An unusual dearth of water for sluicing kept the product down for the half-year of 1911, but it should be largely increased in 1912. The total gold production of the colony for 1911, including private mines, is estimated at $500,000 to $600,000.

Bolivia.—Gold is found along the slope of the Cordillera Real and also in the beds of the rivers which flow east from this range. The annual production of gold in Bolivia may be reckoned at 17,460 oz. In 300 years the production of Bolivia is estimated at $206,000,000, mostly from alluvial deposits. Such lodes as the Yaui, Pallaya and Araca can be worked economically only on a small scale. Dredging is

to begin on the Rio Kaka by the Incahuara Gold Dredging Co., Ltd., a London firm; the bars run between 78 and 33 cents per yard. The richest silver region in Bolivia lies in the west, although in the north, south and east there are some silver lodes near the summit of the Andean chain. The total production of silver in 1909 was valued at 8,060,461 bolivianos as compared with 11,769,040 bolivianos in 1902 and 18,431,198 in 1895. The production in 1910 is estimated at 7,000,000 oz. of silver. The falling off in this output is due to the attractiveness of tin mining, the price of tin having risen and that of silver having fallen. The tin-bearing areas lie close to the silver-bearing areas.

Canada.—Gold production in 1911 is estimated at $9,762,096 which, compared with the 1910 production, $10,205,835, shows a falling off of $443,739. The Yukon placer production in 1911 is estimated at $4,580,000 as against $4,550,000 in 1910, the total exports on which royalty was paid being 277,431 oz. in 1911 and 275,473 oz. in 1910. British Columbia production in 1911 was $4,989,524, of which the placers contributed $468,000, smelter recoveries and bullion obtained from milling ores being valued at $4,521,524. Production in Nova Scotia is estimated at $142,000 all from milling ores. In Quebec there was a small recovery from alluvial workings and a small content in the pyrite ores shipped, the total value being $12,443. Ontario produced $37,929. Exports of gold-bearing dust, nuggets, gold in ore, etc., in 1911 were valued at $7,493,523. Gold was imported during 1911 in bars, blocks, ingots, etc., to the value of $924,233.

The silver production of Canada fell off about 128,516 oz. The total production of 1911 is estimated at 33,486,938 oz. of which 31,507,880 oz. were from Ontario, 1,910,323 oz. from British Columbia, 50,300 oz. from the Yukon and 18,435 oz. from Quebec. The production in Ontario was slightly greater than during the previous year while British Columbia fell off nearly half a million ounces. For British Columbia the figures represent recovery as silver bullion or silver contained in smelter products while for Ontario the figures represent total silver content of ore and concentrates shipped, less 5 per cent. allowed for smelter losses, together with bullion shipments. Total shipments of ore and concentrates from Cobalt District and adjacent mines were 16,234 tons, containing approximately 28,817,198 oz., in addition to which 3,334,052 oz. were shipped as bullion. The average silver content of ore and concentrates shipped was thus about 1744 oz. or $929.62 per ton as compared with an average of 867 oz. in 1910 and 840 oz. in 1909. The 1911 shipments were chiefly high-grade ore averaging over 3400 oz. and concentrates averaging over 850 oz. Shipments in 1910 were 28,684 tons of ore containing 23,797,111 oz. of silver or an average of 830 oz. per ton; 6943 tons of

concentrates containing 7,111,579 oz. or an average of 1024 oz. per ton; and bullion containing 1,003,111 fine ounces.

Exports of silver in ore, etc., were 31,216,725 oz., valued at $15,807,-366. Imports of silver in bars, blocks, sheets, etc., were valued at $847,645.

British Columbia (By W. F. Robertson).—Practically all the placer gold comes from the Cariboo and Cassiar districts, only about one-thirtieth of the total production coming from the remaining districts of the province. The gravel-washing season was shorter than usual in both Cariboo and Cassiar, hence the smaller total recovery of gold. In both districts provision is being made for operating on a large scale in the future. Placer mining on a small scale is done in several parts of Fort Steele division of East Kootenay; in Nelson, Revelstoke, and Trout Lake divisions of West Kootenay; in Boundary, Similkameen, and Yale districts; and in Lillooet and Clinton, which last-mentioned divisions yield twice as much placer gold as any other of the smaller producing districts.

Less lode gold was produced in 1911 than in any other year since 1907. About one-half of the decrease resulted from suspension of production at two or three Boundary district mines, though this was in part compensated for by larger production by the British Columbia Copper Co. and Hedley Gold Mining Co. Approximate figures of totals of lode gold from individual districts give an apportionment about as follows: Rossland, 117,000 oz.; Boundary, 51,000 oz.; Similkameen, 30,000 oz.; Nelson, 20,000 oz.; and Coast, 7000 oz. Of Rossland mines (which are in Trail Creek mining division), the Consolidated Mining & Smelting Co.'s Center Star group produced nearly three-fourths of the total, while the mines of the Le Roi No. 2, Ltd., yielded the greater part of the remainder. Hereafter the production of the Le Roi (not of Le Roi No. 2) will probably be included in that of the Center Star group, for the Consolidated company last summer added that mine to its other properties in Rossland camp. In the Boundary district the bulk of the gold was from mines of the British Columbia Copper Co. and the Granby company, while in Similkameen the Hedley Gold Mining Co. made a production estimated at the quantity above shown. In Nelson, the Queen, Granite-Poorman, Nugget, Arlington, Second Relief, and Yankee Girl mines together made up the output of lode gold from that division. On the Coast, the Marble Bay mine was the chief producer and the mines of the Portland Canal Mining Co. yielded the bulk of the remainder.

Of the estimated total production of 1,910,000 oz. of silver, Slocan mines contributed about 875,000 oz.; East Kootenay, 360,000 oz.; Boundary, 310,000 oz.; Nelson and Rossland, 180,000 oz.; Lardeau,

70,000 oz.; and the Coast, 125,000 oz. The mines that produced the silver in considerable quantity were as follows: In the interior parts of Slocan district, the Rambler-Car boo, Richmond-Eureka, and Ruth-Hope; and in the Slocan Lake section, the Van Roi, Hewitt (Silverton Mines, Ltd.), and Standard. More than one-half of the production of East Kootenay was from the Sullivan mine, and in excess of one-third from the St. Eugene. In Nelson division the Molly Gibson, and at Rossland the Center Star group, Le Roi, and Le Roi No. 2 company's group, made the output of those regions respectively. In Boundary district the mines of the Granby and British Columbia Copper companies contributed most of the silver from that part, while in Lardeau the Silver Cup, and on the Coast the Britannia, Marble Bay, and the Portland Canal Mining Co.'s mine each added its share. The Skeena country may be expected to add to the province's production of silver after railway transportation facilities, now being provided, shall be available.

Nova Scotia.—Gold production for 1911 was the smallest since 1862. The direct cause was the closing down of the Richardson mine at Goldboro and the Oldham-Stirling mine at Oldham. In all, 18,320 tons of ore was mined and crushed, producing 8390 oz. of gold, valued at $159,411 ($19.00 an ounce), being an average yield of $8.70. Operations were carried on at 21 properties in 18 districts, in very few cases, however, during the whole year. Much of the work was done by tributors.

Ontario (By T. W. Gibson).—The entire silver output came from the mines of Cobalt, which from their opening in 1904 to the end of 1911 have yielded 125,500,000 fine ounces of silver, worth say $64,000,000, each year showing an increase over the previous one. The phenomenally rich ore first obtained is now less in evidence, although by no means worked out, and more and more dependence is being placed on the low-grade concentrating material. While the aggregate tonnage of shipments from the camp was less than in 1910, the shipments of concentrates increased from 6874 tons to over 9000 tons. Some of the high-grade ores are also now being refined on the spot.

The Nipissing company introduced a novel amalgamation method for recovering the silver from their rich ores. Silver bullion was first produced in the camp in 1910 when 980,633 oz. were turned out. There are now 13 concentration plants in operation, and two more—at the Beaver and Cobalt Lake properties—are under construction, with a total capacity of about 1350 stamps.

Power conditions were unsatisfactory during the early part of 1911 when the continued severe frost checked the Montreal and Matabitchouan Rivers, the falls on which supply the camp with electric power, and also to some extent with compressed air. Hydraulic operations are being

projected by the Nipissing for removal of the drift on their property, thus facilitating the discovery of veins. Production costs for the camp average about 15 cents per ounce, while the costs for one mine, extending over a period of a year, have been as low as 7 cents per ounce.

While the United States smelters obtain the bulk of the tonnage from the Cobalt camp, the Canadian smelters get the pick of the ore. In 1911 the United States smelters at Perth Amboy, Carnegie and Denver treated 66.57 per cent. of the tonnage that left the camp, and the Canadian smelters at Coppercliffe, Deloro, Therold and Orillia only 32.5 per cent.; but, as the ore despatched to Denver and Perth Amboy in the majority of cases was low-grade, the bulk of the silver was smelted in Canada.

Porcupine (By T. W. Gibson).—The actual output of gold for 1911 was small, not exceeding 3400 oz. Genuine production would doubtless have been begun had it not been for the disastrous fires of last summer which destroyed the mills at the Hollinger and Dome. These mills are being rebuilt. The owners are satisfied that there is plenty of ore of milling grade. Numerous other properties are being developed. The Vipond mill is far advanced, and the McIntyre and several other mines are also proposing to put in stamps. The fire-swept towns of Golden City and South Porcupine have been rebuilt. The construction of the Temiskaming & Northern Ontario railway spur into the camp gave much needed facilities for transportation.

Electric power from Sandy Falls is already on the ground, and another large installation is going in at Wawaitin, both on the Mattagami River. At Swastika, Munro and Larder Lake camps are stamp mills and considerable activity. The St. Anthony mine at Sturgeon Lake, northwestern Ontario, has been taken over by Messrs. Glendenning, Mackie & Kennedy, who have overhauled the mill and resumed production of bullion. At West Shining Tree Lake, gold was found in quartz veins, and one of the largest and most promising claims, known as the Gosselin, Messrs. Mackenzie & Mann are preparing to test by means of a diamond drill.

Chile.—A notable discovery of gold was made at Putú, Chile, in September. Little work was done in the new field before the close of 1911, except by the discoverer who, under the Chilean laws, had a long period of time in which to measure his claims before any others could be registered. The Collahuasi mines, at an altitude of 4800 meters, are worked for their copper, with gold and silver as by-products. Small placers are worked for gold in the same region. The silver produced in Chile in 1911 is estimated at 1,500,000 oz.

Colombia.—A belt along the western coast line, 450 miles long and 20 miles wide, can be considered as prospecting ground for alluvial

deposits. The higher rugged country contains irregular deposits suitable only for hydraulic or drift mining. The deposits nearest the coast contain fine sands and muds in which the fine gold is difficult to extract. In 1910 the output was $3,370,000 in gold. The English syndicate that operates dredges at Oroville, Calif., owns a large area of placer ground on the Niche River near Zaragoza in Colombia. It was the intention of this company to dismantle some of the smaller Oroville dredges and use them in Colombia, but this plan was not carried into effect. The bulk of the production came from a number of small operations, mostly placer mines although a few lode mines were worked. The Tolima Mining Co., Ltd., operates the Frias silver mines and also holds under a 10-year lease the adjoining property of the Colore Mines, Ltd.

Costa Rica.—The Abangarez mine, in the province of Guanacaste, operated throughout 1911 and was the principal producer of the country. There were two mills in operation, with a total of 60 stamps dropping. At the Montezuma mines, in the province of Puntarenas, the mill was operated only in wet season, when there was sufficient water-power. The production was about $70,000. At the Aguacate mine, a 10-stamp mill was operated toward the end of 1911. The first shipment of bullion was received in New York in December. A small cyanide plant was being planned at the close of the year.

The report of the Abangarez Gold Fields for the year ended Dec. 31, 1911, states that 42,514 tons of ore was mined and milled, yielding an average of $21.84 per ton, or a total of $928,587. The cost of operation, including mine development, all repairs and renewals to equipment, and administration and marketing expense, was $721,997, leaving earnings of $206,590. During the year $225,635 was spent in the construction of a new power plant on the Guacimal River about 12 miles from the mill. The completion of this plant should obviate the necessity for the steam power plant, which, during 1911 cost the company about $7000 per month for fuel. Underground development work during the year increased ore reserves by 55,000 tons, after allowing for the tonnage milled. There is now 340,200 tons of ore developed of an estimated average assay of $13.29 per ton.

Ecuador.—A company was formed late in 1910 to operate placer ground in the province of Esmeraldas, and in 1911 a dredge was ordered. The principal producer of Ecuador was the lode mine operated by the South American Development Co. north of Guayaquil.

Guatemala.—The Guatemala Mining & Development Co. obtained a concession covering all the mineral deposits of the country. A corps of engineers was at once set to work for thorough exploration. The only producing property in 1911 was that at Quebradas de Oro, near Morales,

on the Guatemala railroad. This is a hydraulic mine, owned and operated by John Knight. The production is not made public, but has been variously estimated from $10,000 to $30,000 per year.

Guiana, British.—The production of gold for the fiscal year 1910–11 was 54,989 oz. against 64,830 oz. in 1909–10. This decrease was caused principally by the shutting down of a quartz and a hydraulic mine.

Guiana, French.—Some progress was made in dredging on the Elysee placers. The principal producer was the Syndicat Mana. The gravels are richer than those of Colombia, but hostile climatic conditions and transportation difficulties hindered progress even to a greater degree than in Colombia.

Honduras.—The New York & Honduras Rosario Mining Co., operating mines at San Juancito, north of Tegucigalpa, continued to be the principal producer. The new mill, to be equipped with 1800-lb. stamps, was not completed. Development work was continued. American engineers examined the iron mountain at Agalteca, near Tegucigalpa. Various small mines were in operation in the department of Choluteca, on the Pacific Coast. In the department of Olancho, some gold prospects were examined, and several placer deposits in other parts of Honduras were tested. During 1911, the New York & Honduras Rosario Mining Co. crushed 35,813 tons of ore, recovering therefrom 1,395,137 oz. of silver and 8103 oz. of gold, compared with 1,289,174 oz. silver and 10,621 oz. gold in the preceding year. Mining cost $10.21 and milling $4.58 (U. S.) per ton. Reserves at the end of 1911 had been increased to 121,088 tons worth $3,168,739, or $26.16 per ton.

India.—From 1883 to 1910 inclusive, the Mysore gold mine produced nearly £13,000,000 in gold, and paid in dividends £6,285,844. The deepest working is 2800 ft. vertically. The main shaft was sunk vertically 1360 ft. and on the incline 1866 ft. At the eleventh level (the lowest), the lode was reported to carry 1 oz. 7 dwt. for 6 ft. in width or 1 oz. 8 dwt. for 4 1/2 ft. Ore crushed during 1911 totalled 291,477 short tons, and by amalgamation and cyanidation, chiefly the former, gold worth £896,651 was recovered. The working cost was £335,341, royalty £53,201, income tax £26,318, depreciation £13,173. The dividend absorbed £366,000, being at the rate of 120 per cent. During the year 31,766 ft. of development work was done, and on December 31, 1911, the reserve was calculated at 1,300,369 tons, being an increase of 67,862 tons during the year. The development in depth was extremely satisfactory at many points. Edgar's shaft was put in commission, and is an important factor in decreasing the cost. It is intended to commence the sinking of a similar shaft in MacTaggart's section. Additional ground has been acquired to the west of the property on the dip of the

lode, so as to protect the deep levels. The grade of ore treated in 1910 was worth £3 12s. 9d. per ton, as against £5 0s. 1d. in 1901.

The annual report of the Ooregum Gold Mining Company, operating in Mysore district, stated that during 1910 development amounting to over 3 miles had increased reserves by 7000 tons, being estimated at the close of 1910 at 164,534 tons, or about 13 months' supply for the mill. The ore milled during 1910 averaged 12 dwt. 3 grains, a decrease of 12 grains from the tenor of the 1909 ore. The cost per ton milled, including 5s. a ton for development, amounted to £1 1s. 3d., a decrease of 3s. 2d. on the previous year's figures. Total receipts for the year were £341,535; the total expenditure £175,151, leaving a profit of £166,377. The company has taken an option on another property on the Kolar field and is developing it.

The first stage of a scheme for supplying power to the Kolar gold field and for providing a canal system for irrigation has been sanctioned by the durbar of the Mysore state. A great masonry dam, 4200 ft. in length, will be built across the River Cauvery, at a point 8 miles west of Seringapatam, at a cost of $2,700,000. The dam will eventually store 41,500,000,000 cu. ft. of water.

The various companies working in the Dharwar gold field have determined, after spending over £500,000, to give up further work. The greatest depth attained was 1260 ft. These reef deposits did not prove rich in depth, and were untreatable by the ordinary cyanide process, although cyanidation and agitation are now being tried.

Korea.—The principal gold mining operations are conducted by American companies. The Oriental Consolidated Mining Co., operating in the Unsan district has yielded over $15,000,000 since its start. The Seoul Mining Co., operating in the Suan district produced $230,200 in 1910. Among others may be mentioned the Korean Exploration Co., in Chiksan district, and Morris & McGary, in Yeng Byen district. A concession, 50×88 miles in the Kapsan district is held by the Collbran-Bostwick Development Co.

The report of the Oriental Consolidated Mining Co., for the year ending June 30, 1911, states that a net profit of $672,720 resulted. Seven mines produced 344,097 tons of ore averaging $5.19 gold per ton. Five mills using 240 stamps yielded $2.68 per ton by amalgamation and three cyanide plants treating the concentrates yielded $1.69 per ton. During the year, 23,037 ft. of development work was done, the ore reserves being 392,280 tons in June, 1911.

Mexico.—During the revolution the operating mines were seriously hampered by the interruption of the usual means of transportation, and particularly by the restrictions placed, as a war measure, upon the

distribution of dynamite. However, most of the large operating companies continued active, and all of them were in position to resume activities at the close of the war.

Chihuahua.—The Batopilas, Palmarejo, Dolores, Sierra Consolidated, Cocheño, Yoquivo and others in the western Sierra Madre were in practically continuous operation. With the operation of the railroad to Cusihuiriachic much work was done in that old camp. Santa Eulalia continued to show remarkable richness in new and deep development. Much of the mining in northern and eastern Chihuahua was suspended throughout the year. Recently some interesting rich gold discoveries were made at San Sostones, about 60 miles east of the city of Chihuahua.

The Parral camp made much progress during 1911. The Palmilla, Sierra Plata and Veta Colorada mills were put in commission and several new smaller mills were planned. The Esmeralda disclosed rich ore at depth and the Europa opened an extension on the El Tajo vein. The famous Palmilla mine is still idle, the mill finding its ore supply from the dumps and from other properties owned by the company on the Veta Colorada. The Francisco del Oro mine, near Parral, made changes in its mill plans and expects to ship soon.

El Oro District.—The mines from which most of the production came in 1911 are those of the Dos Estrellas Mining Co., Mexico Mines of El Oro, Ltd., El Oro Mining and Railway Co. and the Esperanza Mining Co. Other mines which produced a minor quantity were the Oro Nolan and the Victoria. Among those in a prospecting state are Descubridora, Reforma, Carmen, Buen Despacho, Planeta, La Lucha and San José. During 1911, the Mexican Mines of El Oro, Ltd., passed from the London-Mexico Exploration Co. to French owners, representing practically the same interests as those in the Dos Estrellas company. The only new company of any size to enter the field in 1911 was the Compañia Minera San Vidal y Anexas, which has organized with a capitalization of 600,000 pesos to undertake the exploration of the northwestern extension of the El Oro-Tlalpujahua mineral zone. This company will start work on the above-mentioned San José claim, acquired by the company within the year.

According to the report of the El Oro Mining and Railway Co., the total ore mined during the year ending June 30, 1911, was 360,504 tons as compared with 317,824 tons in 1910. The ore reserves are estimated at 484,139 tons. The southern ore body, for the first time, yielded more than the northern section, in the San Rafael vein. Changes were made in the No. 2 mill so that the sulphide ore from the lower levels could be crushed and cyanided separately. The mill yielded bullion valued at $2,389,349. Six additional agitation tanks were installed at the cyanide

plant in order to get better results from the sulphides. It was found that the same monthly profit could be obtained on the lower grade of ore by increasing the tonnage, although the percentage of recovery was somewhat lower than the previous year. The ore ran $5.99 in gold and $1.64 in silver of which 91 and 73 per cent. respectively, were extracted. The costs have been notably reduced in the mines, the mining being $1.55 per ton and an additional $.74 per ton for development. A plant for the retreatment of the sand tailings has also been completed.

Guanajuato.—The political upheaval in Mexico effected Guanajuato only to the extent of keeping out capital which otherwise might have been invested for the furtherance of established plans. The production was at least equal to that of 1910, when it was about 14,000,000 pesos. The Peregrina Mining and Milling Co. had a successful year, treating 7000 tons monthly from its own property and 6000 tons per month from the Villapando mine of the Cubo Mining and Milling Co. The Peregrina continued to pay on its preferred stock. A 200-ton cyanide plant of the Providencia Mining and Milling Co. is obtaining good results from the all-slime treatment of the Tajo de Dolores ores. In the Nueva Luz, the property of the Mineral Development Company, work was resumed, the mechanical plant being overhauled and improvements being made. It is expected to cut the mother lode early in 1912. Some of these operations are described in the *Engineering and Mining Journal* of October 7, 1911, by C. Henrich. During 1911, the Guanajuato Reduction and Mines Company treated 20,000 tons monthly, one-third of this being taken from the old Rayas and Tepeyac mines and two-thirds from the fills and dumps. The earnings of the company have been sufficient to pay interest on the $3,000,000 bond issue. The Guanajuato Consolidated Mining and Milling Co. operated only 40 of its 80 stamps during the latter part of the year, but on high-grade ore. The Pinquico Mines Co. did not treat so much high-grade ore as in 1910, its monthly output being 6000 tons. The year's production from the rich El Monte de San Nicolas mine was over 1,000,000 pesos. Shipments were regular and the lower-grade ore was treated in arrastras, the ore being taken out of the adjoining Pasadena property, where a parallel vein has been cut. The San Cayentano Mines Co. abandoned operations some time ago. The third power plant of the Guanajuato Power and Electric Co. was completed on the Angulo River and the power capacity was doubled thereby; a branch transmission line to San Luis Potosi supplied the properties at San Felipe. Arrangements were completed for the building of a transmission line from Guanajuato to supply the mines of the Angustias Mining Co. at Pozos.

Jalisco.—This state was not much disturbed by the revolution.

The Chapala Hydro-electric and Irrigation Co. completed its Puente Grande power plant which will furnish power to the Etzatlan and Hostotipaquillo mining districts. This is 15 miles from Guadalajara and has a capacity of 12,000 h. p. In the Etzatlan district the Amparo Mining Company did not make so good a showing as in 1910, although it continued to pay 3 per cent. quarterly on its capital of $2,000,000. Enlargements in the milling capacity have been made. Direct cyaniding was commenced at the El Favor plant, treating Mololoa ore, with very high extractions. This mine was purchased by the Makeever Syndicate of New York from W. M. Mathews. A big tonnage of high-grade ore was shipped to the smelter. The Casados mine is under option and much development work was done during the year. The milling of El Favor ores was suspended pending the development of the sulphides, which are free from manganese, but developing and stoping, as well as shipments to the smelter, are in progress. Much milling ore has been found in the Cinco mines. Early in 1911, the Virginia and Mexico Mine and Smelter Corporation leased its plant to the Espada Mines Co. which will ship ores to the mill.

Oaxaca.—The Natividad and San Juan mines, increased their production considerably over that of previous years. The total gross production of the state for 1911 was probably 1,200,000 pesos of which over 80 per cent. came from these two mines. The San Juan produced 400 tons of shipping ore monthly. A 10-stamp cyanide mill is planned. Other small producers were the Escuadra and Zapote, also the San Francisco mine. In the Taviche district, siliceous silver ores, largely sulph-antimonides, were shipped, containing 0.5 per cent. of silver and a small amount of gold. The Natividad mine ran two-thirds of the year but, lacking power for its 30-stamp cyanide mill, could not run full time. A new hydro-electric plant was built during the year and began operation early in 1912. The ore of the Natividad mine contains pyrite, galena and blende; 80 per cent. of the value is in gold and 20 per cent. in silver. An extraction of about 85 per cent. of the gold and 50 per cent. of the silver is secured. Development work was carried on at the Santa Gertrudis in the Villa Alta district by British investors. The mines of the Peras district shipped ores to the 20-stamp mill of the Los Reyes company.

Sonora.—Most mines in the Sonora district maintained continuous operation. The Chispas mine recently suspended work on account of labor disputes. Little progress was made in the Altar district or in any of the West Coast properties. Development operations were continued at the San Antonio in the Yaqui River district. At the El Tigre mill improvements have been made in the cyaniding of silver ores.

At the Minas Nuevas in the Alamos district, the Zambona Development Co. completed a new stamp mill, the capacity now being 100 tons per day. The Pulpito mine west of this property was closed down during the revolution but resumed work shortly. The vein has been cut by a tunnel about 450 ft. deep, where a tetrahedrite ore carrying up to 3000 oz. of silver per ton was encountered. The Sobia camp, 35 miles northeast of Alamos, which has been known as a producer of placer gold, is again attracting attention.

Nicaragua.—The total output of gold in 1911 was about $1,000,000. A great disappointment was the collapse of negotiations regarding the Bonanza gold mine, the most promising in the Pis Pis district. La Luz y Los Angeles made a good showing and the purchase of 18 1/2 per cent. indicated a total value of $1,000,000 on this property. The Bonanza and Lone Star mines are running but the Siempre Viva mine recently closed down. It is said that the oxidized ore is nearly exhausted. A regrinding plant will therefore be necessary. The Concordia mine is also in successful operation.

Peru.—The Santo Domingo mine in recent years is credited with the largest production of gold. The lack of development in this industry is probably due to the dryness of the climate. Some placer mining has been carried on in the southern part of Peru in the Department of Puno. A dredge is under construction at one place. In 1908, $647,245 worth of gold was produced in the form of ore, copper cakes, mattes and gold bars; 80 per cent. of the production was in the form of bars which remained in the country. Cyanide practice has been tried at the Santo Domingo mine. Considerable development work was done in 1911 on the placer deposits in the southern part of the country. The principal company engaged was the Inambari Gold Dredging Concessions, Ltd., which took over the placer property of the Inca Gold Development Co. in the Carabaya and Sandia districts. Another company which operated in the latter district is the Aporoma Gold-fields, Ltd., of London, which owns placers near the Hauri Hauri River. Hydraulic machinery was installed. In the same vicinity, an Argentine company operated the Poto deposits with 15 monitors. Hardly any silver mining is carried on, and practically all the output comes as a by-product from copper or lead-silver ores. In the Department of Cajamarca silver sulphides are produced. In other districts, as the Ancash, some of the ore is exported. In Lima, the output is contained almost entirely in silver-bearing copper mattes. In Junin, the Cerro de Pasco and Yauli provinces are the only silver contributors. In Arequipa, the Caylloma mines produce nearly all the concentrates and silver-bearing ores that are shipped. The production for 1908 amounted to 200,000 kg., 60 per cent. in the form of

copper cakes and mattes and 25 per cent. in ores and concentrates. In 1911 the production of silver is estimated at 6,500,000 oz.

Rhodesia (By H. F. Marriott).—The number of gold producers increased from 145 in 1910 to 163 in 1911. As in previous years, a large proportion of these were very small units working with 5 stamps and under. Only five concerns crushed regularly 5000 tons a month or more.

The total tonnage crushed during the year was slightly less than in 1910, which in turn showed a decrease on the two previous years. Notwithstanding this, however, the total yield was £200,000 in excess of 1910 and the yield per ton increased from 30.84s. to 35.66s. per ton.

Considerable activity in prospecting was observable during 1911, many of the leading Transvaal mining houses having agents in the field. Early in the year the Consolidated Gold Fields of South Africa, Ltd., floated a company with a capital of £2,000,000 to take over its assets and interests in Rhodesia. The working capital provided was £500,000. Negotiations were on foot in December last to consolidate this company's interests with those of the Rhodesian Exploration & Development Co., Ltd.

The Shamva Mines, Ltd., working a low-grade vein of great width, developed by the end of September, 1911 over 1,500,000 tons of ore of an average value of 20.27s. This valuation was made after reducing abnormally high assays. It was officially stated that the cost of developing this tonnage averaged just over 6d. per ton. Plans for the erection of a mill are being discussed, and it is believed that, owing to the width of the ore body, working costs will be well below 10s. per ton.

The output of silver fell from 217,633 oz. in 1910 to 187,641 oz. in 1911. The leading producing gold mines have been described by F. O. Maxwell.[1] Also A. E. Zealley[2] gives a description of districts.

GOLD PRODUCTION OF SOUTHERN RHODESIA TO DECEMBER 31, 1911.

	Tons Milled.	Gold, Oz.	Value.	Value per Ton.
Prior to Sept. 1, 1898	(a)	6,471	£23,456	.....
Sept. 1, 1898, to June 30, 1899	81,841	48,847	177,072	43.26s.
July 1, 1899, to June 30, 1900	104,746	57,621	208,877	39.88
July 1, 1900, to March 31, 1901	140,716	89,258	320,457	45.54
Year ending March 31, 1902	249,667	180,910	640,661	51.32
Year ending March 31, 1903	338,156	201,107	709,461	41.96
Year ending March 31, 1904	516,747	234,693	845,359	32.71
Year ending March 31, 1905	787,936	309,516	1,113,068	28.25
Year ending March 31, 1906	1,100,609	435,019	1,556,741	28.28
April 1, 1906, to December 31, 1906	1,051,908	426,333	1,531,481	29.10
Year ending December 31, 1907	1,610,875	612,052	2,178,886	27.05
Year ending December 31, 1908	1,819,230	606,962	2,526,006	27.77
Year ending December 31, 1909	1,807,771	623,388	2,623,785	29.00
Year ending December 31, 1910	1,666,223	609,956	2,568,201	30.84
Year ending December 31, 1911	(c)1,620,000	628,521	2,647,894	32.69
	12,896,436	5,070,654	£19,671,414	34.83

(a) No details available.

[1] Abstracts from Appendix to Ninth Annual Report of the Rhodesia Museum.
[2] *South African Mining Journal*, July 15, 1911.

Russia. (By J. P. Hutchins).—The gold output of 1911, allowing an estimate of 10 per cent. for gold not reported to the official assay offices, was equivalent to 1,202,952 fine ounces[1]. Of the total gold production of the Russian Empire since 1900, European Russia produced about 25 and Siberia about 75 per cent. In the total gold production of the world to date, Russia and Siberia rank third, being surpassed by the United State and Australasia, but still remaining ahead of South Africa. The gold producing region is about 4000 miles in length and is the longest auriferous district in the world, unless we consider the Cordillera in South America. Siberia has been exploiting only during the last 50 years whereas Russian dates back 100 years. Increasing interest is being shown in the Russian and Siberian industry, due largely to English, German and French capital. Unfortunately, some of the Russian regulations do not encourage foreigners in Eastern Siberia, and transportation problems are difficult. It is probable that a change in policy will occur which may attract experienced miners from the Klondike and Alaska. More gold is recovered in Russia and Siberia by drift mining than by either dredging or open cutting. In a drift mine, shafts about 14×14 ft. are sunk to bed-rock, which may be about 150 ft. These are 300 to 500 ft. apart. Drifts are then run out to the limit of the pay gravel. Close timbering is used and the ground is thawed by wood and charcoal fires when necessary. The dirt is loaded into wheelbarrows. Horses are used for hoisting. The methods are primitive and much retimbering is required. In the open-cut mines, which work from 30 to 50 ft. in depth, the costs are much reduced, and may be as little as 6 to 10 cents per cu. yd., with hand shoveling into horse carts. The cost of drifting, with labor at 50 cents per 12-hour shift and wood and timber at the rate of $2.25 per cord, would be about $2.50 per cubic yard. The Russian policy is to encourage emigration to Eastern Siberia and about $30,000,000 is expended for this purpose. It is difficult to find good hard-rock miners in Russia, or indeed an adequate supply of any kind of miners. Those that can be obtained are not chief miners, although their wages are 30 to 60 cents per day. In Eastern Siberia it is difficult to get laborers, the Russians discouraging the importation of Chinese and Koreans.

In the production of silver, the Ural district stands first. In 1911, the Verch-Isset works in Northern Ekaterinburg yielded 4937 oz.; the Kyshtim copper refinery in Western Ekaterinburg yielded 93,324 oz.; and the A. Ph. Poklieffsky-Koziel smelter shipped to Hamburg, for refining, such quantities of argentiferous copper and lead bullion as to bring the total production of silver originating in the Urals up to 263,330 troy ounces for 1911. The output of the district, similarly computed, was

[1] See article by C. W. Purrington on a later page of this volume.

238,050 oz. in 1909, while in previous years the annual output did not exceed 41,000 oz.

Salvador.—The Butter's Salvador property in Santa Rosa produced about $60,000 per month in gold during 1911. The treatment included roasting, but toward the end of the year changes were made in the treatment process, without interfering with regular production which when completed will avoid the necessity of roasting. The ore will be crushed in rolls and cyanided. At the Divisidero mine in La Union, the production was $40,000 per month in gold and silver bullion. This property is equipped with a 20-stamp mill and cyanide plant.

The Transvaal (By H. F. Marriott).—The production of gold from the mines of the Witwatersrand in 1911 showed an increase of approximately £3,000,000, with a total production for the year of £33,543,000, obtained from 23,888,000 tons crushed, an increase of 2,450,000 tons on 1910. A total of 10,500 stamps were at work at the end of 1911 and 245 tube mills, as compared with the maximum totals of 9756 and 190, respectively, in 1910; and the stamp duty was increased from 7.19 tons per stamp per day to 7.82 tons. Individual rates of duty per stamp were 14 tons per day at the Bantjes, 14.6 at the Brakpan, 13.73 at the City Deep, 17.26 at the Jupiter, 20.6 at the Roodepoort United, and 17.5 at the Simmer Deep. These figures show the tendency to increase the duty of stamp mills.

The principal additions to the list of gold producers were the Brakpan, City Deep, and Modderfontein B, which are together crushing about 130,000 tons a month for a yield of roughly £200,000. Besides these the Benoni made a beginning. The new mills of the Randfontein Central and the Crown Mines also swelled the total.

In 1911 amalgamation of the Randfontein Central and the Randfontein South, two important groups of mines under the Robinson control, took place. The amalgamated company—the name of which is the Randfontain Central—has a crushing capacity of 3,100,000 tons per annum or 258,000 tons per month; at present, the crushing is about 200,000 tons per month. The crushing plant of this huge concern consists of 1000 stamps (of which 600 are in the new Central mill), and 23 tube mills. Another amalgamation was that by which the Princess Estate absorbed the West Roodepoort Deep and the Roodepoort Central Deep.

The yield per ton milled was again reduced slightly, the average for the year being 28s. as against 28s. 6d. during 1910 and 29s. 1d. in 1909. The average working costs for the whole of the Witwatersrand showed an increase of 5d. per ton, due, chiefly to the higher cost of native labor.

The total yield of the districts outside the Witwatersrand again showed a gratifying increase, the figures constituting a record.

YEARLY PRODUCTION OF TRANSVAAL MINES.
(Chamber of Mines Returns.)

Year.	Witwatersrand District.			Outside Mines Value.	Transvaal Total.
	Tons Milled.	Value.	Value per Ton Milled. Shillings.		
1890	730,000	£1,735,491	47.4	£134,154	£1,869,645
1891	1,154,144	2,556,328	44.2	367,977	2,924,305
1892	1,979,354	4,297,610	43.4	243,461	4,541,071
1893	2,203,704	5,187,206	47.0	293,292	5,480,498
1894	2,830,885	6,963,100	49.2	704,052	7,667,152
1895	3,456,575	7,840,770	45.2	728,776	8,569,555
1896	4,011,697	7,864,341	39.2	739,480	8,603,821
1897	5,325,355	10,583,616	39.74	1,070,109	11,653,725
1898	7,331,446	15,141,376	41.3	1,099,254	16,240,630
1899	6,872,750	15,067,473	48.84	661,220	15,728,693
1900	459,018	1,510,131	65.82	.....	1,510,131
1901	412,006	1,014,687	49.25	81,364	1,096,051
1902	3,416,813	7,179,074	42.00	74,591	7,253,665
1903	6,105,016	12,146,307	39.79	442,941	12,589,248
1904	8,058,295	15,539,219	38.46	515,590	16,054,809
1905	11,160,422	19,991,658	35.82	810,416	20,802,074
1906	13,571,554	23,615,400	34.8	964,587	24,579,987
1907	15,523,299	26,421,837	34.04	981,901	27,403,738
1908	18,196,589	28,810,393	31.6	1,147,217	29,957,610
1909	20,543,759	29,900,359	29.1	1,025,429	30,925,788
1910	21,432,541	30,703,912	28.5	1,297,823	32,001,735
1911	23,888,260	33,542,967	28.08	1,448,653	34,991,620

RETURNS FROM TRANSVAAL ORES.

Year.	Recovery Per Ton	Costs per Ton.	Dividend per Ton.	Year.	Recovery per Ton.	Costs per Ton.	Dividend per Ton.
	s.	s.	s.		s.	s.	s.
1899	43.8	34.8	8.9	1907	34.0	25.0	9.0
1903*	39.8	28.8	11.0	1908	31.6	22.2	9.4
1904	38.5	29.0	9.5	1909	29.1	20.0	9.1
1905	35.8	27.2	8.5	1910	28.5	18.4	11.0
1906	34.8	26.4	8.2	1911	27.7	18.0	9.7

* 1900-1902 War period.

Silver is produced in the Transvaal only so far as it is obtained in the gold ores. It amounted in 1911 to 892,000 oz., the highest previous amount for any year having been the 830,000 oz. in 1910.

(By A. Cooper Key[1]).—The Rand gold-ore deposits exhibit a persistency and regularity, elsewhere unequaled, and developments have resulted in the blocking out of ore reserves of such magnitude as to give the mines a stability of which no other gold-field can boast.

At the end of December, 1910, the producing mines had 75,000,000 tons of ore in sight, while three developing mines in the eastern Rand had another 5,000,000 tons blocked out. The total value of these reserves is estimated at ove[redacted]10,000,000 of which £93,500,000 may be said to

[1] *Eng. and Min. Journ.*, June 10, 1911.

TRANSVAAL ORE RESERVES.

Mine.	Tons.	Tenor, Dwt.	Thickness, Inches	Mine.	Tons.	Tenor, Dwt.	Thickness, Inches.
Aurora West	339,000	5.85	36	Nourse Mines	(b) 2,142,000	6.5	.....
Banties	958,500	7.6	41	Process Estate	283,000	8.6	.....
(Pillars)	87,000	7.6	41	(Pillars)	11,000	12.6	.....
Cinderella Cons.	422,000	6.8	.....	Randfontein Central	5,828,000	7.4	.....
City Deep	2,273,000	8.2	58	Robinson Deep	(c) 1,100,000	7.2	.....
(Pillars)	141,000	8.2	58	Robinson (Main Reef) Leader and South Reef.	1,794,000	11.77	.....
City & Suburban	829,000	7.2	.....	Robinson (Main Reef)	2,202,000	4.24	.....
Cons. Langlaagte	401,000	(a) 6.37	.....	Roodeport United	617,700	5.47	38.5
Deep Level	179,300	9.25	.....	(Partial)	112,500	4.97	42
Crown Mines	6,283,000	7.6	.....	Rose Deep	3,553,500	6.1	55
Durban Rood Deep	867,550	6.7	.....	Salisbury	162,000	.....	.....
(Pillars)	194,426	7.3	.....	Simmer & Jack	2,650,000	6.3	.....
East Rand Prop.	10,250,000	6.6	.....	Simmer & Jack East	925,750	5.4	.....
Ferreira	600,000	8.1	77	Simmer Deep	1,255,000	5.0	.....
Ferreira Deep	1,779,000	9.6	.....	(Partial)	114,000	5.17	.....
Geduld Prop	445,000	9.09	37	Van Ryn	1,591,000	6.4	.....
(Probable)	102,300	10.7	41	(Partial)	203,000	6.9	.....
Geldenhuis Deep	2,521,500	6.1	46	Village Deep	1,704,000	6.1	61
Ginsberg	341,000	(a) 6.55	.....	Village Main Reef	1,775,000	.....	.....
Glencairn	1,121,000	4	.....	Vogelstruis est	223,500	.....	.....
Jubilee	37,000	.....	.....	West Rand Cons	1,125,000	5.5	46
Jupiter	1,074,000	5.8	.....	(Partial)	216,000	5.5	46
(Partial)	320,000	4.98	.....	Witwatersrand	1,233,000	5.8	.....
Knight Central	700,000	6	62	Witwatersrand Deep	1,700,000	7.16	.....
Knights Deep	(b) 1,700,000	5.5	.....	Wolhuter	760,000	6.4	.....
Lancaster West	431,000	6.76	39.4	Developing companies:			
(Pillars, etc.)	43,500	9.33	45.7	Apex	75,000	.....	.....
Luipaardsvlei, est.	398,000	.....	.....	Brakpan	1,720,000	6.56	.....
Main Reef West	430,100	7.9	48	Benoni	606,000	6.4	.....
May Cons	300,500	.....	.....	Modder B	2,788,000	7	48
Meyer & Charlton	249,000	8.6	.....				
New Goch	742,000	5.3	53				
New Heriot	485,600	8.4	46				
New Kleinfontein	1,057,400	7.47	39.57				
New Modderfontein	2,581,000	7	.....				
New Primrose	491,000	(a) 7.8	.....				
New Unified	285,000	(a) 6.8	.....				

(a) "Recovery Value." (b) July, 1910. (c) March, 1910.

be recoverable. The balance, representing the inevitable losses in mining and treatment, has been assumed as 15 per cent. The 80,000,000 tons of ore in sight have an extraction value of about 24s. per ton, and with working costs averaging 17s., there would be a margin of profit of 7s. per ton. It may, therefore, be calculated that there is a prospective profit of £28,000,000 in the ore in sight in the Rand mines. The estimate of grade compares with 28s. 6d. secured in 1910. But it must be remembered that current operations are greatly benefited from inclusion of ore from the richer mines in the neighborhood of Johannesburg, whereas the developed aggregate includes large quantities of ore of lower grade.

Three of the developing mines began to crush in 1911. These three companies show an average value of reserves of 6 3/4 dwt. and it may be pointed out that only 45 per cent. of the tonnage standing developed in the producing mines exceeds that value. Brakpan stands out as an example of sound financing, while the rapid development accomplished

on the Modder B is one of the most striking features of recent mining work on the Rand. It is only about 19 months since really serious work was started, but the company now has 3,000,000 tons of profitable ore blocked out. Owing to the flat dip in the eastern Rand, the area of blocks developed is much greater than in the central section. As values might be patchy, it was deemed advisable to avoid spending a considerable portion of the funds of the company in developing unprofitable areas in small blocks. On the Modder B the size of the blocks is 800×1000 ft., the development extending over a distance of 7000 ft. on the strike by nearly 2000 ft. on the dip. Samples taken at the periphery of these large blocks represent the value only in an approximate degree, and therefore allowances have to be made. A similarly cautious attitude is adopted in the case of the Geduld, which has been opened in much the same way. At the Modder B mine a mechanical haulage level is being driven in the foot wall of the reef across the whole property. Haulage roads driven in the foot wall will also serve as main intakes for the air for ventilation of the mine. The main haulage level has been the means of avoiding the sinking and equipping of at least two vertical shafts which, during the life of the mine, represent a large saving of capital expenditure. So far little unprofitable ore has been encountered. The company now has ore in sight equal to 100 months' supply for the mill.

The bulk of the Rand's ore reserve has a value of between 6 and 8 dwt. About 11 1/4 per cent. is above 8 dwt. and 19 per cent. is below 6 dwt. The two big Rand consolidations, the Crown Mines and the East Rand Proprietary, augmented their ore reserves enormously during 1910, the former having some 4,500,000 tons in sight and the latter had 6,500,000 tons at the end of 1909. To the contrary, some of the companies have reduced the quantity of their reserves in consequence of the exclusion of certain blocks of doubtful value. In future, the tendency will be for aggregates to increase, but for values to lessen, owing to the smaller proportion of high-grade tonnage.

West Africa.—Operations at the mines of the Gold Coast were greatly hampered during 1911 by shortage of unskilled labor. To this were ascribed the comparatively high cost of mining; Abosso $8 and Taquah $11.50, as compared with an average of $4.10 on the Rand. An association was organized to recruit savages from the outlying country, but it was deemed unwise to make any such attempt without government sanction, which was withheld. The production in 1910 was valued at £755,985 and in 1911 at £1,069,442.

Of the 1910 output, 168,511 oz. of fine gold were obtained by mining and 15,180 oz. by dredging in the Ancobra, Fura, and Offin Rivers. From the conglomerate mines 115,794 short tons were crushed, producing

GOLD PRODUCTION OF GOLD COAST COLONY AND ASHANTI, SINCE 1882.

Year.	£	Year.	£	Year.	£	Year.	£	Year.	£
1882.....	61,188	1888...	86,510	1894..	76,795	1900...	38,006	1906...	877,568
1883.....	52,435	1889...	103,200	1895..	91,497	1901...	22,186	1907...	1,154,885
1884.....	66,188	1890...	91,657	1896..	86,186	1902...	96,880	1908...	1,182,680
1885.....	89,981	1891...	88,112	1897..	84,797	1903...	254,790	1909...	955,635
1886.....	74,878	1892...	98,805	1898..	63,837	1904...	345,608	1910...	755,985
1887.....	81,168	1893...	79,099	1899..	51,299	1905...	657,330	1911...	1,069,442

62,349.905 oz. of fine gold, being an extraction of 10.77 dwt. per ton crushed; while from the quartz mines 188,366 tons were crushed, producing 106,161.118 oz. of fine gold, being an extraction of 11.27 dwt. per ton crushed. The various dredging companies treated 2,903,016 cu. yd. of alluvium with a production of 15,180 oz. of fine gold, being an extraction of 0.1 dwt. per yard.

AVERAGE PRICE OF BAR SILVER IN LONDON, 1833–1911.

(In pence per standard ounce, 0.925 fine.)

Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.
1833.	59.1875	1843	59.1875	1853	61.5000	1863	61.3750	1873	59.2500	1883	50.5625	1893	35.6250
1834.	59.9375	1844	59.5000	1854	61.5000	1864	61.3750	1874	58.3125	1884	50.6250	1894	28.9375
1835.	59.6875	1845	59.2500	1855	61.3125	1865	61.0625	1875	56.8750	1885	48.6250	1895	29.8750
1836.	60.0000	1846	59.3125	1856	61.3125	1866	61.1250	1876	52.7500	1886	45.3750	1896	30.7500
1837.	59.5625	1847	59.6875	1857	61.7500	1867	60.5625	1877	54.8125	1887	44.6250	1897	27.5625
1838.	59.5000	1848	59.5000	1858	61.3125	1868	60.5000	1878	52.5625	1888	42.8750	1898	26.4375
1839.	60.3750	1849	59.7500	1859	62.0625	1869	60.4375	1879	51.2500	1889	42.6875	1899	27.4375
1840.	60.3750	1850	60.0625	1860	61.6875	1870	60.5625	1880	52.2500	1890	47.6875	1900	28.2500
1841.	60.0625	1851	61.0000	1861	60.8125	1871	60.5000	1881	51.6875	1891	45.0625	1901	27.1875
1842.	59.4375	1852	60.5000	1862	61.4375	1872	60.3125	1882	51.6250	1892	39.8125	1902	24.0900

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1903....	21.98	22.11	22.49	23.38	24.89	24.29	24.86	25.63	26.75	27.89	27.01	25.73	24.75
1904....	26.423	26.665	26.164	24.974	25.578	25.644	26.760	26.591	26.591	26.349	26.760	27.930	26.399
1905....	27.930	28.047	26.794	26.108	26.664	26.910	27.160	27.822	28.528	28.637	29.493	29.977	27.839
1906....	30.113	30.464	29.854	29.984	30.968	30.185	30.113	30.529	31.483	32.148	32.671	32.003	30.868
1907....	31.769	31.852	31.325	30.253	30.471	30.893	31.366	31.637	31.313	28.863	27.154	25.362	30.188
1908....	25.738	25.855	25.570	25.133	24.377	24.760	24.514	23.858	23.877	23.725	22.933	22.493	24.402
1909....	23.843	23.706	23.227	23.708	24.343	24.166	23.519	23.588	23.743	23.502	23.351	24.030	23.706
1910....	24.154	23.794	23.690	24.483	24.797	24.651	25.034	24.428	24.567	25.596	25.680	25.160	24.670
1911 ...	24.865	24.081	24.324	24.595	24.583	24.486	24.286	24.082	24.209	25.594	25.649	25.349	24.592

AVERAGE PRICE OF SILVER IN NEW YORK.

(In cents per fine ounce.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1903....	47.57	47.89	48.72	50.56	54.11	52.86	53.92	55.36	58.00	60.36	58.11	55.375	53.57
1904....	57.055	57.592	56.741	54.202	55.430	55.673	58.095	57.806	57.120	57.923	58.453	60.563	57.221
1905....	60.690	61.023	58.046	56.600	57.832	58.428	58.915	60.259	61.695	62.034	63.849	64.850	60.352
1906....	65.288	66.108	64.597	64.765	66.976	65.394	65.105	65.949	67.927	69.523	70.813	69.050	66.791
1907....	68.673	68.835	67.579	65.462	65.981	67.090	68.144	68.745	67.792	62.435	58.677	54.565	65.327
1908....	55.678	56.000	55.365	54.505	52.795	53.663	53.115	51.683	51.720	51.431	49.647	48.769	52.864
1909....	51.750	51.472	50.468	51.428	52.905	52.538	51.043	51.125	51.440	50.923	50.703	52.226	51.502
1910....	52.375	51.534	51.454	53.221	53.870	53.462	54.150	52.912	53.295	55.490	55.635	54.428	53.486
1911....	53.795	52.222	52.745	53.325	53.308	53.043	52.630	52.171	52.440	53.340	55.719	54.905	53.304

Cyanidation During 1911

By Charles H. Fulton

Introduction.—The year is noteworthy for the number of large mills erected throughout the world, which embody the most recent methods as worked out by American metallurgists. Cyanidation practice as developed in Mexico in recent years is receiving world-wide application and the process is becoming standardized. The question as to how far it is necessary to go in concentration in the treatment of complex sulphide ores is receiving marked attention, and the tendency is to reduce the amount of concentration and make the process simpler, as the chemistry of the cyanidation of these ores is being better understood. The treatment of concentrate at the mills is receiving increased attention and application with good success. Efforts are being made to make cyanidation a truly continuous process and it is probable that this will succeed in the near future. Whether much will be gained by this is, however, questionable. Continuous agitation by air is now widespread and successful. Efforts to displace filtration by a system of continuous decantation can hardly make its way, and seems like a step backward. The question of grinding for cyanidation, including tube milling, etc., is referred to elsewhere in this volume, in Prof. Richards' articles.

New Milling Practice

United States.—The practice at the Goldfield Consolidated Mill, Nevada, is described in great detail by the superintendent J. W. Hutchinson.[1] Certain specific operations are again referred to in this article under agitation and treatment of concentrate. An interesting feature of the mill is the crushing department as now operated. The ore is crushed by 100—1050 lb. stamps through four mesh (0.18 in. aperture) screens, the pulp classified in two—7 ft. spitzkasten, the underflow from which passes to six, 6 ft. Trent Chilean mills crushing through 16-mesh screens. The pulp from the mills is classified in a cone, spitzkasten and Dorr classifiers, and the sands from the latter pass to five 22-ft. Gates tube mills for regrinding. The capacity of this system is 846 tons per day, equivalent to a stamp duty of 8.46 tons, the pulp all passing an 80-mesh screen and 79 per cent. of it a 200-mesh screen. The six Chilean mills are the equivalent of an addition of 40 stamps and three tube mills, or five extra tube mills, to the original equipment of 100 stamps and 10 tube mills, which had a capacity of 600 tons per day. The Chilean mills were in-

[1] *Min. and Sci. Press*, CII, 616, 652, 686, 716, 782.

stalled at a considerable saving in cost as against that of extra stamps and tube mills and the cost of crushing per ton by the *three-stage* reduction is 40.6 cents per ton as against 43.4 cents per ton for the *two stage* reduction.

The operations of the Tonopah-Extension mill[1] in the Tonopah district is described by C. T. Rice. The flow sheet of this mill has been given.[2] Recently sand concentration and slime concentration have been discontinued and the mill is now operated as a direct all slime cyaniding plant. The mill treats 145 tons of sulphide silver ore, typical of the Tonopah district, with about $13 silver and gold per ton, at a cost of $3.186 per ton. Without concentration the extraction is 93.63 per cent. of the gold and 89.74 per cent. of the silver. The practice is interesting from the fact that the sulphide ores are cyanided without concentration of either the coarse or fine material, which is a decided departure from accepted standards for this type of ore.

Another interesting feature is the employment of the Trent agitator[3] which is doing the work successfully. The agitator tanks are 24 ft. in diameter and 16 ft. deep and have a capacity equivalent to 70 tons of dry slime. The revolving air-jet arms in the bottom of the tank make 3 r. p. m. Each tank is fitted with a 5-in. centrifugal pump operating at only 400 r. p. m. and consuming 0.5 h. p. Air is supplied to the pump at 20 lb. pressure from a small compressor, at the rate of 13 cu. ft. per minute. The pulp is agitated for 48 hours. The cost of the plant was $132,000.

The Tonopah-Belmont[4] plant at Millers, Nevada, is described in detail as regards practice and costs by C. T. Rice. The mill was built in 1907 and the practice is concentration and the cyanidation of sands and slimes on sulphide ores of the Tonopah district. A feature in the mill treatment is the heating of the cyanide solutions to about 90° F. which is stated to materially increase extraction.

The mill of the Nevada Wonder[5] at Wonder, Churchill Co., Nevada, recently completed is described by P. E. Van Saun. Cyanide solutions are heated before treatment. This mill has the heaviest stamps in the United States and the plan of crushing in three stages is approximately the same as described for the Goldfield Consolidated Mill. See also Gold Milling, p 372.

The new 20 stamp mill of the Nevada Hills Mining Co. at Fairview, Nevada, is described by J. B. Fleming.[6] Details are given on p. 373. The Rawhide Queen[7] mill at Rawhide, Nev., has a capacity of 50 tons per

[1] *Eng. and Min. Journ.*, XCII, 1085.
[2] MINERAL INDUSTRY, XIX, 291.
[3] MINERAL INDUSTRY, XIX, 304.
[4] *Eng. and Min. Journ.*, XCII, 111.
[5] *Eng. and Min. Journ.*, XCI, 520.
[6] *Eng. and Min. Journ.*, XCI, 1001, and *Min. and Sci. Press*, CII, 491.
[7] *Min. and Sci. Press*, CIII, 49.

day, and is a small mill on modern lines cyaniding the ore as slime, using Dorr classifiers and thickeners, Trent agitators and Oliver filters. A feature is the use of amalgamating pans for the recovery of the metallic gold and silver in the ore. Milling at the Florence-Goldfield Mill[1] is described by C. T. Rice. The mill which treats the ore by amalgamation, concentration and cyanidation, has been briefly described.[2] The proposed North Washington Power and Reduction Co. Mill[3] near Republic, Washington, is briefly described by M. H. Joseph.

The present practice at the Desert Mill[4] of the Tonopah Mining Company at Millers, Nevada, is described by C. T. Rice. The practice while somewhat changed as regards concentration is still essentially the same as previously described.[5] J. W. Kniffin describes the Deadwood Mill[6] at Mogollen, N. M. The chief gangue minerals of the ore are quartz and calcite and the gold and silver is contained in sulphides. The average contents in gold and silver are $12 per ton. The ore is crushed in a 9×15 in. Blake crusher and in 15–1050 lb. stamps crushing in cyanide solution. The pulp is classified and then concentrated on three Wilfley tables. The tailings from the tables are classified into sand and slime in a Bayliss screw-conveyor classifier and the sand leached in tanks. The slime is agitated in three conical bottom agitator tanks by means of air. The continuous overflow from these tanks passes to a slime thickener. The thickened slime is filtered in a Portland filter. Zinc dust precipitation is used. The cyanide in the tailings is destroyed by a continuous drip of ferrous sulphate before discharging to prevent the poisoning of cattle. F. C. Landguth describes the new 150 ton cyanide mill of the Empire Mines[7] at Grass Valley, Cal. The tailings from the stamp mills are conveyed to the cyanide plant through 425 ft. of 15 in. terra cotta tile laid on a 3 per cent. grade. At the cyanide plant they are classified by the Merrill system of cone classifiers.[8] The slime is treated by a system of continuous agitation in four tanks, 10×18 ft., by compressed air and then filtered in two Oliver continuous filters, vacuum for which is furnished by an Oliver vacuum pump. The sand is leached in four 24×10 ft. tanks with sloping bottoms and a central discharge gate. Precipitation is carried out by means of zinc dust.

J. W. Swaren[9] describes the mill of the Trinity Mine in Trinity county, Cal. The ore is crushed by stamps in cyanide solution, classified by cone classifiers into sand and slime, the sand leached, the slime treated in

[1] *Eng. and Min. Journ.*, XCI, 761.
[2] MINERAL INDUSTRY, XIX, 290.
[3] *Min. and Sci. Press*, CIII, 43.
[4] *Eng. and Min. Journ.*, XCI, 1212.
[5] MINERAL INDUSTRY, XVI, 531.
[6] *Eng. and Min. Journ.*, XCII, 744.
[7] *Min. and Sci. Press*, CII, 237.
[8] MINERAL INDUSTRY, XV, 419.
[9] *Eng. and Min. Journ.*, XCI, 907; *Mining Mag.*, IV, 194.

Pachuca agitators and filtered by Oliver filters. Precipitation is carried out by the Merrill zinc dust system.

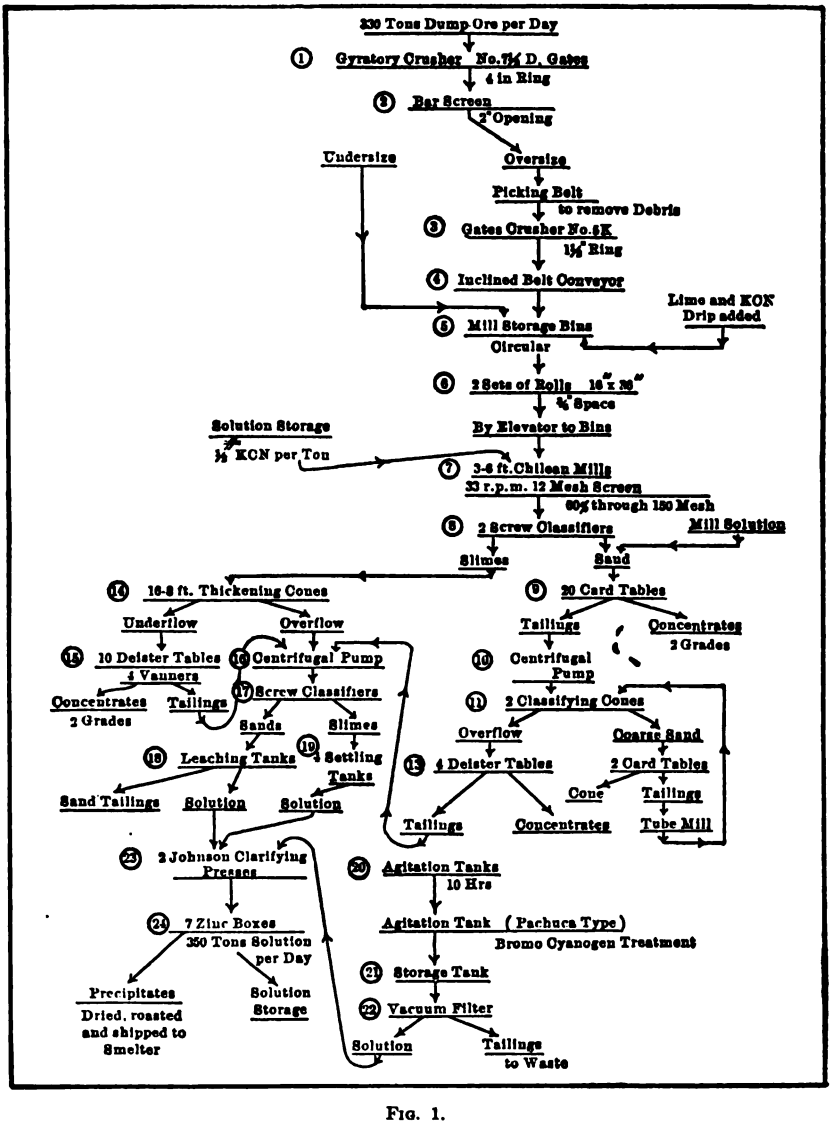

FIG. 1.

Stratton's Independence[1] Mill is described by Philip Argall. The flow sheet is shown in Fig. 1. The features of the mill are the low grade

[1] *Mining Mag.*, V, 361. *Min. and Sci. Press*, CII, 263; CIII, 684. *Eng. and Min. Journ.*, XCII, 1069. *Met. and Chem. Eng.*, IX, 192.

of material treated, viz., about $3.50 to 3.80 per ton, with a recovery of 71.5 per cent. at a cost of $1.24 for milling and $1.33 for mining and milling. It will be seen from the mill plan that extensive concentration is practised and that slime is treated by the aid of bromo-cyanogen. This method is practised at Kalgoorlie, Australia, and has been described in previous volumes of MINERAL INDUSTRY.[1] Two grades of concentrates are made on the tables—a high grade product containing 5 to 7 oz. gold per ton, which is shipped to smelters, and a lower grade product, 0.5 to 0.75 oz., which is treated by chlorination mills. The total concentrate removed by the sand and slime concentration amounts to 3 per cent. of the weight of the ore and contains 40 per cent. of the value of the gold and silver in the ore. The Portland mill at Cripple Creek is treating similar material, but at present no details of the mill treatment are available. It is stated that in place of bromo-cyanogen, potassium bromate is used as the extra chemical.

The Moore-Clancy process for complex ores has been rather widely discussed in the technical press during the year, but no new data as to actual working of the process is as yet available.

The Ajax Mill[2] at Victor, Colo., started operations late in November with a rated capacity of 200 tons per day. The ore is crushed by gyratory crushers, rolls and tube mills, so that the pulp will all pass a 150-mesh screen. In the tube mills the ore is crushed in solution. The pulp passes to the five specially designed air agitators provided with the Clancy electrolyzing baskets. From the agitation tanks the pulp passes to the loading tanks of the Moore filter. Precipitation is carried on by means of zinc dust. The electrolyzing[3] apparatus of the Clancy process comprises essentially a circular beam within the agitation tank, from which are hung the anodes and cathodes. The anodes are fused magnetite, copper coated and the cathodes are iron plates. The anodes are cylindrical in form, closed at the lower end, 46 in. long, and have a surface of 2.2 sq. ft. The cathodes are 48 in. long and 18 in. wide and 3/32 in. thick. The electrodes are suspended about 2 ft. below the surface of the pulp. The chemistry of the process has been described,[4] but has been somewhat modified recently.

The chlorination process[5] has practically been abandoned in the Cripple Creek district. The Portland Company has decided to change its mill at Colorado City from chlorination to cyanidation and the United States Reduction and Refining Co. has announced that the Golden Cycle

[1] MINERAL INDUSTRY, XVII, 444.
[2] *Eng. and Min. Journ.*, XCII, 1077.
Consult also: *Eng. and Min. Journ.*, XCI, 355, 649, 748, 1064. *Min. and Sci. Press*, CII, 270, 436, and CIII, 599. *Met. and Chem. Eng.*, IX, 336.
[3] *Eng. and Min. Journ.*, XCI, 904.
[4] MINERAL INDUSTRY, XIX, 311.
[5] *Eng. and Min. Journ.*, XCI, 195, 1172.

Cyanide Mill will hereafter handle ores formerly shipped to its plant on contract.

Charles A. Chase describes the present practice at the Liberty Bell Mill,[1] near Telluride, Colo., giving valuable detail. New features, differing from the old practice previously described, are the employment of Richards' classifiers and Bunker Hill screens in the concentrating department, no concentration after regrinding, and the employment of Hendrix agitators in series for continuous agitation of slime and Dorr continuous settlers. Amalgamation in cyanide solution after the batteries and tube mills is still practised, but is more expensive in labor and material and requires more skill than when amalgamating in water. The consumption of copper due to solution is considerable. The plates are kept rather wet and any quicksilver drip is caught in a trap. Muntz metal plates were tried, but proved unsatisfactory on account of the hard glassy surface which they assumed. The practice is kept up for local reasons, such as coarse gold and spotted nature of the ore.

The cyanidation practice of the Homestake Mills,[2] S. Dak., is described by C. C. Semple on information furnished by A. J. Clark and W. J. Sharwood. The article goes into valuable detail concerning the classification of pulp and cyanide treatment for which reference is made to the original paper.

The mill of the Trojan Mining Co.[3] at Portland, in the Black Hills, S. Dak., is described by Jesse Simmons. The mill is representative of Black Hills practice for the Cambrian siliceous ores of the district and is similar to the Mogul Mill at Pluma, S. Dak. The ore is crushed by a Gates crusher, rolls and Trent Chilean mills, in cyanide solution, and classified into sand and slime by Dorr classifiers. Dorr slime thickeners and a special form of air agitation tank is used. The capacity of the mill is 300 tons per day in two units of 150 tons each.

The new dry crushing plant of the Wasp No. 2 Mining Co.,[4] near Deadwood, S. Dak., is described by Jesse Simmons. This mill replaces one destroyed by fire and previously described.[5] The capacity of the mill is approximately 500 tons per day. The ore contains $2.50 per ton in gold, of which about $2 is recovered as bullion at a total cost of mining and milling of $1.02. The ore is mined by open cut work.

Canada.—The Dome Mines Mill[6] is the first large mill to be erected in the Porcupine district. The flow sheet of the mill, now in the course of erection, follows, Fig. 2.

[1] *Bul.* 60, A. I. M. E., 981.
[2] *Eng. and Min. Journ.*, XCI, 719.
[3] *Eng and Min. Journ.*, XCII, 357.
[4] *Eng. and Min. Journ.*, XCII, 214.
[5] MINERAL INDUSTRY, X, 342.
[6] *Min. and Sci. Press*, CII, 694; *Canadian Min. Journ.*

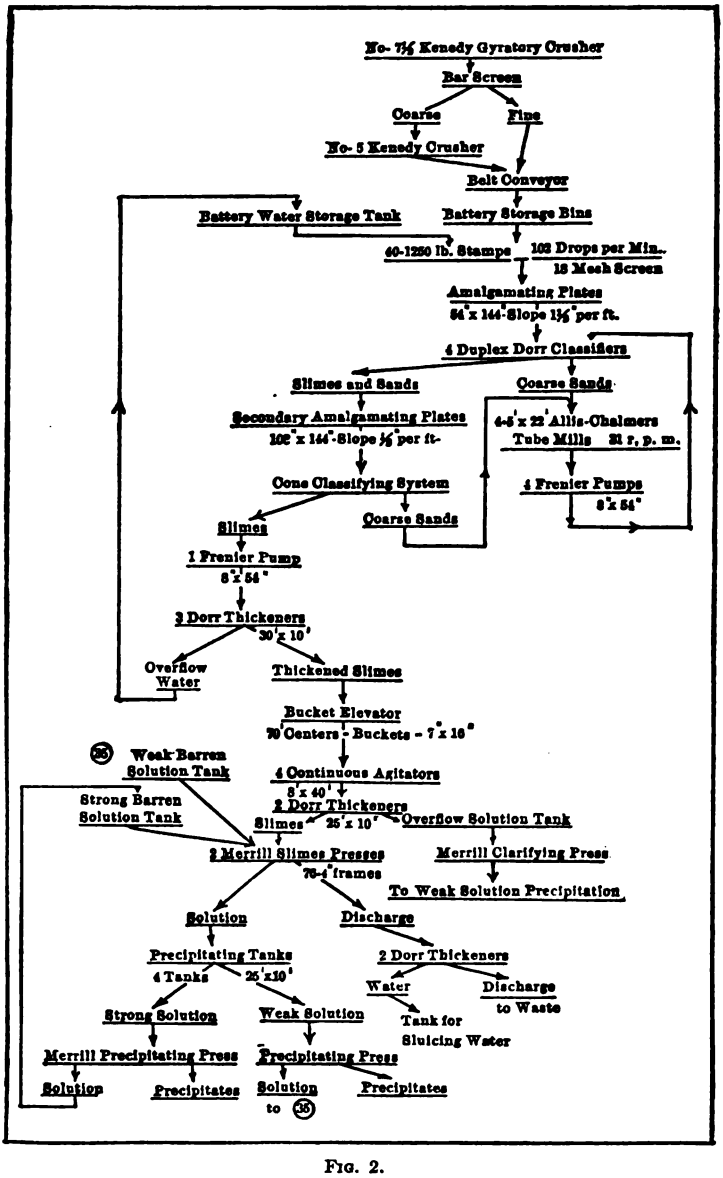
No. 7½ Kenedy Gyratory Crusher
Bar Screen
Coarse
Fine
No. 5 Kenedy Crusher
Belt Conveyer
Battery Water Storage Tank
Battery Storage Bins
40-1250 lb. Stamps
102 Drops per Min.
18 Mesh Screen
Amalgamating Plates
84"x 144"-Slope 1½" per ft.
4 Duplex Dorr Classifiers
Slimes and Sands
Coarse Sands
Secondary Amalgamating Plates
102"x 144"-Slope ½" per ft.
4-5'x 22' Allis-Chalmers
Tube Mills 21 r. p. m.
Cone Classifying System
4 Frenier Pumps
8"x 54"
Coarse Sands
Slimes
1 Frenier Pump
8"x 54"
3 Dorr Thickeners
30'x 10'
Overflow Water
Thickened Slimes
Bucket Elevator
70 Centers - Buckets - 7"x 16"
(26) Weak Barren Solution Tank
4 Continuous Agitators
8'x 40'
Strong Barren Solution Tank
2 Dorr Thickeners
Slimes
25'x 10'
Overflow Solution Tank
2 Merrill Slimes Presses
76-4" frames
Merrill Clarifying Press
To Weak Solution Precipitation
Solution
Discharge
Precipitating Tanks
4 Tanks
25'x10'
2 Dorr Thickeners
Water
Discharge to Waste
Strong Solution
Weak Solution
Tank for Sluicing Water
Merrill Precipitating Press
Precipitating Press
Solution
Precipitates
Solution to (26)
Precipitates

FIG. 2.

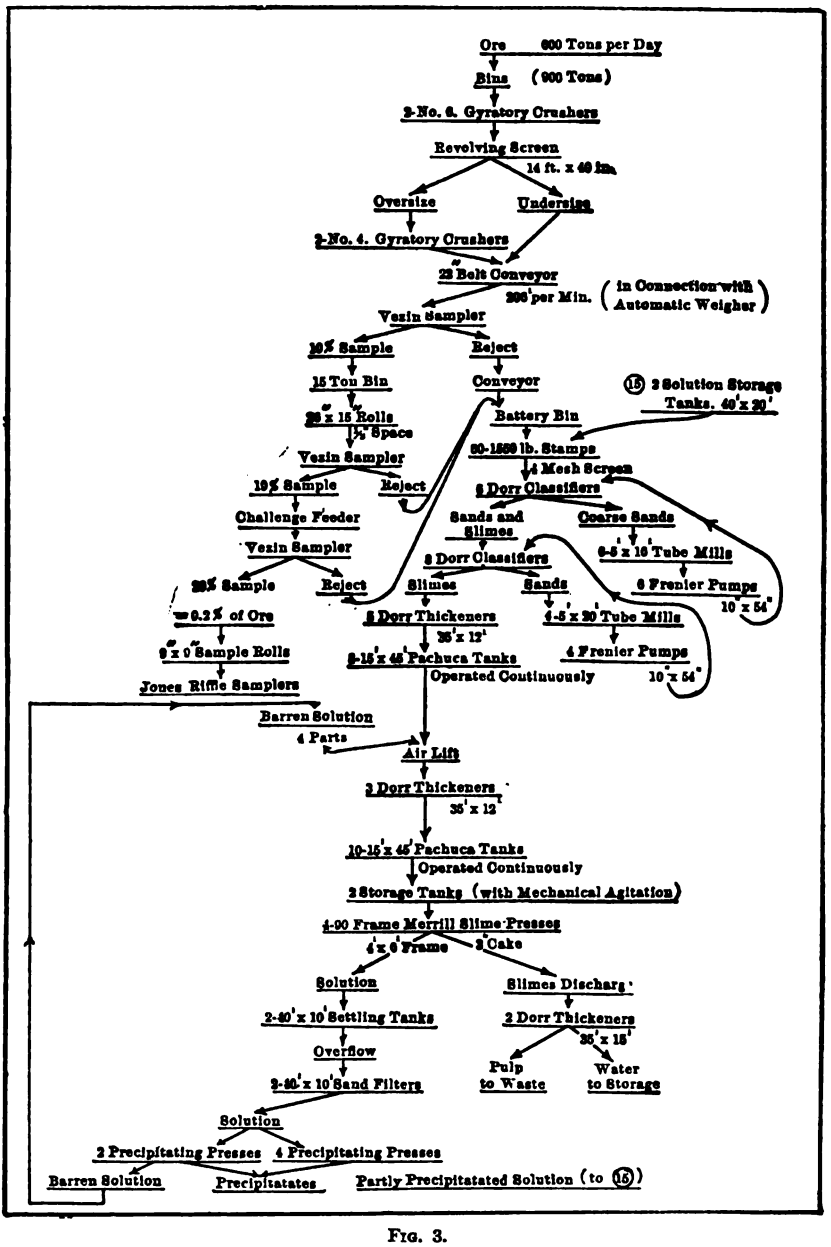
Ore 900 Tons per Day
Bins (900 Tons)
2-No. 6. Gyratory Crushers
Revolving Screen
14 ft. x 40 in.
Oversize
Undersize
2-No. 4. Gyratory Crushers
23" Belt Conveyor
200' per Min.
(in Connection with Automatic Weigher)
Vezin Sampler
10% Sample
Reject
15 Ton Bin
Conveyor
(15) 2 Solution Storage Tanks, 40' x 20'
24" x 16" Rolls
½" Space
Battery Bin
Vezin Sampler
20-1250 lb. Stamps
19% Sample
Reject
4 Mesh Screen
6 Dorr Classifiers
Challenge Feeder
Sands and Slimes
Coarse Sands
Vezin Sampler
6-5 x 16' Tube Mills
20% Sample
Reject
Slimes
Sands
6 Frenier Pumps 10" x 54"
6.2% of Ore
5 Dorr Thickeners 35' x 12'
4-5 x 20' Tube Mills
8" x 0" Sample Rolls
8-15' x 45' Pachuca Tanks
4 Frenier Pumps 10" x 54"
Operated Continuously
Jones Rifle Samplers
Barren Solution
4 Parts
Air Lift
3 Dorr Thickeners 35' x 12'
10-15' x 45' Pachuca Tanks
Operated Continuously
2 Storage Tanks (with Mechanical Agitation)
4-90 Frame Merrill Slime Presses
4' x 6' Frame
Cake
Solution
Slimes Discharge
2-40' x 10' Settling Tanks
2 Dorr Thickeners 35' x 15'
Overflow
Pulp to Waste
Water to Storage
2-40' x 10' Sand Filters
Solution
2 Precipitating Presses
4 Precipitating Presses
Barren Solution
Precipitates
Partly Precipitated Solution (to (15))

FIG. 3.

R. E. Hore[1] states that there are now 14 plants operating in the Cobalt district on the low-grade silver ores which contain from 27 to 40 oz. of silver per ton. The design of these plants is manifold, but practically all of them concentrate, or concentrate and cyanide.

Mexico.—The Santa Gertrudis[2] Mill of the Compania Beneficiadora de Pachuca, about 3 miles from Pachuca is described by W. Weston. This important mill was designed after competitive tests on different types of mill apparatus and marks an advance in the treatment of silver ores. A noteworthy feature is the absence of concentration, the series crushing in a double set of tube mills, the employment of continuous agitation in two stages, and the precipitation of the solution in two circuits, one for complete precipitation and the other for partial precipitation, in order to economize in zinc. The flow sheet is shown in Fig. 3.

P. E. Van Saun[3] describes the 75-ton mill of the Bostwick Mining Co., at Guazapares, Chihuahua. The mill treats silver-gold ore by a system of continuous agitation and continuous decantation, the latter serving in place of filtration. The continuous decantation is carried out in thickening tanks with a continuous overflow.

H. C. Field describes the practice at the Pinguico Mill[3] of the Guanajuato Development Co. at Guanajuato. The method of treatment is crushing by stamps, concentration, regrinding, sand leaching, agitation of slime in mechanical arm agitators provided with compressed air pipes, and filtration of slime in Butter's filters. F. K. Boyd describes the 10-stamp mill of the Atlixtac Mining Co.[4] at Atlixtac in Jalisco. It is a typical fine crushing plant with Pachuca tanks and Butter's filter.

The Chispas cyanide plant of the Minas Pedrazinni Gold and Silver Mining Co.,[5] in the Arizpe district of Sonora is described by E. L. Dufourcq. The ore containing silver with a small amount of gold is crushed by light stamps in water, and the pulp passed over Wilfley tables to separate out concentrate. The tailings flow to a mixing tank where they are mixed with old tailings, then elevated by a bucket elevator to Dorr classifiers which classify out the sand product that goes to the tube mill. The tube mill product passes to a second Dorr classifier, the sand from which are returned to the tube mill, while the overflow joins that from the first classifier, the combined pulp passing to a Dorr thickener. The slime is treated in a system of modified Pachuca tanks and in a Burt filter. Zinc dust precipitation is used.

H. B. Kaeding[6] discusses experiments in the treatment of Nicaraguan gold ores by cyanidation.

[1] *Eng. and Min. Journ.*, XCI, 717.
[2] *Eng. and Min. Journ.*, XCII, 109.
[3] *Mexican Min. Journ.*, XIII, No. 1., p. 24.
[4] *Mexican Min. Journ.*, XIII, No. 3, p. 20.
[5] *School of Mines Quart.*, XXXIII, 18.
[6] T. A. I. M. E., *Bul.* 60, 1029.

S. F. Shaw gives additional detail[1] from that previously described on the practice at the Montezuma Mines[2] of Costa Rica.

The question of the disposal of cyanide tailings into streams, thereby doing damage to agricultural interests, has been recently discussed in Mexico. For interesting comments, reference is made to the original paper.[3]

South Africa.—J. G. Hamilton[4] gives the following milling statistics for the Transvaal for 1910. Tons milled, 21,974,358; stamps at work, 9775; tube mills at work, 175; duty per stamp, 7.1; total value of production, £32,000,735; value per ton, 28s 8d; cost per ton, 17s 8d; profit per ton, 8s 8d. F. H. Hatch, discusses the evolution of Rand cyanide practice[5] and states that the Butter's filter is now in successful operation at the Crown Mines, and will be installed at the Bantjes, the New Modderfontein, the Robinson, and the Modder B mines. H. A. White discusses the development of gold extraction methods on the Witwaters Rand.[6] F. H. Hatch describes the progress in milling on the Rand[7] and states that the conservatism of Rand metallurgists toward new methods is justified in view of the enormous interests involved, which makes it necessary to demonstrate fully the value of new ideas on the spot before they can be adopted. Mr. C. O. Schmitt discusses in detail proposed future economies in Rand reduction plants[8] in order to compensate for increased costs due to greater depth of mining. The City Deep Mill and one-half of the Randfontein Central Mill started operations early in the year. R. W. Schumacher discusses the successful introduction of the Butter's vacuum filter on the Rand.[9] The Merrill zinc dust precipitation has been ordered for the New Modderfontein and the Modderfontein B Mills. It possesses a distinct advantage for Rand plants as it gives opportunity for a complete clean-up.

The new mill of the Roodepoort United Main Reef Gold Mining Company[10] has 100–1900-lb. stamps, one 5 ft. 9 in. × 16 ft. 5 in. tube mill for each 20 stamps and seven shaking amalgamating tables after each tube mill. The plates are all in one plate house. Sand is treated by the usual leaching and slime by the decantation process. Precipitation is by means of zinc thread. The capacity of the mill is estimated at 1500 tons per day. The new Randfontein Central Mill[11] of the Randfontein Estates Gold Mining Company, has 600–1650-lb. stamps and 16 tube

[1] MINERAL INDUSTRY, XVIII, 298.
[2] *Min. and Sci. Press*, CII, 179.
[3] *Met. and Chem. Eng.*, IX, 588.
[4] Annual Address, Pres. Transvaal Chamber of Mines, *Min. Mag.*, IV, 180.
[5] *Eng. and Min. Journ.*, XCII, 165
[6] *S. A. Journ. of Sci.*, VII, 96, and *Journ. Chem. Met. and Min. Soc. S. A.*, XI, 448.
[7] *Eng. and Min. Journ.*, XCII, 265.
[8] *Journ. Chem., Met. and Min. Soc. S. A.*, XI, 464, 563 and XII, 16.
[9] *Min. and Sci. Press*, CII, 659.
[10] *Journ. Chem. Met. and Min. Soc. S. A.*, XI, 440.
[11] *Min. and Sci. Press*, CII, 332 and 490.

mills. All amalgamation plates are placed in the gold recovery house. Sand is treated in the usual manner and slime by decantation. Precipitation is by zinc thread. This is the largest mill under one roof in the world.

The new mill of the Modderfontein B Company[1] now in the course of erection embodies the most recent cyanide practice as employed on the

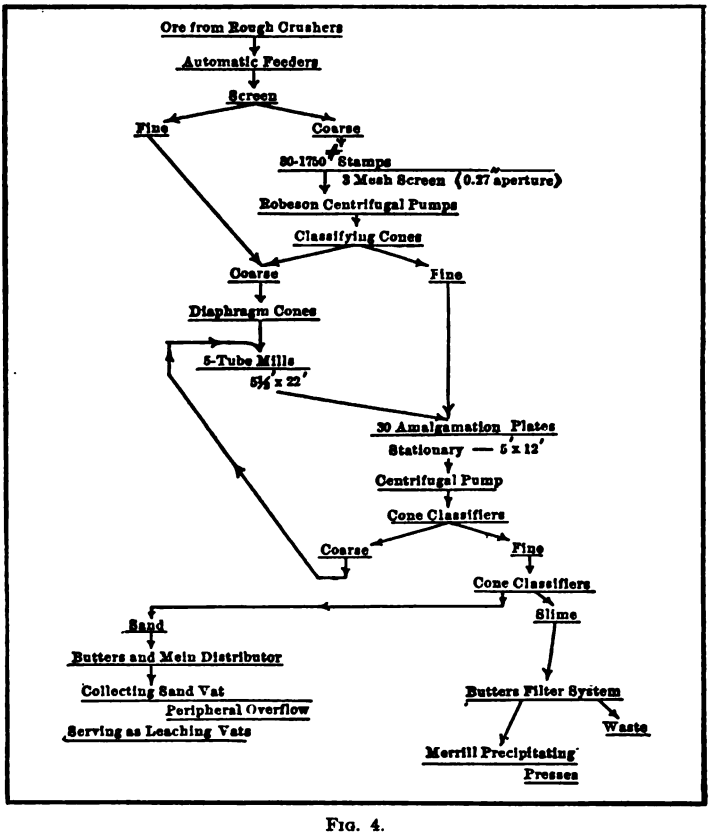

FIG. 4.

Rand, using Butter's vacuum filters, zinc dust precipitation and the Homestake method of smelting precipitates.[2] Fig. 4 is the flow sheet.

The mill of the Brakpan Mines, Ltd.,[3] is described by C. A. Tupper. The mill is situated on the far East Rand, 21 miles from Johannesburg. It is the most modern mill on the Rand and employs in addition to the common Rand practice, vacuum filtration after air-agitation in Pachuca

[1] *Min. Mag.*, IV, 85.
[2] MINERAL INDUSTRY, XVI, 551.
[3] *Min. and Sci. Press*, CIII, 796 and *Eng. and Min. Journ.*, XCII, 753.

tanks and zinc dust precipitation. The mill went into operation in June and has been in continuous operation since. During October, with 100 stamps and six tube mills in operation for 30 days, 42,089 tons of ore were milled, a recovery of $6.29 being made per ton. The ore has a value

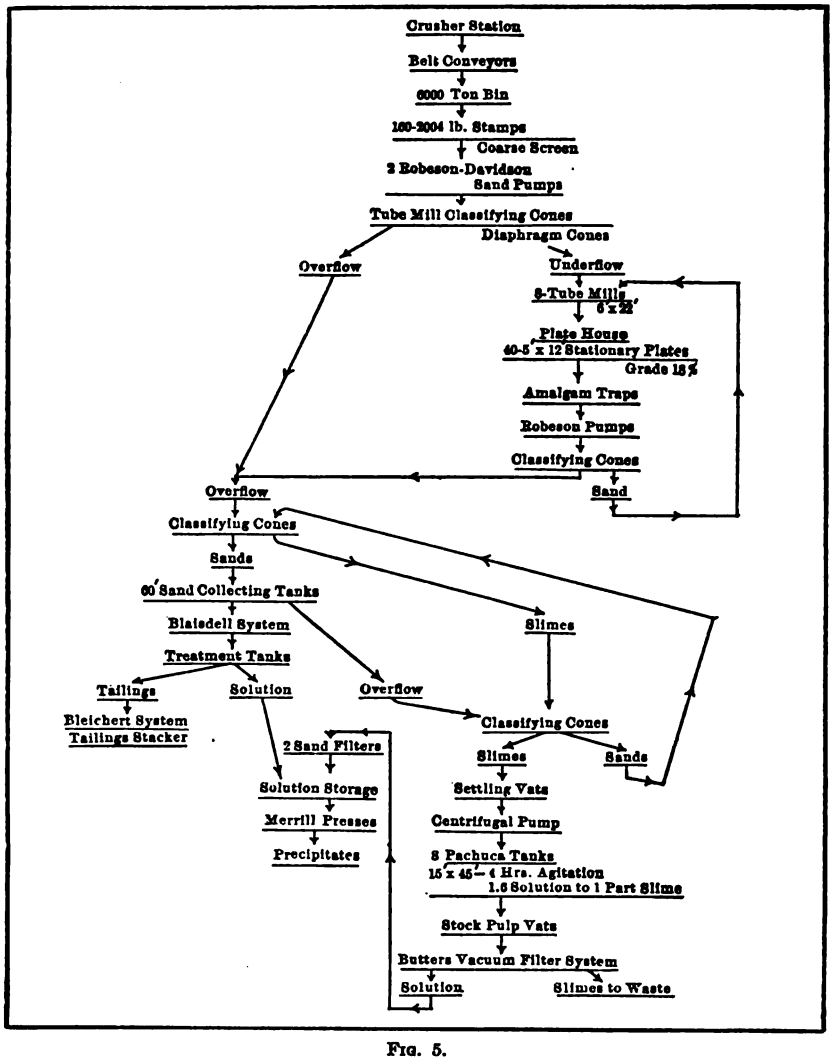

FIG. 5.

between $6.50 and $7.00 per ton. The total operating costs were $3.98 per ton. Mr. F. L. Bosqui had much to do with the design of the mill. The flow sheet is represented in Fig. 5.

The Benoni cyanide plant[1] on the Rand, in the course of erection, employs the Arbuckle-Way cyanide process. The chief point in the design is that the sand and slime is mixed for treatment with solution, then separated by Stadler classifiers and the solution withdrawn from the slime by Arbuckle cones and from the sand by Caldecott sand filters. The same type of machines are also used in thickening the sand and slime after tube milling and before cyaniding. The ore is crushed by 55–2000-lb. stamps through a 3-mesh screen. The pulp passes to four Stadler classifiers from which the underflow passes to four tube mills. The overflow joins the discharge from the tube mills and is pumped to another set of four Stadler classifiers. The underflow from this set is returned to the tube mills and the overflow goes to 28 shaking amalgamating tables. The pulp from these passes to primary and secondary Stadler classifiers. The underflow from the secondary classifier is the sand and is thickened on a Caldecott table while the overflow from both sets is the slime and is thickened in three Arbuckle cones.

The thickened sand and slime is combined and sent to four mixing agitators, and mixed with cyanide solution. The pulp is then sent to 32 Way-Arbuckle treatment agitators, in which continuous agitation by means of compressed air is carried out. The total time of agitation is only 4 to 5 hours. Subsequently the pulp goes to another set of Stadler classifiers, again making a separation of sand and slime, the first of which has solution removed on Caldecott tables, while the second is thickened and decanted in Arbuckle cones. Then again, the sand and slime, together with weak barren solution, is mixed in agitators and again classified and treated as before, the sand and slime being now discharged while the solution is precipitated. It is stated that this process has been tried and abandoned at the East Proprietary Mines. The process essentially seems to be a continuous agitation-decantation cyanide process, which presents novel points as regards the apparatus in which it is carried out. It is difficult to see where it presents any advantage over continuous agitation and filtration, rather the contrary, for it has been shown a number of times that continuous decantation systems cannot effect the results of filtration.

Australia and New Zealand.—The milling practice at the Komata Reefs Mine[2] on the Hauraki peninsula of the North Island, New Zealand, is described by S. D. McMiken. The ore containing about five parts of silver to one of gold is unoxidized and is crushed by 20 stamps weighing 1100 lb. through 4-mesh screens in water. The usual classification and tube milling is practised and after tube milling the pulp is passed

[1] *Mining Mag.*, IV, 307, 383.
[2] *Min. and Sci. Press*, CIII, 4.

over Muntz metal plates for amalgamation. Slime treatment is carried out in Dorr thickeners, a modified Pachuca tank system, and a modified Moore filter. Sand is treated by percolation. The description includes interesting details of operation.

E. G. Banks[1] describes milling and treatment at the Waihi Mine, New Zealand.

The flow sheet of the Great Boulder Perseverance Mill[2] shows a typical roasting cyanidation plant of the Kalgoorlie district.

The plant for the treatment of old tailings at the Oroya Brownhill Mine[3] at Kalgoorlie is described by T. B. Stevens and W. R. Degenhardt. The plant was erected in 1908 when the dump contained about 500,000 tons of filter-press tailings. About 92 per cent. was material that had been finely ground and treated by the bromo-cyanogen process, while 8 per cent. represented the residues from roasted concentrate treatment. The fineness of the material is such that 97.5 per cent. passes a 100-mesh screen. The tailings contain $2 in gold per ton of which 50 cents is present as dissolved gold. Of the remaining $1.50, 50 cents only can be profitably extracted by re-treatment, unless regrinding were resorted to, which, however, is not feasible on account of cost. The extraction is therefore approximately $1 per ton. The capacity of the plant is 730 tons per 24 hours, 97 cents per ton is recovered at a cost of 50 cents per ton. The tailings in the form of a conical heap, are loaded on a belt conveyor, which transports them to a bin from which another conveyor carries the material to a vortex mixer where it is mixed with barren cyanide solution containing 0.04 per cent. KCN. The mixer is 5 ft. in diameter and 4.5 ft. deep, and has rapidly revolving cast-iron plates driven from a vertical central shaft. Heavy foreign bodies, as pieces of iron, accumulate in the bottom and are periodically removed. From the first mixer the pulp passes by launder and over a screen, to remove wood and débris, to a second mixer from which it passes to six agitation vats 17.5 ft. $\times$ 9 ft., of galvanized iron, the sides of which are corrugated. Agitation is effected by means of four angle-iron arms revolving at 12.5 r. p. m. One and one-half pounds of lime per ton of tailings is added in the agitators. The pulp is agitated for 8 hours when the cyanide strength of solution will have dropped to 0.02 per cent. The s. g. of the pulp is maintained at 1.61. From the agitators the pulp is transferred by centrifugal pump to two 17.5 ft. $\times$ 9 ft. storage vats from which it passes to a vacuum filter of the Butter's type containing sixty-eight 9 ft. $\times$ 5.25 ft. frames, set 6.5 in. center to center. The filter puts through 16.5 charges of 44 tons each per 24 hours. The solution from the

[1] *Mining and Engineering Review*, March 6, 1911, 240. *Journ. Chem. Met. and Min. Soc. S. A.*, XI, 636.
[2] *Min. Mag.*, IV, 389.
[3] *Min. Mag.*, IV, 460, also *Monthly Journ. Chamber of Mines, West Australia*, March, 1911.

the vacuum filter passes to gold solution tank in which the cyanide strength is brought to 0.05 per cent. KCN to assist precipitation and prevent formation of magnesium hydroxide on the zinc. From the gold solution tank, the solution passes to a "constant head" tank, thence through three filter-presses for clarifying and thence to seven zinc boxes, 522 cu. ft. capacity, for precipitation.

M. W. von Bernewitz[1] discusses roasting at Kalgoorlie. At the Associated Northern Blocks, ores containing 3 to 4 per cent. sulphur are roasted, after crushing through 27-mesh screens, in six Merton furnaces. These have three 30 ×6.5 ft. hearths and a finishing floor. The furnaces require 2.5 h. p. each for the mechanical part. The tonnage per furnace is from 100 to 120 per day, depending on the sulphur and graphite contents of the ore. The fuel consumption is 14 per cent. of the weight of the ore charged and the cost of roasting is 66 cents per ton of roasted ore. On heavy sulphide ores two Edwards duplex furnaces are used. The hearth area is 11.75×107 ft. There are three fire-boxes, each 6×2 ft., along the length of the hearth. The rabbles make 4 r. p. m. The capacity is about 95 tons per furnace per day and the cost of roasting is 60 to 80 cents per ton. The fuel consumption is 11 per cent. of the ore roasted.

Agitation and Slime Treatment

Agitation.—Mr. A. W. Warwick discusses air agitation and slime treatment[2] and states in reference to the Pachuca agitator that it is an imperfect device and that the agitation is not efficient in so far as in a 12-hour agitation, he concludes that an individual particle of pulp circulates 12 times through the air lift and tank, taking about a minute on each trip to pass through the air lift where it receives agitation in the solution, while it takes an hour to fall in the tank and again reach the air lift, and during this fall in the thick pulp, no agitation practically takes place, since the pulp particles and the solution have about the same velocity, relative to each other. In other words, the ore particle falls in its enveloping mantle of solution, which is chemically inactive, owing to saturation, and no new solution of precious metal takes place.

He states also that the shape of the Pachuca tanks makes them unduly expensive as regards construction and housing. Mr. A. Grothe[3] replies to these criticisms successfully. In an experimental tank 54 in. × 18 in., with an air lift proportioned as in large tanks, the air lift discharge was collected at intervals and examined with the following results. It is con-

[1] *Min. and Sci. Press*, CII, 660.
[2] *Min. World*, XXXIV, 243.
[3] *Mex. Min. Journ.*, XIII, No. 1, 26. See also *Chem. and Met. Eng.*, IX, 233. *Min. and Sci. Press*, CIII, 80.

ceded that the best agitation is obtained when the velocity of the falling pulp particles differs, as much as possible, from that of the descending solution in the tank proper. The velocity of the solution is designated by V and that of the ore particle by V'. The value of V' is derived by calculating from the ratio of solids in the air lift discharge to that of solids in the pulp in the tank. The following data were obtained.

Grams of Solids per Liter of Pulp.	Ratio of Solution to Solids.	Value of V', V = Velocity of Solution.	Sizing Test of Pulp of Air Lift Discharge			Velocity of Different Sized Particles			Velocity of the Silver in the Ore.
			+100	+200	−200	+100	+200	−200	
			%	%	%				
294	3 : 1	1.45V	14.2	39.4	46.4	2.23V	2.05V	1.07V	1.53V
415	2 : 1	1.21V	12.5	32.4	55.1	1.64V	1.40V	1.06V	1.27V
524	1.5 : 1	1.11V	11.3	30.3	57.4	1.36V	1.20V	1.03V	1.12V

The last column shows the velocity of the silver in all the particles referred to the solution velocity and was obtained by assays of the pulp as fed to the tank and the assay of the pulp discharged by the air lift. No data as regards the actual displacement of ore particles in reference to the solution is as yet available. From data furnished by H. Adams[1] it may be calculated that in tanks used at Natividad, Mexico, 35 × 12 ft. with a lift pipe 15 in. diameter, amount of air and pressure not specified, the tanks operated continuously in a series of six, that the speed of pulp in the lift pipe, which may be assumed to be the speed of solution in the lift, is 235 ft. per minute. This figure gives a speed of 2.55 ft. per minute for the solution descending in the tank. The ratio of solution to solids was 2.3 to 1. Using the value of $V' = 1.21V$ as above, it may be calculated that the displacement of the ore particle as regards the solution in a 35-ft. fall (it is actually about 32 ft.) will be 6.10 ft., plainly showing that solution and ore particle are not stationary in reference to each other.

Air-lift agitation of slimes pulp is reviewed by R. Allen[2] who gives a description of large diameter air-lift vats in use on slime at the Luipaards Vlei Estates Co. in the Transvaal. E. J. Laschinger[3] discusses air agitation from the standpoint of power consumed and states that the power consumption usually stated is much underestimated. J. E. R. Adendorff[4] discusses and describes the experiments on accumulated ore slime on the Rand by means of an experimental Pachuca tank.

J. W. Swaren gives historical notes on the air-lift agitator.[5] He states that in 1880 the Tonite Powder Company employed an air-lift agitator to wash gun cotton in their factory in Contra Costa County, California.

[1] *Min. World*, XXXIV, 697.
[2] *Journ. Chem., Met. and Min. Soc. S. A.*, XI, 422, 563; also *Min. and Sci. Press*, CII, 628.
[3] *Ibid.*, 501.
[4] *Journ. Chem., Met. and Min. Soc. S. A.*, XII, 6.
[5] *Min. and Sci. Press*, CIII, 409.

Name of Mine.	No. of Agitators.	Size in Feet.	Type of Agitator.	Capacity per Charge, Tons of Dry Slime.	Method of Agitation.	Speed r.p.m.	Time hrs.	H.P.	% KCN in Solution.	Cost per Ton.
Associated	6	17×6	A.-Z.	22	Propeller.	60	1½	12	.065	$0.24
Associated Northern	5	22×6	Ordinary arm agitator.	32	Arms.	6	4–8	2½	.08	0.26
Great Boulder Perseverance.	24	20×4.5	Ordinary arm agitator.	15	Arms.	6	2	2½	.07	0.22
Great Boulder Proprietary.	18	22×5 18×5	Ordinary arm agitator.	40	Arms.	6	3–8	2½	.08	.....
Kalgurlie	20	12.5× 6	Air tank, no central lift.	4.5	Air.	.....	3½	2	.07	0.21
South Kalgurlie	5	21×9	Ordinary arm agitator.	40	Arms.	14	4–6	2½	.06	.....
Hainault	3	22×6	Ordinary arm agitator.	40	Frenier pump.	7	4–5	.....	.....	.....
Ivanhoe	11	20×8	Ordinary arm agitator.	53	Arms.	.....	18	3	.08	0.23
Lake View and Star	6	20×6.5	Ordinary arm agitator.	44	Arms.	11	4–6	2	.05	.....
Oroya Links	6	17.5×9	Ordinary arm agitator.	50	Arms.	13	7–8	2½	.02	.....

The central air lift was introduced into the agitator by W. L. Oliver. These agitators were also used previous to 1904 at the North Star Mines, Grass Valley, Calif., by E. L. Oliver in connection with an electrolytic cyanide process, in which the pulp was agitated with solution and then electrolytically precipitated on amalgamated copper cathodes in the agitation tank. This use of the agitator of the Pachuca type thus antedates their use of them by F. C. Brown and S. D. McMiken at the Komata-Reefs Mine in New Zealand, but there can be no question that Brown and McMiken first practically worked out in detail the conditions under which successful air agitation can be carried out on a commercial scale. M. W. von Bernewitz describes methods of slime agitation at Kalgoorlie, Australia. A new agitator, successfully used at the Associated Mine, is known as the A.-Z. agitator. This is similar to the Hendryx agitator[2] and has been briefly described.[3] The accompanying table gives data on agitation at the various Kalgoorlie Mines.

[1] *Min. and Sci. Press*, CII, 749.
[2] MINERAL INDUSTRY, XVI, 544.
[3] MINERAL INDUSTRY, XIX, 306.

H. T. Zembler[1] describes the "air-pocket" agitator. This is a tank that has a cone bottom with a 50 degree slope. The total outside height is 20 ft. and the diameter is 16 ft. The cone ends in a 6-in. discharge pipe at right angles to the vertical axis of the cone and the air is introduced into this discharge pipe about 8 to 12 in. from the apex of the cone instead of into the cone proper just above the apex. This method of introducing the air is stated to prevent the settlement of heavy particles on the side of the cone and to give excellent agitation. L. P. Hills[2] has patented an air-lift agitator, with cone-shaped bottom, which resembles the Pachuca tank system in having a central air lift. But surrounding the air lift is a second tube, adjustable as regards vertical position, which forms an annular tube with the air-lift pipe, terminating below the air-lift discharge. At the top of the air lift is placed a cylindrical baffle plate about one-half of the diameter of the tank and of some depth. When it is desired to cease pulp agitation, pulp feed to the tank is stopped and the adjustable tube is lowered by a hand wheel until the lower edge touches the cone, when solution will begin to be drawn down the annular tube and discharged by the lift tube within the baffle plate. Presently the solids settle and solution only is circulated. When it is desired to again start pulp agitation, the adjustable tube is raised a little at a time, and the pulp again circulates.

W. C. Patterson[3] has patented a continuous agitator that may be operated singly or in series. It has a cone-shaped bottom and a central lift, the solution and slime being drawn off at the top by a pipe which feeds a centrifugal pump, the discharge from which is fed to the bottom of the lift. An agitation tank employing a centrifugal pump for agitation is described by an anonymous writer.[4]

A. L. Glover describes a simple laboratory agitator.[5]

Continuous agitation of slimes pulp in Pachuca tanks has made rapid progress during the year and is being adopted at many plants, as is evident from description under the caption of new milling practice. It is a step toward the goal of making the cyanide process a continuous operation, which has been achieved at some plants where continuous agitation, followed by the Dorr thickener and the Oliver continuous filter have been installed. In place of the Oliver filter or similar devices, a continuous decantation by means of an apparatus like the Dorr thickener has been proposed, which question is referred to again below. Huntington Adams describes the process of continuous agitation as practised at the Natividad Mine, Ixtlan, Oaxaca, Mexico.[6] There are six

[1] *Journ. Chem. Met. and Min. Soc. S. A.*, XI, 578 and *Pacific Miner*, Feb., 1911, p. 58.
[2] *Met. and Chem. Eng.*, IX, 423.
[3] *Met. and Chem. Eng.*, IX, 660.
[4] *Mex. Min. Journ.*, XIII, No. 4, p. 34.
[5] *Eng. and Min. Journ.*, XCII, 247.
[6] T A. I. M. E., *Bul.* 56, 595. *Eng. and Min. Journ.*, XCII, 670, 696. *Met. and Chem. Eng.*, IX, 336.

tanks 35 ft. high and 12 ft. in diameter, and the central air-lift tube is 15 in. in diameter. The tanks are arranged for continuous agitation by means of the following device. Wooden boxes, 7 in. wide, 10 in. long and 6 in. deep, inside measurements, are fixed against the central air-lift tubes, with their tops flush with those of the tubes, and 4-in. pipes placed horizontally, pass from the bottom of each box to the next tank in the series. By-pass pipes, fitted with valves, join each pipe connection with the next in the series, and a 4-in. drop from tank to tank is made in the by-passes. Since the level of the inflow pipe in each tank is 2 in. below the tops of the air-lift tube and of the overflow box connected with the outflow pipe, but is slightly above the pulp level in the tank, no part of the pulp entering can pass out of the tank without having first gravitated to the bottom, and risen through the air-lift tube, thoroughly mixed with the whole content of the tank. On the tops of the overflow boxes are sliding iron covers, which open at right angles to the direction of the overflowing stream. The regulation of the flow from tank to tank is done entirely by means of these covers, and the valves in the by-pass pipes are used only when it is desired to pass the pulp by one or more of the tanks. The boxes are sufficiently large so that the covers need only be opened an inch or two to give the required flow. The openings which sample the pulp stream are rectangular, with their long axis parallel to the radial overflow at those points, and while theoretically this is not as correct a shape to sample the stream as would be a sector of a circular ring, in practice it has been found to offer no difficulty, while it is simpler to install and to keep in order in the pulp stream. As the last tank of the series does not discharge to a thickening tank, but directly to the pulp tank of a Moore filter, where an intermittent feed is important, it is arranged with two pairs of sliding doors on the air-lift tube, so as to permit agitation at various levels. The pulp is drawn off intermittently to the filter through the bottom discharge opening of the tank, either by hydrostatic pressure in the tank, if full enough, or by 4-in. Butter's centrifugal slime pumps. All the tanks retain their bottom discharge connections to the centrifugal pumps which lift to the filter, so that whenever it is necessary to empty any one of them, they may be cut out of the series to prevent pulp entering, and after sufficient agitation, may be discharged to the filter. The continuous system of agitation has removed completely former difficulty in starting tanks with pulp containing fine sulphides, which were of common occurrence when intermittent agitation was practised. Other methods for connecting Pachuca tanks for continuous agitation have been described.[1]

At the Goldfield Consolidated[2] the pulp is agitated in ten 45×15 ft.

[1] MINERAL INDUSTRY, XVIII, 348; XIX, 342.
[2] *Min. and Sci. Press*, CII, 652, J. W. Hutchinson.

Pachuca tanks connected in series. The s. g. of the pulp is 1.35, representing a dilution of 1.5 solution to 1 of dry slime. The tanks are connected by 8-in. pipes after the manner in use at the Esperanza Mine.[1] The results were at first unfavorable, due to a lack of sufficient air, but the practice is now successful since an additional compressor has been installed. Approximately 80 per cent. of the value of the ore after concentration is dissolved in the agitators. Approximately 75 cu. ft. of free air per minute is required for each tank, holding 85 dry tons of slime, equivalent to 6 h. p. per tank. The accompanying table gives extraction figures for the different tanks.

Gravity of Pulp.	Oz. Gold per Ton in Dry Pulp.	Oz. Gold per Ton in Solution.	No. of Tank.
1.35	0.31	0.21	2
1.34	0.26	0.256	4
1.335	0.20	0.30	6
1.33	0.14	0.328	8
1.32	0.10	0.348	10

Assay of dry slime entering tanks, 0.62 oz. per ton; assay of filter tailings, 0.085 oz. per ton. The lighter gravity in the last tanks is accounted for by the addition to tanks 4 and 8 of solution containing the chemicals, extra cyanide and lead acetate.

The subject of continuous agitation and decantation is discussed at some length by J. E. Rothwell,[2] J. V. N. Dorr[3] and D. F. Irwin.[4] Mr. Rothwell's proposed flow sheets are shown in the diagrams, Figs. 6 and 7. Mr. Rothwell states that the present practice in the treatment of gold and silver ores by the cyanide process when using the agitation method requires the mixture of a barren cyanide solution with the ore at its highest value; then by the continued agitation of this mixture dissolution of the gold and silver is accomplished and the metal transferred to the solution, the rate of dissolution proceeding more slowly as the point of chemical equilibrium is approached. Thus it is necessary to agitate the mixture for such length of time as required to obtain the highest economical transfer of the metal into the solution, and in some cases it means a re-treatment of the charge after the removal of the first solution, in order to obtain a satisfactory recovery of the metal content of the ore.

"The ideal and logical method of continuous treatment and operation is the reverse of this method, that is, the solvent of lowest metal value and highest in free cyanide should be applied to the ore after the greater value in metal has been dissolved. The solution is then separated from

[1] MINERAL INDSTRY, XVIII, 348; XIX, 342.
[2] *Met. and Chem. Eng.*, IX, 373 and 439.
[3] *Met. and Chem. Eng.*, IX, 436.
[4] *Min. and Sci. Press*, CIII, 102. See also L. B. Eames, *Mex. Min. Journ.*, Sept., 1910. *Mex. Min. Journ.*, XII, No. 4, p. 24, Bruno Mierisch.

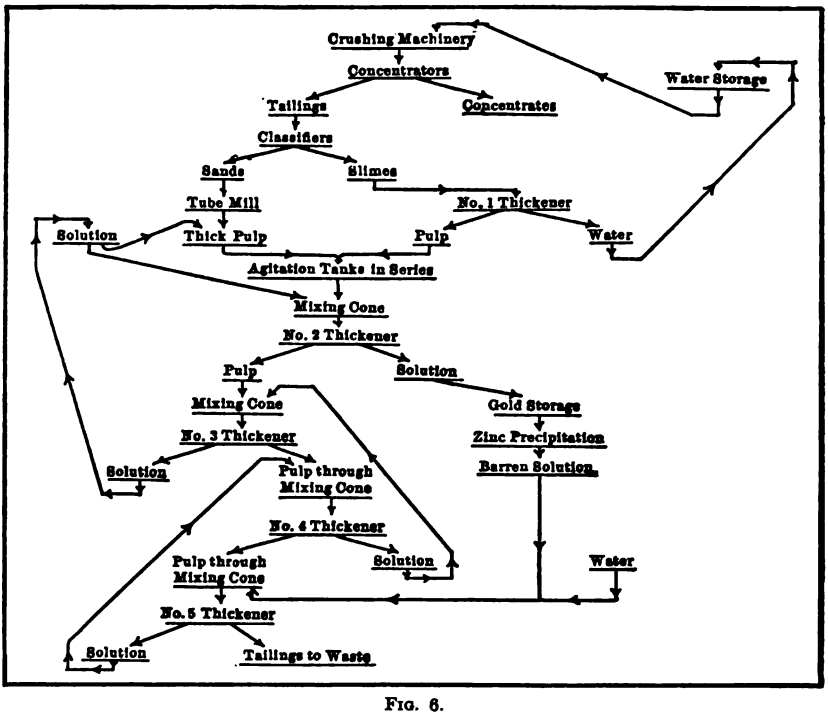
Crushing Machinery
Concentrators
Water Storage
Tailings
Concentrates
Classifiers
Sands
Slimes
Tube Mill
No. 1 Thickener
Solution
Thick Pulp
Pulp
Water
Agitation Tanks in Series
Mixing Cone
No. 2 Thickener
Pulp
Solution
Mixing Cone
Gold Storage
No. 3 Thickener
Zinc Precipitation
Solution
Pulp through Mixing Cone
Barren Solution
No. 4 Thickener
Pulp through Mixing Cone
Solution
Water
No. 5 Thickener
Solution
Tailings to Waste

FIG. 6.

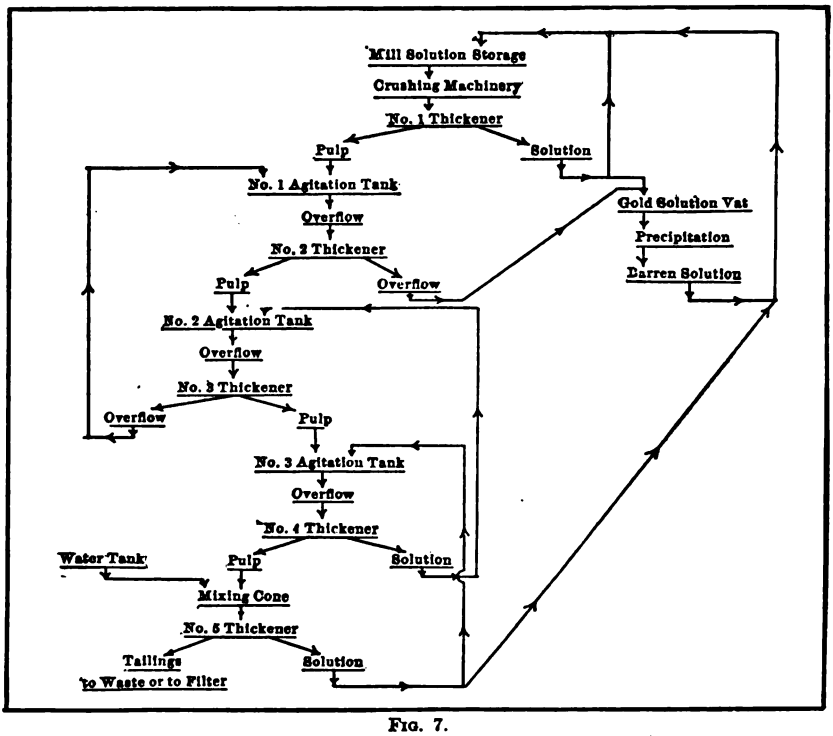
Mill Solution Storage
Crushing Machinery
No. 1 Thickener
Pulp
Solution
No. 1 Agitation Tank
Gold Solution Vat
Overflow
Precipitation
No. 2 Thickener
Barren Solution
Pulp
Overflow
No. 2 Agitation Tank
Overflow
No. 3 Thickener
Overflow
Pulp
No. 3 Agitation Tank
Overflow
No. 4 Thickener
Water Tank
Pulp
Solution
Mixing Cone
No. 5 Thickener
Tailings to Waste or to Filter
Solution

FIG. 7.

the ore and applied to ore of higher value, and so back up the line of treatment, until the highest grade of solution is in contact with the ore at its maximum value, after which it is separated and passed to the precipitation department where the metal content is precipitated and recovered and the barren solution returned to the mill system to go through the work of gathering up value again in the same sequence of operation." It will be noted that this develops the "counter current of solution to the ore" idea which has much to commend it and should not be mistaken for the method of continuous decantation in which a rising current of solution is caused to flow against settling pulp, in the effort to displace gold-bearing solution, as for instance, is practised in the Adair-Usher process.[1] In Mr. Rothwell's suggestion the pulp from the batteries is agitated with solution high in metal content and then thickened, the thickened pulp again agitated with solution which has been in contact with lower-grade pulp, again thickened, and the agitation repeated, always with solution from the overflow of thickeners situated below the agitator in the series of apparatus.

This method has been practised in the old decantation process, in some instances, and in principle represents nothing very new, but with the advent of improved thickening tanks that can discharge pulp of certain kinds (quartz slime) with moisture contents as low as 38 per cent., takes on new interest. Whether a filter can be satisfactorily replaced by thickening tanks remains to be seen. As already stated, the addition of a continuous filter, like the Oliver, to the series of apparatus gives a continuous process and it is possible that one or two extra thickeners plus the Oliver filter will equal the results of a Butter's or Moore, at a somewhat lower cost.

There is one objection to the "counter current of solution" idea that may be urged. While no doubt the rate of dissolution of the precious metals decreases with the increase of the solubility product ($KAu(CN)_2$), in the solution and the decrease of the solvent (KCN), this factor is not the most vital in most instances. The most vital factor seems to be the accumulation in the solution of reducing agents and precipitating agents such as alkaline sulphides, and these increase in proportion to the amount of ore the solution has been in contact with and the time of contact. In the scheme of operation under discussion, solutions, perhaps high in deleterious substances will come in contact with pulp of the highest metal content where the precipitating effect will be most pronounced in spite of considerable free cyanide.

Sand Treatment.—A discharge door[2] for cyanide sand tanks used in West Australia, is suitable for conditions such as obtain in regions where

[1] MINERAL INDUSTRY, XVII, 449.
[2] *Eng. and Min. Journ.*, XCII, 105.

water is scarce. A description of a discharge gate for sand tanks[1] in use at the North Star Mills, California, is given by an anonymous author. The Argall discharge gate[2] for sand leaching tanks is described by an anonymous writer. C. T. Rice[3] describes the method of sluicing out sand tanks in use at North Star Mills in California. A Butter's and Mein distributor which is connected by means of a tight hopper to a 20-lb. water pressure is used. H. T. Durant[4] discusses the upward leaching of sand. It is suggested that alternate upward and downward leaching is the best practice in many instances, since it affords better aeration, and the upward leaching with fresh solution brings oxygen in contact with sand in the bottom of the tank. J. G. Banks[5] describes difficulties arising during the treatment of sand. An air lift[6] for the raising of sand is fully described by A. R. Mill. The lift is in two stages and raises the sands 24.25 ft. Amount of air is 47 cu. ft. per minute raising 320 lb. of sand and 800 lb. of water, equivalent to 23.83 lb. of sand and water per cubic foot of air.

Slime Treatment.—The classification of pulp prior to cyanidation is discussed by E. H. Johnson[7] and others. Cooper Shapley[8] discusses a method of returning pulp from a tube mill to the classifiers. J. G. Young[9] in a very interesting paper on slime filtration which sets forth the result of experimental work also gives the accompanying classification of slime filters.

CLASSIFICATION OF SLIME FILTERS

I. Suction filters, or filters in which a vacuum is used to accelerate filtration.

A. Appliances using a thin slime cake and practically continuous in their action. (Oliver and Ridgway filters.)

B. Appliances using a thick slime cake and intermittent in their action. (Moore and Butter's filters.)

II. Pressure filters, or filters in which hydrostatic head, compressed air or pumps are used in order to secure greater pressures than are possible with a vacuum pump.

These filters are intermittent in their action.

C. Ordinary filter presses.

D. Sluicing filter presses (Merrill filter press).

E. Filtering chambers or cylinders; filters in which the filtering basket is inclosed in a cylinder. (Burt, Kelley, and Sweetland filter presses.)

III. Centrifugal filters, or filters in which centrifugal force is used to separate solution from slime.

These filters are continuous in action.

[1] *Eng. and Min. Journ.*, XCI, 951.
[2] *Eng. and Min. Journ.*, XCII, 343.
[3] *Eng. and Min. Journ.*, XCI, 209.
[4] *Eng. and Min. Journ.*, XCI, 417.
[5] *Eng. and Min. Journ.*, XCII, 149, 199.
[6] *Journ. Chem., Met. and Min. Soc. S. A.*, XI, 398 and *Min. Mag.*, IV, 141; also, E. A. Rix, *Min. and Sci. Press*, CI, 505.
[7] *Journ. Chem., Met. and Min. Soc. S. A.*, XI, 320, 373.
[8] *Eng. and Min. Journ.*, XCI, 258.
[9] T. A. I. M. E. *Bul.* 59, 839.

The following general conclusions are drawn from the experimental work. Reference should be made to the original article for detail.

1. The proportion of clayey material in ores which are to be subjected to "all-sliming" and filtration should be maintained at a minimum.

2. The slime pulp should be as free as possible from sands coarser than a 150-mesh screen, and as large a proportion of the pulp as possible should consist of material passing a 200-mesh screen.

3. The slime pulp before filtration should be settled to as thick a consistency as possible consistent with ready handling by pumps and in pipes.

4. The temperature of the slime pulp should be maintained between 20° and 30° C. or higher.

5. The temperature of the wash water and the pulp should be the same.

6. Vacuum pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be filtered, as much fine sand (limited as stated above) should be crowded into the pulp as it will carry without undue settling and clogging.

8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering machines.

9. With slime containing a large proportion of colloid or clayey material pressures greater than those obtainable with vacuum apparatus are of questionable advantage.

10. With slime containing a large proportion of clayey material the vacuum filters should be used.

11. With slime containing a small proportion of clayey material and much fine sand both vacuum filters and pressure filters could be used with perhaps equally good results.

12. With slime containing much coarse and fine sand the chamber filters with air agitation and high pressures would perhaps give the best results.

13. Of the vacuum filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

It is probable that many metallurgists will hardly agree fully with the last statement.

E. Burt[1] describes the new Burt filter in use at El Oro, Mexico. The filter consists of a steel cylinder 40 ft. long and 42 in. in diameter. The head is closed by a casting bolted to the shell and at the discharge end there is a special door operated and held in place by six heavy toggles

[1] *Eng. and Min. Journ.*, XCI, 169.

and levers, operated by hydraulic power. Through this door the tailings are discharged. The cylinder is set in a horizontal position and when in operation revolves continuously on tires. The inside circumference of the cylinder is lined with double filter cloths, twenty in number, each 26 in. wide and 10 ft. long, allowing five to the circle and four to the length. Wooden gratings are placed in the space formed by the double cloths at certain intervals, and provide for the filtered solution. They are held in place by curved 1.5 in. angle irons which are staggered throughout the interior and fastened to studs in the shell at the junction of two cloths in such a position and pitch that they act as a spiral for discharging the slime cake. The filtering area is connected to the outside of the cylinder by special nipples for the discharge of filtered solution. The solution flows from these to the cement floor from where it goes to the sump. Pulp and solution are charged to the filter by means of a pipe, with the requisite valve connections, which passes through a stuffing box in the hollow trunnion in the head. In operating, the slime pulp is charged until gages show a pressure of 35 lb., when it is shut off and air pressure turned on until the gage registers about 40 lb., at which point it is maintained until all the solids in the pulp are in the cake and the moisture in this is down to 20 per cent. Then air is replaced by solution and wash water. The cake is removed by opening the door, and continuing to revolve the cylinder. By adding a little water during discharging, the cylinder is readily cleaned. The discharged cake contains 25 per cent. moister, At the El Oro plant six Burt filters handle 1000 tons of slime per day at a cost of 4 cents per ton.

Additional patents have been granted to E. G. Spilsbury[1] on applications of the silica sponge filter [2] which has been described. G. Johnston[3] has invented a continuous vacuum filter of the drum type. B. W. Taylor has been granted a patent on a continuous filter of the traveling belt type.[4] T. H. Neikirk has invented a revolving filter[5] employing centrifugal force to form the slime cake. It operates continuously. H. Wallace has invented the Eclipse filter, in which the slime is fed to a horizontal traveling belt, the solution being squeezed out of the slime by pressure rollers. J. E. Rothwell[6] has invented a filtering or thickening tank, in the bottom of which is a traveling belt which discharges settled material to a sump at one end while the solution, practically clear, rises between a series of baffie plates and overflows at the other end. T. B. Stevens and W. R. Degenhardt[7] describe the process of slime filtration by vacuum

[1] *Met. and Chem. Eng.*, IX, 422.
[2] MINERAL INDUSTRY, XIX. 306.
[3] *Met. and Chem. Eng.*, IX, 282.
[4] *Met. and Chem. Eng.*, IX, 660.
[5] *Met. and Chem. Eng.*, IX, 609.
[6] *Eng. and Min. Journ.*, XCI, 667.
[7] *Met. and Chem. Eng.*, IX, 394. *Eng. and Min. Journ.*, XCI, 999, 1091. *Eng. and Min. Journ.*, XCII, 433.

filters of the Butter's type as practised in Western Australia, giving much interesting detail, and describing construction applicable to local conditions. J. H. Oates[1] urges the use of an air jet in the bottom of the storage tanks of the Butter's filter in place of the usual mechanical stirring device in order to save power when electric power is purchased on basis of peak load. H. F. Julian[2] discusses the relative merits of slime filters and the methods of their operation, and losses inherent in the different types.

The Butter's filter at the Crown Mines Ltd.[3] South Africa, treats 900 tons of dry slime per day. 2700 tons of pulp are treated in Pachuca tanks, the dilution being 2 to 1. This pulp with additional wash solution passes to stock tanks and thence to two Butter's filters each fitted with 150 leaves, and acting as an independent unit. Each unit is served by a 14×14 in. Gould vacuum pump. One 10-in. Robeson-Davidson slime pump handles pulp and solution to and from the filter boxes and the stock vats serving the boxes alternately. The cycle of operations is as follows: filling and forming cake, 45 minutes; transferring and washing, 70 minutes; discharging, 20 minutes; total time of cycle 2 hours and 15 minutes. The thickness of cake is 1.25 in. and 100 tons are treated per cycle. The plant is eminently successful.

D. L. H. Forbes[4] describes the process of slimes thickening at the El Tigre mill, at Yzabal, Sonora, Mexico. The thickeners are of the Dorr type, but in place of the short straight blades of angle irons acting as rakes on the arms, four balanced spiral blades hung from a vertical shaft are used to impel the thickened slime continuously toward the discharge at the center. M. R. Lamb[5] has patented a slime thickener. The thickener has revolving arms hung from a hollow vertical shaft which passes through the bottom of the tank by a stuffing box. It is revolved by a worm gear. The hollow shaft is connected above with a pump, and just below the rake arms has a number of openings through which the settled slime enters and is withdrawn by the suction of the pump. The clear solution overflows at the periphery. It will be noted that this device differs from the common thickener in being driven from below, and by having the thickened slime drawn off through the shaft at the top. A. C. Nahl[6] describes a thickening tank, termed an intermittent slime decanter. It consists of a cone-shaped vat with a peripheral overflow and a slime-discharge gate at the bottom. A two-armed revolving scraper works against the sides of

[1] *Mex. Min. Journ.*, XIII, No. 3, p. 22.
[2] *Min. Mag.*, V, 284.
[3] *Journ. Chem. Met. and Min. Soc. S. A.*, XI, 389; *Met. and Chem. Eng.*, IX, 300. *Min. Mag.*, IV, 112.
[4] *Eng. and Min. Journ.*, XCII, 691.
[5] *Eng. and Min. Journ.*, XCII, 1190.
[6] *Eng. and Min. Journ.*, XCI, 312.

the cone and scrapes the settled slime to the bottom. The slime is discharged intermittently. C. Semple[1] describes a small filter press used to de-water slime samples in the assay office. This device originated at the Homestake assay office. A filter of a different type for the same purpose is described by an anonymous author[2] and is used at the Tonopah Belmont Mill. E. G. Acheson[3] describes experiments in defloculation which are of interest in slime treatment. H. A. White[4] states that in taking slime samples for the purpose of assay to determine whether any undissolved gold remains, it is necessary to insure the stopping of any further solution by the cyanide in the sample, between the taking of the sample and the drying. This is accomplished by putting the sample as soon as obtained into a bucket containing water made pink by potassium permanganate. This completely destroys any cyanide present and prevents further solution of gold.

Treatment of Concentrate

The treatment of concentrate by cyanidation is being practised to an increasing extent, and the most noteworthy plant to go into operation is that of the Alaska Treadwell Company which is described below. At some plants[5] there is a tendency to do away with concentration and to cyanide the sulphides (in refractory ores) with the slime after grinding it very fine as has been done for some time at the Grand Junction Mine, Waihi, New Zealand. The practice of the Waihi Gold Min. Co. of New Zealand, in treating concentrates by fine grinding, agitation and filter pressing, is described by E. G. Banks.[6] R. Lindsay[7] describes the treatment of the so-called mill concentrate at the Geldenhuis Deep Ltd. on the Rand.

W. P. Lass[8] describes the cyanidation of concentrate at the Treadwell Mines, Alaska, with much detail. The 100-ton cyanide plant treats the 90 tons of concentrate from 360 vanners obtained from the 5-stamp mills (900 stamps), which crush about 5000 tons of ore daily. The concentrate has the following composition: Fe, 40 per cent.; SiO_2, 11 per cent.; S, 43 per cent.; and contain from 2.5 to 4 oz. gold and 0. 75 oz. silver per ton. The screen analysis is: on 20 mesh, 0.44 per cent.; on 40 mesh, 8.23 per cent; on 60 mesh 10.96 per cent.; on 80 mesh, 12.49 per cent.; on a 100 mesh, 10.38 per cent.; on a 120 mesh, 13.37 per cent.; on a 150 mesh, 7.69 per cent.; through a 150 mesh, 36.46 per cent. The

[1] *Eng. and Min. Journ.*, XCI, 407.
[2] *Eng. and Min. Journ.*, XCII, 931.
[3] *Eng. and Min. Journ.*, XCII, 1272.
[4] *Journ. Chem., Met. and Min. Soc. S. A.*, XII, 89.
[5] *Min. and Sci. Press*, CII, 223. *Eng. and Min. Journ.*, XCII, 1023.
[6] *Eng. and Min. Journ.*, XCII, 248. *Min. and Sci. Press*, CII, 142.
[7] *Journ. Chem. Met. and Min. Soc. S. A.*, XII, 2.
[8] T. A. I M E., *Bul.* 62, 183. *Min. and Sci. Press*, CIII, 514.

gold in the concentrate is essentially free and is mechanically held in the pyrite. The flow sheet, Fig. 8, outlines the method of treatment. The essential features are the grinding of all the concentrate through 200-mesh screen in lime water, and a 3-hour preliminary agitation in Pachuca

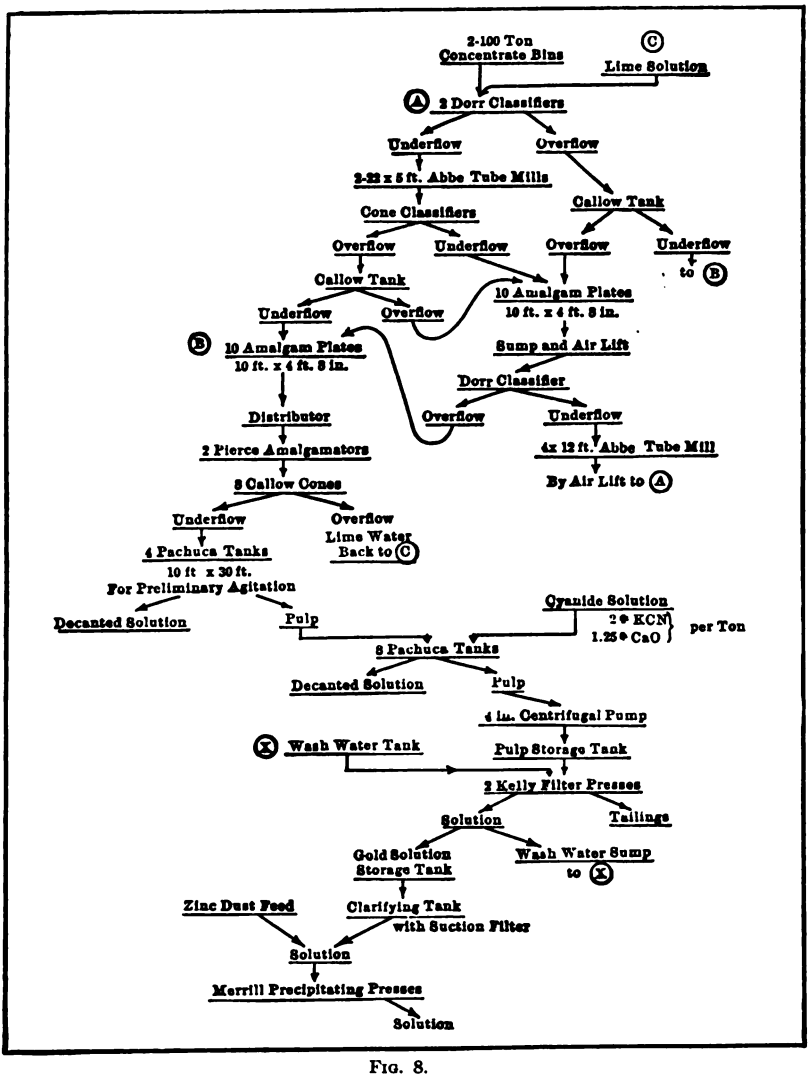

FIG. 8.

tanks with water containing 2 lb. lime per ton of water. The cyanide solution used for agitation contains but 2 lb. KCN and 1.25 lb. CaO per ton. The pulp is settled in Pachuca tanks and the solution decanted,

then fresh barren solution, 1.5 lb. KCN, added, and agitation started. This can be done in spite of the heavy sulphide pulp by employing a "spider with air jets" which slides down over the air-lift column and practically bores a passage through the pulp, which enables the lift to again start the pulp. The total time of treatment including charging and discharging in the Pachuca Tanks, is 24 hours.

Kelly filter presses are used to separate solution from solids. The flow sheet shows the use of amalgamation plates. The present practice is to dispense with amalgamation entirely, and the concentrate is ground in solution. The two sets of plates are circumvented by launders, the mill having been designed with the idea that amalgamation would probably be dispensed with. During August, 1911, 2010 tons of concentrate were treated at a cost of $2.8764 per ton and an estimated extraction of 97.02 per cent.

J. W. Hutchinson[1] and A. B. Parsons[2] describe the treatment of concentrate at the Goldfield Consolidated Mill. Much of the visible gold in the ore from this company's mines, although apparently free, is so coated that it cannot be recovered by amalgamation, and is only slowly soluble in cyanide solution. After a preliminary acid wash approximately 90 per cent. of the gold in 20-oz. concentrate was dissolved in 8-hours contact with a 4-lb. solution of cyanide. The management leased the old Kinkead mill for the purpose of treating the accumulation of concentrate from the Combination Mill. During the summer of 1908 approximately 300 tons were treated and yielded 94 per cent. of the gold content, at a cost which made it seem advisable to build a plant of similar type at the 100-stamp mill. It was found during experimental work, that the solutions became inert after 8-hour contact, and that it would be necessary to remove them and re-treat the concentrate with a solution freshly precipitated and freed of reducing agents. The process in use now consists in agitating the concentrate for 8-hour periods, and settling and decanting in the Pachucas, and is very satisfactory. During the first year's run, much difficulty was experienced in evolving a mechanical stirrer for the pulp during the acid wash and the subsequent water washes.

The concentrate from the mill gravitates through wooden launders to four 48 in. by 16 ft. amalgamating tables. The last section of these tables is covered with carpet for removing the coarser particles of gold, which do not amalgamate. The recovery by the combined process of amalgamating and carpeting amounts to approximately 35 per cent. of the value of the concentrate, or 25 per cent of the value of the ore. Of this amount, 10 per cent. is contained in the carpet concentrate, which

[1] *Min. and Sci. Press*, CII, 782.
[2] *Eng. and Min. Journ.*, XCI, 368.

is briquetted with the precipitate from the cyanide plant, and supplies the necessary sulphur for matting the copper in it.

It has been found most necessary, not to let the concentrate be in contact for any length of time with a weak solution of cyanide. For this reason the amount of cyanide required to bring the solution on the charge to 4.5 lb. KCN per ton, is first dissolved in the Pachuca agitator and the charge pumped into this. Exactly what effect a dilute solution of cyanide has on a heavy sulphide ore is not known here, but that the effect is deleterious has been demonstrated beyond question.

At the end of the fourth hour in alternate periods, peroxide of sodium is added to the charge which increases the activity of the solution and reduces the time of treatment. One-tenth lb. of lead acetate per ton of dry concentrate is added to each solution at the beginning of the period.

For two years the extraction of gold from the pulp in the Pachuca agitators has averaged 93 per cent., and the total extraction by amalgamating, carpeting, and cyaniding has averaged 95.23 per cent. A Kelly filter press (type B) containing 400 sq. ft. of filter surface is used for filtering the concentrate. It was found necessary to set this machine at a much greater inclination than is required for ordinary slime, since the heavy pulp has a tendency to pack in the bottom of the cylinder, which prevents the carriage from running out freely. The cost of treating concentrate is $5.85 per ton.

W. M. Brodie[1] describes in some detail the treatment of concentrate containing native silver at the Batopilas Mines in Mexico.

Chemistry of Cyanidaton

J. E. Clennell discusses experiments in electrolytic cyanide regeneration[2]. It is known that by far the larger part of the cyanide used in the treatment of ores is consumed in reactions other than those involved in the formation of the soluble double cyanides of gold and silver. Cyanide is lost in the following manner: (1) A portion is converted into hydrocyanic acid which volatilizes; (2) a portion, generally larger than the first is converted into sulphocyanides, ferrocyanides and other more or less complex combinations with constituents of the ore; (3) a portion combines with the zinc and takes the form of double cyanides of zinc, some of which may be regenerated by the precipitation of the zinc during ore treatment; (4) a portion is entirely decomposed, being changed into carbonic acid and ammonia, the former forming carbonates with alkalies and the latter escaping. A regenerative process can deal only

[1] *Mex. Min. Journ.*, XII, 20 (No. 1).
[2] *Eng. and Min. Journ.*, XCI, 1064.

with the second and third forms. Attempts have been made to recover cyanide in solutions by the addition of reagents,[1] but these attempts affect only the double cyanides, such as $K_2Zn(CN)_4$ and do not effect the sulphocyanides which are usually the most important quantitatively considered.

Recently, electrolytic cyanide regeneration[2] has been prominent and experiments were carried out, electrolyzing pure solutions containing sulphocyanides, ferrocyanides and alkali or lime, with carbon or peroxidized lead as anodes and lead as a cathode, and also on mill solutions and on ore pulp and solutions. No chemical, such as cyanamide, etc., were used. It was found that in sulphocyanide solutions with lime free KCN could be formed to some extent by electrolysis, and that the solution was efficient for extracting gold. In ferrocyanide solutions containing lime the regeneration of KCN was trifling only. In working mill solutions the regeneration by electrolysis was either nil or very small and the anodes were rapidly attacked. In the treatment of ore pulp negative results only were obtained; in some instances a slightly higher extraction, in others a reprecipitation of precious metals and destruction of cyanide. Current strengths of 6 to 70 amperes per square foot of anode surface were used. From the tests it seems that under ordinary circumstances the regeneration of cyanide from working solutions is imperfect and in many cases the action may be reversed. In the treatment of certain refractory ores it would appear that electrolysis is of some benefit, a slightly higher extraction being obtained than was possible by ordinary cyanide treatment, but on the whole the results are not promising.

Hans Foersterling[3] has been granted a patent relating to a method of generating halogen cyanide in connection with an oxidizing agent. He forms bromo-cyanide as follows: Sodium bromide is melted with sodium cyanide in the proportion of one molecule of each and the mixture is allowed to solidify. No decomposition takes place during this process. In order to make bromo-cyanogen the mixed fused salt is added with sodium peroxide into a cold dilute solution of H_2SO_4, all ingredients being present in proper proportion, according to the following reaction: $NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O$. This solution is then added to the cyanide solution from time to time as required.

L. W. Bahney[4] gives a rapid method for the estimation of CaO that is available in lime used in cyanidation. The value of lime depends upon the amount of free CaO present. The sample is ground to pass a 200-mesh screen, and 650 mgs. are added to 50 c.c. of distilled water in a

[1] *Mineral Industry*, XVIII, 358.
[2] MINERAL INDUSTRY, XIX, 311.
[3] *Eng. and Min. Journ.*, XCII, 1145.
[4] T. A. I. M. E., *Bul.*, 59, 896.

300 c.c. Erlenmeyer flask, which is shaken vigorously. Add 2 drops of phenolphtalein solution, then run in standard solution of oxalic acid until the pink color disappears, then shake again, and if the color returns, if it be due to lime, it will be vivid pink, and the addition of 0.5 c.c. will again cause the color to disappear. If the color is a faint weak pink, returning slowly, it is due to magnesia and the lime end-point has been reached. If the titration is allowed to stand for 15 to 30 minutes the bright pink color will return. If an oxalic acid solution of 14.6068 grm. pure crystal per liter is used the reading of the burette is the per cent. of CaO in the sample. The phenolphtalein solution is made by dissolving 0.5 grm. in 50 c.c. of alcohol and 50 c.c. of water. T. H. Aldrich Jr.[1] discusses the use of electrolytic oxygen in cyanide solution as an accelerator and purifier. The oxygen is generated in the solution by electrolysis. G. A. James[2] discusses the necessity of examining water for deleterious salts when using the same for cyanide solutions and states that in many instances it will pay to purify the water. He considers the presence of carbon dioxide in both water and air used in cyanidation as more destructive of cyanide than is generally supposed. The same author [3] discusses and describes interesting laboratory apparatus for making cyanide tests, which duplicate conditions existing in practice. W. D. Treadwell[4] investigated the titration of cyanide solutions containing $K_4Fe(CN)_6$ with silver nitrate and found that the ordinary method employed using a KI indicator is satisfactory. In the presence of $Na_2S_2O_3$ in the cyanide solution, correct results may be obtained by still further increasing the amount of indicator.

E. M. Hamilton[5] discusses the effect of considerable silver in cyanide solutions on the determination of sulphocyanates and ferrocyanide by the permanganate titration. He found that on acidifying with H_2SO_4, the silver precipitated largely as AgCNS, and that the filtrate, which usually contains the sulphocyanate, was practically free from it. When HCl is used in place of H_2SO_4 the sulphocyanate is not precipitated, particularly if a very small excess of the acid be used. He therefore concludes that in determining sulphocyanates in silver bearing cyanide solutions, the amount of HCl necessary to precipitate the silver be determined by preliminary trial and then a small excess over this added to the solution, or preferably the solution is added to the acid. The precipitated silver salt is filtered out. An aliquot portion of the filtrate is then taken and any ferrocyanide that many still be present is removed by the addition of ferric chloride, care being taken to wash the precipitate

[1] T. A. I. M. E. *Bul.*, 62, 177.
[2] *Min. and Sci. Press*, CIII, 415.
[3] *Min. and Sci. Press*, CIII, 839.
[4] *Min. and Sci. Press*, CIII, 655.
[5] *Min. and Sci. Press*, CII, 364.

and filter paper thoroughly. The filtrate and washings are then further acidified and titrated with permanganate in the usual way. A. W. Allen[1] calls attention to the fact that in silver bearing cyanide solutions, determinations for total cyanogen do not indicate that which is present combined with silver, so that pregnant solutions entering zinc boxes show less total cyanogen then the effluent solutions from which the silver has been precipitated and the cyanogen that was combined with silver, transferred to zinc as $K_2Zn(CN)_4$.

T. B. Crowe[2] has been granted a patent on the use of an alkaline hypochlorite in connection with an alkaline cyanide for dissolving gold and silver from telluride ores. R. C. Kline[3] in discussing the treatment of silver ore at Guanacevi, Mexico, gives some data on the reactions of silver minerals with cyanide solutions.

Cyanide Poisoning.—J. L. Jond[4] recommends the following treatment for cyanide poisoning: (1) Adrenalin to be immediately given—12 c.c. of a 1:1000 solution, diluted with water, or better 120 c.c. of 1:10000 solution which may be kept made up, although it deteriorates in time; (2) followed by Martin's antidote, 30 c.c. of a 23 per cent. ferrous sulphate solution, 30 c.c. of a 6 per cent. caustic potash solution, 2 grm. of light powdered magnesium oxide; (3) the stomach to be throughly washed out; (4) a further small dose of adrenalin to be given to follow up any cyanide which the antidote or wash has failed to reach. About one-half the dose in direction(1); (5) a good saline purge to be administered and the patient put to bed under the care of a physician. The usual remedy is Martin's antidote, which to be effective must be given within five minutes after taking the poison. By the use of adrenalin first, the time may be increased four minutes more.

The Mining Regulations Commission[5] of the Transvaal recommends the following measures to prevent cyanide poisoning and for treatment of men poisoned.

Preventive Measures.—(1) Provision of an adequate supply of wholesome drinking water about plants and assay offices, the same to be distinctly labeled 'Drinking Water'; (2) the replacement of strong cyanide solution used in the precipitation-boxes by water previous to the zinc being handled; (3) the effective hooding of the dissolving bath in which the gold precipitate is treated, and the use of some form of mechanical agitator in order to eliminate the necessity of raising the hood.

Treatment.—It is recommended that boxes labeled 'Antidote for Cyanide,' with directions for use affixed to the lids of the boxes, should

[1] *Min. and Sci. Press*, CIII, 51.
[2] *Met. and Chem. Eng.*, IX, 282.
[3] *Min. and Sci. Press*, CII, 402.
[4] *Eng. and Min. Journ.*, XCII, 625.
[5] *Min. and Sci. Press*, CIII, 418.

be kept in prominent and easily accessible parts of the cyanide plants. Each box should contain: A spoon and a metal receptacle to hold about 1 pint; one blue hermetically sealed vial containing 30 c.c. of 33 per cent. solution of ferrous sulphate; a white vial containing 30 c.c. of caustic potash, and one package of oxide of magnesium (light). The directions for the use of the antidote should be as follows:

1. Preparation of antidote. Quickly empty the contents of the blue vial, of the white vial, and of the magnesia package into the metal receptacle, and stir well with the spoon. This should be done as rapidly as possible, as the patient's chance of life depends on promptness.

2. Administration of the antidote. If the patient is conscious, make him swallow the mixture at once, and lie down for a few minutes. If the patient is not conscious, place him on his back and pour the mixture down his throat in small quantities, if necessary pinching his nose in order to make him swallow.

3. Incite vomiting. After the antidote has been given, try to make the patient vomit by tickling the back of his throat with a feather or with the finger, or give a tumblerful of warm water and mustard.

It must always be kept in mind that promptness of action is the first essential, as the poison acts quickly.

Precipitation and Treatment of Precipitates

C. W. Merrill[1] discusses zinc dust precipitation, which is finding increased application in cyanide plants, amongst others five Rand, South African plants now using this method of precipitation. The advantages of zinc dust precipitation may be summarized as follows: (1) Lower zinc consumption than with shavings, (2) Lesser cost of zinc dust. The dust costs 20 to 25 per cent. less than sheet zinc; (3) The ability to precipitate copper bearing cyanide solution, zinc shavings rapidly becoming polarized and non-effective while the dust, which is added continuously always presents fresh surface. At the Goldfield Consolidated athough the precipitate contains 40 per cent. copper, the precipitation with dust is uniformly good; (4) The method of applying zinc dust as now practiced insures reducing conditions in the solution, by absence of contact with the air; re-solution of precious metals, due usually to oxidizing conditions obtaining, is avoided; (5) A minimum consumption of other chemical aside from zinc, viz. KCN, is obtained by the small time of contact between the zinc and the solution that is necessary for precipitation; (6). With zinc-dust precipitation in presses, it is possible to make a complete clean-up of all precious metal in zinc and in solution; (7). The chance of loss of precipitate due to fire or

[1] *Eng. and Min. Journ.*, XCII, 407.

theft is minimized as the product is continuously collected and stored in heavy cast-iron presses which may be locked. The accompanying table gives data on zinc consumption.

Name of Mine or Region.	Consumption Per Ton Solution Precipitation.	Consumption per Ton Ore.	Consumption per Pound Bullion (fine).	Form of Zinc.
	lb.	lb.	lb.	
Homestake (gold)	0.17	.....	.....	Shavings.
Homestake (gold)	0.12	0.10	.....	Dust.
Pittsburgh-Silver Peak	0.167	.....	.....	Dust.
La Blanca (silver) Pachuca	.....	.....	0.93	Dust.
Santa Gertrudis	.....	.....	1.20	Dust.
Mexican Mills (average)	.....	.....	1.50	Shavings.

C. T. Rice[1] describes the zinc-dust feeder invented by B. A. Bosqui, of the Montana-Tonopah Mill. A. J. Clark[2] discusses some features of zincdust precipitation not generally considered. In some mills it is now and has been the practice to agitate the pregnant solution in the gold tanks by means of air. These solutions often highly charged with dissolved lime, precipitate $CaCO_3$, due to absorption of carbonic acid from the air, and this $CaCO_3$ is pumped with the zinc dust into the mains leading to the precipitate presses, tending to clog the mains and the filter cloths of the presses and interfering with precipitation. It is also in some instances customary to charge the zinc dust from the mechanical feeder into a cone mixer in which enough barren cyanide solution is added to flush the dust into the intake pipe of the pumps to the precipitate presses. The solution and dust are mixed in the cone by compressed air, and the lime in the solution may be precipitated on the particles of zinc dust, decreasing their efficiency. The same effect may be produced when water is used in the cone containing bicarbonate. At the Homestake Mines, South Dakota, a feeder has been introduced consisting of a hopper containing the zinc dust, resting on a roller revolving at 7 r. p. m., the feed being controlled by a gate on the front of the hopper. The dust falls at a steady rate, into a mixing cylinder, below the roller, into which a stream of barren solution enters, washing the dust into the intake of the pump. No air is used to mix dust and solution. Since the introduction of this device the zinc consumption at Plant No. 2 has fallen from 0.228 lb. to 0.139 lb. per ton of solution precipitated. The method of D. N. Hood[3] to detect gold in precipitated cyanide solutions is to slightly acidify with c. p. HCl and pass H_2S gas for 10 minutes, and judge the amount of gold present by the yellowness of the solution.

[1] *Eng. and Min. Journ.*, XCI, 361.
[2] *Min. Mag.*, IV, 289.
[3] *Eng. and Min. Journ.*, XCI, 162.

E. G. Banks[1] and J. G. Goosman[2] describe the treatment and refining of cyanide precipitate at the Waihi Mines, New Zealand. The precipitate is treated by the smelting process as developed at the Homestake Mines, S. D. and previously described.[3] The cupelled bullion is cast into anode plates $10 \times 4 \times 0.75$ inches in size which go to the company's electrolytic refining house, and there refined to fine silver and gold. R. F. Coolidge[4] describes the recovery of gold from the by-products of zinc-gold precipitate by melting them with a litharge flux in graphite crucibles and cupelling the lead bullion obtained in a small cupel furnace with a limestone cement test, the furnace being heated by four 2.5 in. Cary gasoline burner. The gasoline consumption is 0.87 gal. per hour for each burner, at a pressure of 40 lb per. sq. in.

E. R. Haggin[5] describes a method recovering gold and silver from a comparatively low-grade slag from the melting of cyanide precipitate. The slag which contained 112.5 oz. Ag and 2.41 Au per ton, was ground in a tube mill, the heavy metallics sifted out and the slime agitated for 120 hours in a 0.5 per cent. KCN solution. The extraction by the metallics and cyanidation was about 96 per cent. of the gold and silver. H. B. Kaeding[6] discusses the problems of short zinc, previously referred to.[7] W. H. Coghill[8] describes a combination melting and cupelling furnace for the refining of precipitate.

J. B. Aarons and H. Black[9] describe the treatment and refining of precipitates at the Chaffers gold mine in West Australia. An anonymous writer[10] discusses the use of the Faber du Four retort furnace for distilling zinc from cyanide precipitate before melting.

Miscellaneous

W. Neill[11] gives data on the cyanide industry and states that the cyanide of to-day has sodium as a base and is supplied in three grades containing cyanogen equal to (1) 98 to 100 per cent. KCN; (2) 120 per cent. KCN; and (3) 128 to 130 per cent. KCN. The first and third grades are fused cyanides mostly obtained by using methods of the Castner patents employing metallic sodium as a base and ammonia as the source of nitrogen. The second class is in the form of briquettes and is obtained by a wet process, working up a by-product of the beet-sugar industry. The present production exceeds the demand. The world's

[1] *Eng. and Min. Journ.*, XCII, 153.
[2] *Min. Mag.*, IV, 61.
[3] Mineral Industry, XVI, 551.
[4] *Eng. and Min. Journ.*, XCII, 756.
[5] *Min. and Sci. Press*, CII, 746.
[6] *Mex. Min. Journ.*, XII, No. 3, p. 23.
[7] Mineral Industry, XVIII, 362.
[8] *Min. and Sci. Press*, CII, 819.
[9] *Eng. and Min. Journ.* XCII, 1025.
[10] *Eng. and Min. Journ.*, XCII, 111, 1070.
[11] *Eng. and Min. Journ.*, XCI, 902.

gold production in 1910 was 22,537,358 oz. of which about 25 per was produced by the cyanide process. In view of the fact that so of the cyanide now used is sodium cyanide, the practice of estim the cyanogen content of the salt and of solutions in terms of KC pears absurd and useless and a change seems desirable. The Home mills in S. D. at the suggestion of W. J. Sharwood now use the foll data: Standard Silver-Nitrate solution, $\frac{1}{20}$ Normal. 1 c.c. co 0.0054 grm. silver = 0.0085 grms. Ag NO_3; 1c.c. = 0.004 grm. CN = 0. grm. KCN = 0.00491 grm. NaCN. Formula for titrating solutio commercial sodium cyanide; *Using* 10 *c.c. samples of solution;* num c.c. used divided by 20 = percentage of sodium cyanide; number used = pounds NaCN per ton. *Using* 20 *c.c. sample of solution;* nu of c.c. used divided by 40 = percentage of sodium cyanide; num c.c. used divided by 2 = pounds NaCN per ton. The weights an centages thus found are those of the commercial cyanide in use as 97 per cent. pure, equivalent to 129 per cent. KCN or 51.5 per CN. J. McCombie[1] gives a brief account of the early history cyanide process in New Zealand. The cyanide plant[2] of A. B. built in 1891, near the Calumet mine, Shasta Co. Calif. is describe

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Progress In Gold Milling In 1911

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[5] *Min. and Sci. Press*, CIII, 254.
[6] *Eng. and Min. Journ.*, XCII, 151.
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Single Stamps Versus Five-stamp Batteries.[3]—The single-stamp unit has higher capacity owing to its greater area of discharge but it is believed that this disadvantage would disappear if quadruple-discharge five-stamp mortars were used. One big feeder is preferable to five small ones, one belt is better than two, and two cam boxes are easier to grease than four. The labor cost is higher for the single-stamp units.

Anchoring Battery Posts and Mortars.[4]—Considerable discussion has

[1] P. N. Nissen, *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XII (1911), pp. 111, 182, 225; *Mining World*, Vol. XXXV (1911), p. 1316 *Mining and Scientific Press*, Vol. CIII (1911), p. 840; *Engineering and Mining Journal*, Vol. XCII (1911), p. 1219; *Mining Magazine*, Vol. V (1911), p. 462.

[2] *Engineering and Mining Journal*, Vol. XCI (1911), p. 402.

[3] A. Del Mar, *Mining and Scientific Press*, Vol. CII (1911), p. 633.

[4] F. W. Sewell, *Informes y Memorias del Instituto Mexicano de Minas y Metalurgia*, Vol. II (1911), p. 97; *Engineering and Mining Journal*, Vol. XCII (1911), p. 629. *Mining and Scientific Press*, Vol. CIII (1911), p. 265; *Mining World*, Vol. XXXV (1911), p. 437; *Mining Science*, Vol. LXIV (1911), p. 226; *Metallurgical and Chemical Engineering*, Vol. IX, p. 475; *Journal Chemical, Metallurgical, and Mining Society, South Africa*, Vol. XII (1911), pp. 145, 201; *Mining and Scientific Press*, Vol. CIII (1911), p. 19; C. D. Kaeding and W. H. Storms, *Ibid.*, p. 51; J. B. Fleming and M. P. Boss, *Ibid.*, pp. 82, 236; C. H. Waters, *Ibid.*, p. 173; K. Baumgarten, *Ibid.*, p. 359; F. C. Perkins, *Ibid.*, p. 483, *Ibid.*, p. 527; A. Del Mar, *Ibid.*, p. 747; *Mining World*, Vol. XXXV (1911), pp. 1023, 1180.

TABLE I.—SUMMARY OF COMPARATIVE TESTS BETWEEN NISSEN AND CITY DEEP STAMPS.

Test Number and Date.	1 June 23, 1911.		2 July 5, 1911.		3 July 17, 1911.		4 July 26, 1911.		5 Sept. 11, 1911.		6 Sept. 16, 1911.	
Number of stamps and kind	4N.	10 C.D.	4N.	10 C.D.	4N.	10 C.D.	4N.	10 C.D.	4N.	10 C.D.	4N.	10 C.D.
Weight of stamps in pounds	1932	1863	1927	1859	2245	1855	2237		1993	1775	1991	1773
Number of drops per minute	103	100	103	100	103	100	103		103	100	103	100
Height of discharge in inches	2¾	2½	2	2¾	2	2½	2		2½	2½	2½	2½
Height of drop in inches	8½	8½	8½	8½	8¾	8¾	8½		8½	8½	8½	8½
Class of screen	Tyler.	T. C.	Tyler.	T. C.	Sq. m.	14 (.08)	T. T. C.		Sq. m.	Wire.	Sq. m.	Wire.
Mesh of screen	9	9	9	9	⅜ in.	⅜ in.	9		9	9	⅜ in.	⅜ in.
Width of aperture in inches	0.205	0.205	0.205	0.205	0.375	0.375	0.23		0.277	0.277	0.375	0.375
Length of aperture in inches	0.536	0.536	0.536	0.536	0.375	0.375	0.85		0.277	0.277	0.375	0.375
Dry ore crushed in tons	180.86	192.75	180.25	192.12	193.88	206.66	191.7		185.32	194.53	185.7	191.2
Total run in hours	44.33	25.33	39.0	23.166	31.70	21.83	37.28		37.29	22.28	29.52	18.85
Tons per stamp per 24 hours	24.47	18.26	27.73	19.90	36.69	22.72	30.85		29.81	20.95	37.74	24.34
Horse-power per stamp[1]	4.09	4.21	4.09	4.21	4.95	4.21	4.95		4.23	4.65	4.23	4.65
H.-P. hours per ton of ore crushed[1]	4.0	5.5	3.56	5.06	3.24	4.45	3.8		3.41	5.32	2.71	4.56
Pounds ore crushed per pound falling weight per day.	25.33	19.6	28.78	21.412	32.68	24.49	27.58		29.91	23.6	37.91	27.45
Tons water per ton of ore	5.4	2.7	4.2	2.4	3.1	2.15	3.2		3.3	3.3	3.03	3.26
Wear shoes and dies in pound per ton crushed	0.129	0.145	0.139	0.24	0.198	0.187	0.122		0.147	.....	0.112	0.08
Sizing Test of Feed.												
Over 2 in	36.9	12.1	14.96	12.4	26.9	18.2	52.3	.....	39.0	14.1	23.7	27.1
Through 2 on 1½ in	18.6	14.4	14.50	11.35	18.75	16.0	14.5	.....	19.7	18.0	20.9	17.1
Through 1½ on ½ in	15.1	20.2	20.40	19.35	19.3	18.15	10.4	.....	15.0	21.3	18.8	19.1
Through ½ on ¼ in	11.7	12.7	16.95	17.70	13.05	16.25	9.4	.....	9.3	16.0	12.7	12.6
Through ¼ in	27.5	37.9	33.20	39.20	22.0	31.4	13.4	.....	17.0	30.6	23.9	24.1
Sizing Test of Discharge.												
On 60 mesh	65.2	63.4	64.9	63.4	67.24	69.33	67.3	.....	66.6	66.5	70.7	70.7
Through 60 on 90 mesh	8.3	9.1	8.46	8.70	8.62	9.09	8.75	.....	9.9	10.2	10.1	10.3
Through 90 on 200 mesh	9.7	12.0	8.46	9.10	10.84	9.82	8.55	.....	8.8	10.9	10.2	9.2
Through 200 mesh	16.7	10.7	18.10	19.5	13.3	11.76	15.4	.....	14.7	12.4	9.0	9.8

No competitive tests.

[1] Excluding belt, shafting and motor losses.

TABLE II.—RELATIVE MECHANICAL EFFICIENCY.

Mesh.	Nissen Stamps.			City Deep Stamps.		
	Sizing Test, Per Cent.		Mech. Value Efficiency Units.	Sizing Test, Per Cent.		Mech. Value Efficiency Units.
On 5– 9	19.80		1.782	14.8		1.332
On 8–11	10.75		1.183	11.28		1.241
On 12–13	9.30	66.54	1.209	10.59		1.377
On 20–15	10.04		1.506	10.57	64.78	1.586
On 30–17	7.92		1.346	8.50		1.445
On 50–19	8.73		1.659	8.86		1.683
On 80–21	10.17	10.17	2.136	11.48	11.48	2.411
On 200–24	4.71	23.29	1.130	5.38	23.74	1.291
Through 200–28	18.58		5.202	18.36		5.141
Totals		100	17.153		100	17.507
Duty in 24 hours	27.72 tons			19.9 tons		
Weight of stamps	1927 lb.			1860 lb.		
Relative mechanical efficiency[1]	$\frac{17.153 \times 27.72}{1927} = 0.25$			$\frac{17.507 \times 19.9}{1860} = 0.19$		

taken place over the anchoring of battery posts to the foundation by means of iron shoes and anchor bolts; also over the bolting of the mortar to the concrete mortar block. Several methods of the latter are described using long bolts and short bolts, vertical and inclined bolts, with nuts or with keys, with and without enclosing casings, with and without angle irons or other special forms of anchors imbedded in the concrete especially where short bolts are used, and showing the use of pockets in the concrete block for ease both of installing and also of removing.

Wear of Screens in Single-stamp Mills.[2]—There are two sources of loss, one in excessive wear, and one in time lost in changing screens. Single-stamp screens may last only 2 or 3 days against 20 days for screens in five-stamp batteries. The causes are the proximity of the die to the screen, the angle at which the ore is impelled against the screen, the low height of discharge and the bending of the screen inward and outward by the suction of the stamp. The screen which lasts the longest under this bending action is a blued-steel needle-punched screen. The Boss scheme of having a high discharge loses the only advantage claimed for single-stamp mortars, viz., high capacity. It is suggested that the life of the screens may be increased by inclining them outward and by enlarging the upper part of the mortar to allow a freer expansion and contraction of the air.

Amalgamation in Slow Speed Chili Mills.[3]—At the Gold Hill mill near Quartzburg, Idaho, the soft quartz ore containing gold and about 5 per cent. sulphides, mostly pyrite, is crushed by breaker and stamps

[1] If weight assumed to represent horse-power consumed.
[2] A. Del Mar, *Engineering and Mining Journal*, Vol. XCII (1911), p. 836.
[3] E. E. Carter, *Mining and Scientific Press*, Vol. CII (1911), p. 370; *Mining Magazine*, Vol. IV (1911), p. 306.

to pass through 1/4-in. screen and then ground and amalgamated in a 10-ft. Lane Chili mill making 8 r. p. m. The mill has six wheels and the total crushing weight is about 12.5 tons. The mill has no screen but the pulp passes over an overflow discharge 6 in. high, yielding a product of which 90 per cent. will pass 80 mesh, 80 per cent. 100 mesh, and 50 per cent. 200 mesh. An extraction of 81 per cent. by amalgamation has been attained against 50 to 55 per cent. by fine stamps in the same mill. It is hoped to attain 85 per cent. extraction and thus eliminate the concentration and cyaniding used at present.

Giesecke Ball-tube Mill.[1]—This was tested by the Mines Trials Committee of the Chamber of Mines in South Africa and at first favorable reports were published but the final report was that the wear and tear was too great to make it a success.

New Type of Ball Tube Mill.[2]—This is a cylinder 23 ft. long and 8 ft. diameter for one-third its length and 6 ft. for the rest. There are three compartments and steel balls are used graded from the coarse to the fine end. Inside ribs serve to lift the balls part way up the sides. This appears to resemble the Giesecke mill and to have the same faults.

Fine Crushing with Hardinge Mill.[3]—The following sizing tests made on a lead-zinc ore from Butte, Montana, give comparisons between one-stage Hardinge conical mill using steel balls and seven-stage crushing in rolls with the fines screened out after each passage.

	Once through Hardinge Mill. Per cent.	Rolls Seven Returns. Per cent.
On 60 mesh.....	10.18	32.20
Through 60 on 80 mesh....	9.92	21.03
Through 80 on 120 mesh..	23.15	18.62
Through 120 mesh.....	40.32	21.34
Through 120 mesh as slime.	16.83	8.81

Use of Quartz in Tube Mills.[4]—The time lost in frequent examinations of the interior of the mill and replacement of lining may become a serious matter and offset to a considerable extent the gain in using inexpensive quartz instead of costly pebbles. Twenty per cent. of pebbles in the total load of a mill should be sufficient to insure proper

[1] *Engineering and Mining Journal*, Vol. XCII (1911), pp. 594, 736; *Mining Magazine*, Vol. V (1911), pp. 14, 29, 67, 259; *South African Mining Journal*, May 20, 1911; *Mining and Scientific Press*, Vol. CIII (1911), p. 419; *Mining World*, Vol. XXXV (1911), pp. 236, 688; *Journal Chemical Metallurgical and Mining Society, South Africa*, Vol. XII (1911), p. 256; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 480; *Mines and Minerals*, Vol. XXXII (1911), p. 102; *Canadian Mining Journal*, Vol. XXXII (1911), pp. 471, 684.

[2] *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XI (1911), p. 579; from *Pacific Miner*, Feb. 1911, p. 58.

[3] *Mines and Minerals*, Vol. XXXI (1911), p. 604; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XI (1911), p. 632.

[4] A. G. Quartano, *Engineering and Mining Journal*, Vol. XCI (1911), p. 1017.

protection of the lining. Suggestion is made that mill men should study this problem and determine for their individual mills what is the most economical mixture of quartz and pebbles for them to use.

Critical Moisture in Tube Mill Feed.[1]—This is about 39 or 40 per cent. by weight or about twice as much as is necessary to fill all the voids in the feed sand. The moisture desired depends somewhat on the tonnage and size of the sand. If too little water is present the pulp is too viscous, power consumption is increased and pebbles tend to go out into the discharge. Mills have been run with as low as 28 per cent. moisture.

Method of Returning Pulp to Classifier from Tube Mill.[2]—The scheme proposed (see figure 9) is a combination of the tube mill itself with a Frenier pump. The spiral wheel of the pump is bolted to the feed end

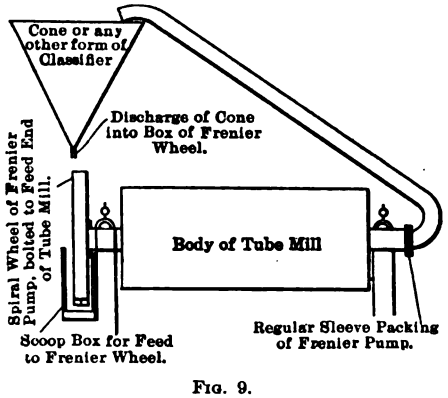
Cone or any other form of Classifier
Discharge of Cone into Box of Frenier Wheel.
Spiral Wheel of Frenier Pump, bolted to Feed End of Tube Mill.
Body of Tube Mill
Regular Sleeve Packing of Frenier Pump.
Scoop Box for Feed to Frenier Wheel.

FIG. 9.

of the tube mill and the sleeve packing joint is bolted to the discharge end of the tube mill. The pumping action keeps the mill always a little more than half full of pulp thus giving it a greater grinding capacity. No extra water is required in this case and there is no slop if the mill stops.

Economics of Tube Milling.[3]—In a lengthy paper Mr. Ball devotes the first part to the history and applications of the tube mill and the theories of rock crushing and the second part to the results of his experimental investigations. The tests may be divided into four sets: for feed, for moisture, for pebble load, and for speed. A general summary of these tests is given in the table. Standard I. M. M. screens were used

[1] J. W. Bell, W. R. Dowling, A. Quartano, *Mining Magazine*, Vol. IV (1911), pp. 263, 439; Vol. V (1911), p. 223; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XII (1911), p. 62.
[2] C. Shapley, *Engineering and Mining Journal*, Vol. XCI (1911), p. 258.
[3] H. S. Ball, *Bulletin* No. 83, Institution Mining and Metallurgy; *Mining Magazine*, Vol. V (1911), p. 388; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 604; *Mining Science*, Vol. LXIV (1911), p. 313; *Canadian Mining Journal*, Vol. XXXII (1911), p. 614; *Mining and Scientific Press*, Vol. CIII (1911), p. 383.

Test.	Pebble Load. Pounds.	Pebble Volume.	Revolutions per Minute.	Peripheral Speed per Minute. Feet.	Moisture. Per cent.	Feed per 24 Hours. Tons.	Horse-power.	Work Done per Unit.	Material below 120 Mesh per 24 Hours. Tons.	Relative Mechanical Efficiency per Horse-power.
Feed Tests.										
P	1200	0.5	41	369	38.0	7.2	6.3	3.89	2.27	4.45
C	1200	0.5	41	369	38.0	9.6	5.0	3.36	2.34	6.45
B	1200	0.5	41	369	38.0	12.6	5.1	2.82	2.95	7.02
O	1200	0.5	41	369	38.0	14.4	5.6	2.91	3.31	7.48
A[1]	1200	0.5	41	369	38.0	18.6	5.8	2.58	3.96	8.29
R	1200	0.5	41	369	38.0	23.0	6.6	2.09	3.79	7.28
Moisture Tests.										
D	1200	0.5	41	369	30.0	12.6	5.7	2.99	2.93	6.61
B[1]	1200	0.5	41	369	37.7	12.6	5.0	2.82	2.95	7.11
E	1200	0.5	41	369	50.0	12.6	6.2	3.05	2.95	6.16
F	1200	0.5	41	369	58.0	12.6	6.0	2.84	2.97	5.96
Pebble Load Tests.										
G[1]	900	0.37	41	369	38.0	12.6	4.9	2.75	2.65	7.07
B	1200	0.50	41	369	38.0	12.6	5.1	2.82	2.78	7.02
H	1500	0.63	41	369	38.0	12.6	6.9	3.31	3.40	6.04
Speed Tests.										
T	1200	0.5	33	297	38.0	7.2	6.8	4.19	2.45	4.43
Q[1]	1200	0.5	37	333	38.0	7.2	5.1	4.17	2.59	5.89
P	1200	0.5	41	369	38.0	7.2	6.3	3.89	2.27	4.45
S	1200	0.5	46	414	38.0	7.2	7.5	4.03	2.48	3.87

[1] Most efficient test of series.

for sizing, and crushing efficiency is calculated by Stadler's method.

The inside dimensions of the tube mill were 3.5 ft. length, and 2 ft. 10 in. diameter. A silex lining was used. The feed was 18-mesh sand and the discharge was sized on 120-mesh screen. The first set of tests shows the best feed to be 18.6 tons in 24 hours and there is a decrease in efficiency both above and below this point. The best percentage moisture as shown by the second set of tests is 37.7 per cent. by weight which figure agrees very well with the conclusions of previous investigators. For pebble load the best figure is 1050 lb., equal to seven-sixteenths of the internal volume. The best speed of revolution for this particular mill under the conditions given in the table is 37 revolutions per minute.

Ball Mill Practice at Kalgoorlie.[1]— The ore is hard, silicious material consisting mainly of quartz or quartz schist. The ball mills are of the Krupp type and for best work should be mounted on a concrete foundation. With eight men a mill can be erected in 24 hours after the foundation is ready. There are two sizes of mills in use at Kalgoorlie with the dimensions shown in the table at top of the following page.

From 24 to 26 r. p. m. is the correct speed for No. 5 and 21 to 24 for No. 8. The mills are loaded with 1 to 2 tons of 5-in. steel balls. A No. 5 mill takes 18 to 23 horse-power and No. 8, 50 to 60 horse-power.

TABLE I.—BALL MILL PRACTICE AT KALGOORLIE (DRY CRUSHING).

Mine.	Number of Mills.	Weight of Balls. Pounds.	Screens. Mesh.	Speed. r. p. m.	Horse-power.	Drive.	Daily Capacity. Tons.	Steel Consumption per Ton. Pounds.	Life of Grinding Plates. Days.
Associated	4 No. 8	4488	30×30	21	60	Belted from shaft	92–95	0.50	170
Associated	6 No. 5	2240	25×25	25	23	Belted from shaft	43	0.50	170
Associated Northern	3 No. 5	2350	27×27	26	18	Belted from shaft	40	0.32	270
Chaffers	3 No. 5	2300	27×27	25	25	Belted from shaft	40	0.74	180
Great Boulder	4 No. 8	4480	30×30	24	60	Belted from shaft and friction clutch	80–90	0.64	105
Kalgurli	9 No. 5	2200	26×26	25	25	Belt from shaft	40	0.45	300
Perseverance	8 No. 8	4400	27×27	24	60	Motor and belt	100	0.65	118
South Kalgurli	3 No. 8	4480	27×27	24	65	Motor and belt	95–100	0.47	210
South Kalgurli	1 No. 5	2800	.....	.....	30	Motor and belt	.....	0.47	210

[1] M. W. Von Bernewitz and D. F. Foster, *Mining Magazine*, Vol. V (1911), pp. 139, 304; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 520; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XII (1911), p. 251; *Mining and Scientific Press*, Vol. CIII (1911), p. 68.

	No. 5. Inches.	No. 8. Inches.
Diameter	89	106
Width	46	54
Mantles for plates	10	12
Inside screens	5	6
Outside screens	10	12
Area outside screens, each	46×26	54×24

A fan is used to suck away the dust from the inside of the housing. Shaking tray feeders supply the ore to the mills and the ore supply is regulated by the attendant according to the sound of the mill. The action of crushing is considered to be one of pure rolling, grinding and abrasion, instead of impact as in the tube mill. The crushed particles can escape through the screens as soon as they are fine enough, thus differing from the case of stamps.

One man can attend eight ball mills. A No. 5 ball mill including foundations may be erected for $3000 and No. 8 for $5000. The former will cost about $1500 yearly for maintenance.

Tables I, II and III, give details of ball mills in various mills, sizing tests of products and costs of running.

TABLE II.—SIZING TESTS.

	Associated.	Associated Northern.	Chaffers.	Great Boulder.	Kalgurli.	Preservance.	South Kalgurli.
On 30 mesh	.....	.....	.....	1.00	.....	0.20	.....
Through 30 on 40 mesh.	16.16	11.20	10.25	5.80	8.20	3.80	10.20
Through 40 on 60 mesh.	15.66	20.10	13.19	14.20	18.00	20.00	21.20
Through 60 on 80 mesh.	10.46	8.30	8.60	8.70	13.40	7.80	6.00
Through 80 on 100 mesh.	6.43	11.80	7.25	9.00	9.10	5.40	2.60
Through 100 on 120 mesh.	3.76	3.20	2.70	2.30	2.40	4.00	3.10
Through 120 on 150 mesh.	3.46	2.00	2.70	2.30	2.40	4.00	3.10
Through 150 mesh.	44.00	43.60	56.30	52.00	45.30	54.20	55.90

TABLE III.—COST IN CENTS PER TON.

	Associated.	Associated Northern.	Chaffers.	Great Boulder.	Kalgurli.	Persevernance.	South Kalgurli.
Wages	12.90	11.74	4.18	3.62	.....	6.42	4.10
Power	27.56	24.16	21.28	19.50	30.46	38.36	41.64
Spares	13.46	9.46	.....	.....	.....	6.64	11.08
Stores	5.34	2.84	0.50	12.52	14.98	0.92	.....
Repairs	0.98	0.20	13.76	.....	.....	0.22	7.36
Sundry	.....	.....	3.42	.....	.....	1.08	0.42
Total	60.24	48.40	43.14	35.64	45.44	53.64	64.60

AMALGAMATION

Inside Amalgamation.[1]—Sometimes inside amalgamation is absolutely necessary. A case of a mill is cited in Colombia, South America, where the ore was high grade and contained 13 per cent. sulphides, half being pyrite and half blende and galena in about equal proportion. The gold was fine and had to be well brightened by the use of a high discharge and a fine screen before it would amalgamate. A black scum formed on the outside plates and it was impossible to keep them bright either by the use of chemicals or by other means, so that if the gold did not amalgamate on the inside it was impossible to catch but a small proportion on the outside. Careful watching and feeling of the plates and the more frequent addition of mercury (as often as every 15 minutes) raised the extraction from a total of about 50 per cent. up to 70 per cent. The extra saving was effected entirely on the inside plates.

The difficulty arising from inside amalgamation is that it requires considerable experience and skill to judge if the inside plates are in condition by the appearance of the head of the outside and the feel of the inside plates. Sufficient mercury must be added to the mortar to take up all the gold but not so much as to soften the plates. It is best added frequently in small amounts.

Electroplating Battery Plates.[2] The details of treatment, especially the electroplating, as used by the Oriental Consolidated Company in Korea, are given in full in this article.

Absorption of Gold by Amalgamated Copper Plates, Cast Iron Retorts and Amalgam Trays.[3]—An old plate which had been in use in the mill of the Crown mines for 12 years and which was 0.13 to 0.14 in. thick after being sweated and scraped, was assayed yielding 1.69 per cent. gold and 1.31 per cent. mercury. Planing successive thicknesses from the top yielded results shown in the table:

Layer No.	Per cent. Gold.	Per cent. Mercury.	Per cent. Lead.	Ratio Gold to Mercury.
1	10.48	9.39	0.085	1.12
2	0.0166	*	0.084	?
3	0.00505	*	0.080	?
4	0.0084	*	0.053	?
5	0.0041	*	0.097	?
6	0.0145	*	0.089	?
7	0.0018	1.89	0.104	0.00105
Total.....	1.21			

* Present in all cases but in amounts too small for determination.

[1] K. C. Parrish, *Engineering and Mining Journal*, Vol. XCI (1911), p. 260.
[2] J. D. Hubbard, *Mining and Scientific Press*, Vol. CIII (1911), p. 135.
[3] G. H. Stanley and M. T. Murray, *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911), p. 218.

The conclusion is that although a scale of gold amalgam may form on the surface of a plate, the actual absorption is very slight amounting to only 0.25 oz. gold for the whole plate out of a total of 46 oz. in the sweated and scraped plate.

A piece of an old retort about 2 1/4 in. thick which had probably been in use for about 2 years was planed with the results shown in the table:

	Per cent. Gold and Silver	Per cent. Gold.	Ratio of Gold to Gold and Silver.
Surface Scrapings	79.643	75.300	0.9452
Slice No. 1, 1/16 in. thick	30.696	29.078	0.9475
Slice No. 2, 1/18 in. thick	6.388	6.051	0.9470
Slice No. 3, 1/4 in. thick	0.016	0.0152	0.9500
Slice No. 4, 1/2 in. thick	0.0013	0.00117	0.9000
Slot No. 5, 1/8 in. thick	None	.....	.....
Slot No. 6, 1/8 in. thick	None	.....	.....
Outside 1/8 in. thick	0.0010	*	*

* Not Determined.

Microscopic examination showed absorption of gold in the cracks in the iron and also in the cavities formed in the iron by the burning out of the graphite.

Although these investigations indicate that gold is absorbed yet the amount is so small that it is practically negligible.

Treatment of Mill Concentrate.[1]—The scheme at the Geldenhuis Deep is shown in figure 10. The material treated is the daily scrape from

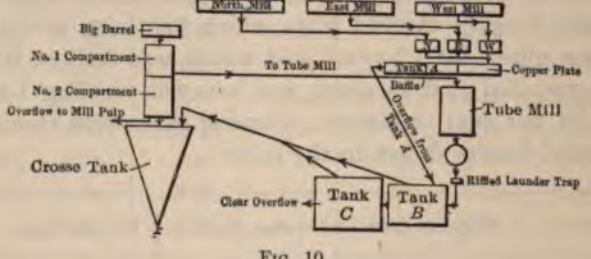

FIG. 10.

the three mills and the big barrel scrape. The former is ground in barrels *N*, *E*, and *W* for 4 or 5 hours and run over copper plate to tank *A*, 4 ft. 6 in. by 2 ft. 2 in., fitted with baffle 18 in. from feed end. The heavy material containing 45 per cent. finer then 200 mesh (0.003 in.) settles behind the baffle and the remainder containing 98 per cent. finer than 200 mesh flows by 2-in. pipe to tanks *B* and *C* outside the

[1] R. Lindsay, *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911) pp. 2, 134, 223; *Mining World*, Vol. XXXV (1911), p. 747; *Mining and Scientific Press*, Vol. CIII (1911), p. 416; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 519.

building. Tank *A* is cleaned out daily and yields 250 lb. of dry concentrates assaying about 30 oz. gold per ton, which is fed to tube mill 4.5 ft. by 3.5 ft., lined with 6 by 6 by 3-in. silex blocks and running at 26 r. p. m., using pebbles for grinding. The tube-mill discharge (98 per cent. finer than 200 mesh) passes to batea running at 100 r. p. m. with a 3-in. stroke, a bath of 250 oz. of mercury being maintained therein. The batea catches 7 oz. mercury and 50 oz. amalgam per ton, and this 50 oz. amalgam yields 11.28 oz. fine gold at a cost of $0.72 for power and $1.20 for labor. The batea overflow passes over riffled launder and trap to tanks *B* and *C*. The final overflow of *C* is perfectly clear and goes to waste. The tank *C* product is all below 200 mesh, assays 12 to 14 oz. gold per ton and amounts to about 5 per cent. of the solids caught in tank *B*. The products of these two tanks go to the Crosse tank cyanide plant at the East mill.

The big barrels in the respective mills discharge over stationary amalgamating plates into a two-compartment tank. No. 1 compartment yields 3.5 tons per month, containing 60 per cent. finer than 200 mesh and assaying 7.5 oz. gold per ton which goes to the tube mill at the central amalgam room every month. The second compartment overflows to the mill pulp launder while the settlings amounting to 2 tons per month, containing a little over 90 per cent. finer than 200 mesh and assaying 6.75 oz. gold per ton, go to the Crosse[1] tank at the East plant.

The average value of the black sand concentrate from the three mills is 16 oz. gold per ton and taking the average assay of the residue at $4.00 per ton and the cost at $6.25 per ton, the profit is $311 per ton.

The Simmer Deep Jupiter plants use much the same process except that lime is added to the amalgam barrels to avoid explosions, the batea is replaced by amalgamating plates, and an ordinary cyanide plant replaces the Crosse tank.

Other Gold Milling Machinery.—The reader is referred to the article on Progress in Ore Dressing and Coal Washing in this volume for other apparatus which may find some place in gold mills.

General Practice and Individual Mills

Cyaniding Versus Concentration and Cyaniding for Silver Ores.[2]—The refining charge amounting to 1.6 cents per ounce of bullion from cyanide precipitate is a factor which is sometimes lost sight of. Taking the case of Pachuca ores in Mexico, one ton of ore assaying 30 oz. silver per ton will yield 10 lb. of concentrates assaying 1200 oz. per ton. The cost per ounce of silver produced from concentrates is as follows:

[1] Min. Industry, XVIII, 350.
[2] G. A. Denny, *Mining Magazine*, Vol. V (1911), p. 433.

Cost to produce 1 ton of concentrates	$48.60
Freight and treatment	10.25
Loss in treatment	29.16
Total	$88.01
Ounces silver produced from 1 ton concentrate	1140
Cost per ounce	$0.77

The approximate cost of producing an ounce of fine silver from 1 ton of ore by cyanide treatment from a 20-oz. ore is as follows:

Cost of cyaniding, precipitating and melting	$1.08
Refining cost	0.54
Loss on treatment	1.32
Total	$2.84
Ounces silver produced from 1 ton of ore	27
Cost per ounce of silver produced	$0.109

Amalgamation of Gold in Banket Ore.[1]—For some time it has been considered that the thickness of the tube-mill discharge pulp necessitated the use of shaking plates but recent tests at the Simmer East plant have shown that two stationary plates after the tube mill yield a gold recovery of 21.54 per cent. of the screen value and 34.17 per cent. of the tailing value against 21.80 per cent. and 34.04 per cent. respectively when five shaking plates were used in the same place. The slope of the stationary plates is 18 per cent. and the pulp flowing over them contains 55 per cent. moisture.

The removal of battery plates has led to the introduction of separate plate houses with the attendant advantages of increased running time of the stamps, better amalgamation, closer supervision, and less danger of theft. There is also a considerable reduction of total plate area. Under the old practice the total battery and tube-mill plate area was about 12 sq. ft. per stamp. At the Randfontein Central it is proposed to use only 4.8 sq. ft. and the Simmer and Jack rearrangement has about 2 sq. ft. This leads to reduced cost of installation, less gold locked up in amalgam, and reduced mercury consumption, the last being one-sixth the former figure in the Simmer and Jack mill.

Homestake Mill, South Dakota.[2]—The mills of this company have been described in Richard's Ore Dressing Vol. II, p. 1025, and Vol. IV, p. 1639. More recently a regrinding plant has been installed to treat tailings from ore coming from the unoxidixed portions of the deposits on the South side. The coarse particles in these tailings contain gold which is not satisfactorily extracted by cyanide and which is partially amalgamated by fine crushing. The regrinding plant handles 210 tons of coarse tailings daily or about 7.5 per cent. of the ore crushed in the

[1] W. R. Dowling, *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XI (1911), pp. 414, 499, 557, 609, Vol. XII (1911), p. 30, 178; *Institution Mining and Metallurgy*, Vol. XX (1911), p. 270; *American Metallurgical Society*, Vol. I (1911), p. 12; *Mining Science*, Vol. LXIII (1911), pp. 565, 595; *Metallurgical and Chemical Engineering*, Vol. IX (1911), pp. 279, 322, 519; *Mining World*, Vol. XXXIV (1911), p. 1083; *Mining Magazine*, Vol. V (1911), p. 232; *Canadian Mining Journal*, Vol. XXXII (1911), p. 345; *Mining and Scientific Press*, Vol. CII (1911), p. 785.

[2] C. C. Semple, *Engineering and Mining Journal*, Vol. XCI (1911), p. 661.

South side mills. Pulp from the last regular mill plates, containing 2795 parts solids to 31,457 parts water, flows in a launder at 1. 5-per cent. grade to (1).

1. Fourteen settling cones 4 ft. diameter 5.5 ft deep with sides sloping 70 degrees. Feed assays $1.32 per ton in gold and the spigots $2.08 per ton. Spigots to (4); overflow to (2).
2. Dewatering tank, 17 ft. diameter and 12 ft. deep. Spigot to cyanide plant; overflow to (3).
3. Two clarifying tanks with cone bottoms each 18 ft. diameter and 20 ft. deep with bottoms sloping 45 degrees. Spigots to cyanide plant; overflow used as clear water for (8).
4. Fourteen classifying cones 1.33 ft. diameter and 3.78 ft. deep with sides sloping 80 degrees. Rising water currents add 1095 parts of water to the pulp. Spigots divided to (5) and (8); overflow to cyanide plant.
5. Dewatering cone 3 ft. diameter, 2.5 ft. deep with sides sloping 60 degrees; 484 parts of water are added. Spigot product with 70 parts solids to 49 parts water and assaying $2.45 per ton solids, to (6); overflow amounting to 581 parts water joins discharge of (6).
6. Tube mill, 5 ft. diameter, 14 ft. long, making 27 r. p. m. To (7).
7. Amalgamated copper plate 12 1/2 ft. wide and 18 1/2 ft. long set at a grade of 12 1/2 per cent. About $0.58 in gold is recovered by this plate equal to 1.1 per cent. of the gold in the feed to (1). Plate tailings to cyanide plant.
8. Seven Wheeler grinding pans, 5. ft. diameter, 2 ft. deep, 58 r. p. m. Feed contains 140 parts solids to 292 of water and assays $2.70 gold per ton solids. Also 969 parts water are added from (3) and the pan discharges contain 140 parts solids to 1261 parts water. To (9).
9. Seven sets of narrow amalgamated plates arranged in series of steps. Each set is 6 ft. long and has an area of 27 sq. ft. About $0.66 in gold is recovered per ton of dry pulp passing over these plates. The gold recovered amounts to 2.5 per cent. of all the gold in the original mill tailings fed to (1). Plate tailings go to cyanide plant.

The gold in the average mill tailings is about $3689 daily of which $133 or 3.6 per cent. is recovered by amalgamation after regrinding. The average cost of stamp milling per ton is $0.219 and $0.015 for regrinding or $0.30 for each ton actually reground.

Liberty Bell Mill in Colorado.[1]—This mill is described in "Richard's Ore Dressing," Vol. IV, p. 1631. The present flow sheet shows considerable modification. They now stamp in cyanide solution and concentration has been put in both for the stamp pulp and for the tube mill pulp. The battery pulp goes over plates to four Richard's vortex hindered settling classifiers yielding spigots to nine Wilfley tables and overflow after thickening to seven Wilfley tables. Two Wilfley tables retreat the middlings of these sixteen tables after the coarse sand has been removed by a Bunker Hill screen. The reground tube-mill pulp passes over plates to ten Deister tables. All the pulp freed from concentrates is treated in Dorr thickeners, Hendryx agitators and Moore filters. The canvas plant has been abandoned. The original article gives details and costs.

Gold Milling at Cripple Creek, Colorado.[2]—The Stratton Independence Mill recently built at a cost of over a quarter of a million dollars has a capacity of 300 tons per 24 hours. It aims at concentration of the low-grade dump ore to remove a small quantity of refractory sulpho-telluride mineral followed by cyaniding of the residue. See flow sheet p. 325.

The dump being treated contains a million tons averaging $3.25. The mill also receives one-tenth its supply from a small breaker plant

[1] C. A. Chase, *Bulletin American Institute Mining Engineers*, No. 60, December (1911), p. 981; *Mining and Scientific Press*, Vol. CII (1911), p. 840; *American Metallurgical Society*, Vol. I (1911), p. 30; *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911), p. 189.
[2] *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 192; *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XI (1911), p. 629; P. Argall, *Mining Magazine*, Vol. V (1911), p. 361; *Mines and Minerals*, Vol. XXXII (1911), p. 319.

where low-grade mine ore is crushed to about 1/4 in. and sent direct to Chili mills.

The breakers run only 4 hours out of 24 and the rolls only 10 hours out of 24. The division of sand and slime is 40 to 50 per cent. of the former with 50 to 60 per cent. of the latter. The concentrating department yields sand concentrates assaying 5 to 7 oz. gold per ton and the sand tailings $1 per ton; the slime tailings assay $2 per ton. Deister tables are preferred since they have one-tenth the maintenance cost of vanners and do just as good work.

New Mill at Nevada Wonder Mine.[1]—This new gold mill treats a hard oxidized quartz carrying a small percentage of high-grade sulphides and both gold and silver. The gold is mostly free while the silver in the sulphides is in the form of argentite and stephanite and some native silver. Preliminary tests indicated that the proper treatment is to crush to 200 mesh or finer, remove the greater portion of the sulphides and native silver by concentration and cyanide the tailings.

The ore is brought in three car-trains a distance of 300 ft. from a 300-ton storage bin and passes over a track scale to (1).

1. Receiving bin. To (2).
2. Shaking grizzly feeder having slow forward stroke and quick backward downward stroke. Stroke adjustable between 0 and 4 in. Oversize to (3); undersize to (4).
3. Blake breaker, 16×10 in., equipped with safety shearing toggles, crushing to 1 3/4 in. To (4)
4. Trough belt conveyor, 14 in. wide. To (5).
5. Mill bin. By two Challenge feeders to (6).
6. Ten stamps, weighing 1400 lb. each, 96 drops per minute; Canda self-tighting cams; concrete foundation with battery posts set in cast-iron sockets and secured to foundation by heavy tie bars. Mortar weighs 12000 lb. and has interlocking steel liners. Screen is 4 mesh. A grinding and mixing mill for supplying lime solution is placed above the stamp level. The ore is crushed in weak cyanide solution and goes to (7).
7. Spitzkasten. Spigot to (8); overflow to (9).
8. One 6-ft. Chili mill grinding in weak cyanide solution through 30 mesh screen. To (9).
9. Dorr duplex classifier. Sand to (10); slime finer than 200 mesh to (11).
10. One 5×22-ft. tube mill with scoop feed and ribbed lining, grinding in weak cyanide solution. Product by either of two bucket elevators to (9).
11. Two 8-ft. Callow tanks. Spigots to (12); overflow to (13).
12. Six No. 3 Deister tables. Concentrates to smelter; tailings to 13.
13. Dorr thickening tank. The thickened slime has about 2 tons of solution to 1 ton of ore and goes to (14); clarified overflow flows to sump tank and is thence lifted by single acting triplex pump to weak solution tank for reuse.
14. Sump. By single acting triplex slime pump to (15).
15. Slime collecting tank with mechanical agitator. By gravity to (16).
16. Four 15×45-ft. treatment tanks. Product agitated and aerated for 48 hours in strong cyanide solution and then discharged to (17).
17. Stock tank with mechanical agitation. To (18).
18. Two Oliver continuous slime filters. Solution to (19); residue to waste.
19. Two gold solution tanks, used alternately. To (20).
20. Five seven-compartment zinc boxes using zinc shavings. Precipitate to (21); barren solution to sump tank and then returned to single acting triplex pump to strong-solution tank for reuse.
21. The gold and silver precipitates are washed in two clean-up tanks, decanted, dried and partly roasted in a double muffle oil-fired furnace and finally melted with fluxes in two Faber du Faur furnaces.

The mill site is on a slope averaging about 22 degrees. Bed rock is found about 5 ft. below the surface. There are nine benches or floor levels in the mill proper. The retaining walls and foundations are of concrete. The mill building is 250 ft. long and 120 ft. wide and is covered with galvanized currugated steel.

Electricity is brought about 125 miles to the mill and is used for

[1] P. E. Van Saun, *Engineering and Mining Journal*, Vol. XCI (1911), p. 520; *Mining Magazine*, Vol. IV (1911), p. 307; *Mining World*, Vol. XXXV (1911), p. 959; *Los Angeles Mining Review*, June 24, 1911, p. 10.

lighting and also for running the motors of which there are seven aggregating 245 rated horse-power. An oil-fired locomotive boiler is used for heating the solutions to facilitate extraction and for keeping the building warm in winter. Water is piped 10 3/4 miles in 6-, 5-, and 4-in. steel pipe to a 100,000 gallon storage tank above the mill.

Nevada Hills Mill.[1]—The mine contains a large tonnage of oxidized silver ores containing manganese dioxide so combined with the silver as to make a combination which does not yield to cyanide treatment and concentration must be adopted to remove this mineral. When the oxidized ores are exhausted concentration can probably be abandoned. The mine ore is dumped from skips to (1).

1. Bin holding 150 tons. By gate to (2).
2. Grizzly. Oversize to (3); undersize to (4).
3. Hercules breaker, 20×12 in. Run only 8 hours out of 24. To (4).
4. Bucket elevator, 12 in. wide. To (5).
5. Sampling plant containing Hercules breaker, set of rolls, three Vezin samplers, and 4-in. belt and bucket elevator delivering reject to automatic weighing machine and thence to (6).
6. Ore bin holding 600 tons. By four Hutchinson feeders to (7).
7. Twenty stamps, each weighing 1250 lb. Four mortars weigh 12,000 lb. each and each is anchored to reinforced concrete foundation by seven 1 3/4-in. bolts. Screens are 20 mesh. To (8).
8. Classifiers. Products to (9).
9. Six No. 2 Deister sand tables and four No. 3 Deister slime tables. Concentrates to (24); tailings to (10).
10. Two 45-in. Akins classifiers. Sands to (11); overflow to (15).
11. Two 18 by 5-ft. tube mills. To (12).
12. Two 18-inch belt and bucket elevators. To (13).
13. One 36-inch hydraulic classifier. Spigot to (14); overflow to (15).
14. Four No. 3 Deister slime tables. Concentrates to (24); tailings to (10).
15. Two continuous thickening tanks 34 ft. diameter with 12-ft. staves, supplied with overflow launders and heavy Dorr thickeners. Thickened pulp to mixing box where proper amount of lime is added and thence to (16); overflow by 6×8 Gould triplex pump to 50,000-gallon tank near top of mill.
16. Nine agitation or treatment tanks in series 12 ft. diameter with 32-ft. staves. Standardized cyanide solution added to first tank and ore is under agitation for 42 hours in passing through the whole nine tanks. Air for agitation and aeration from a 13×10-in. low pressure Ingersoll-Rand compressor. Pulp to (17).
17. Three dilution tanks in series, 34 ft. diameter, 12 ft. staves, supplied with overflow launders and Dorr thickeners. Barren solution from zinc presses is added diluting solution 3:1 in each tank. Thickened pulp from each tank is drawn off at the bottom and returned in opposite direction by diaphragm pumps. The ore from the last tank deprived of 95 per cent. of its values by the dilution goes to (18); rich solution overflowing from last tank goes to (20).
18. Stock tank 30 ft. diameter, 12-ft. staves, equipped with Trent agitator. To (19).
19. Two Oliver continuous filters 11.5 ft. diameter and 12-ft. face. Washed residue to waste by 14-in. belt conveyor 200 ft. long inclined 20 3/4 degrees. Solution to (20).
20. Two 36-in. 40-frame Perrin clarifying presses. Residue to waste; solution to (21).
21. Three precipitation tanks 15 ft. diameter, 8-ft. staves, supplied with Merrill zinc dust feeders. To (22).
22. Two 52-in. 16-frame Merrill presses. Product dried and melted in Faber du Faur furnace producing 5 tons silver bullion per month. Barren solution to (23).
23. Two solution tanks 30 ft. diameter, 10-ft. staves. Solution drawn off and standardized for re-use as needed in a tank 34 ft. diameter with 12-ft. staves.
24. Concentrates are reground in 5-ft. continuous grinding pan, amalgamated and discharged alternately into two filter bottom tanks 6×20×3 ft. each, thence to a steam drier and sacked for shipment.

The stamps have two 35 horse-power motors, the tube mills each have a 75 horse-power motor, the compressor a 50 horse-power motor and the cyanide plant a 100 horse-power motor. The solutions and the mill are heated by two 100 horse-power, oil burning boilers which also furnish steam to operate an auxiliary electric plant of 100 horse-power for mine pumps in case of failure of electric power. Mine water is supplied for milling purposes. The mill is 308 ft. long, 56 ft. wide at the breakers, 84 ft. at the tube mills and 112 ft. in the cyanide depart-

[1] J. B. Fleming, *Engineering and Mining Journal* Vol. XCI (1911), p. 1001.

ment. The mill site slopes 17 1/2 degrees and has a fall of 120 ft. in eight steps. The mill building is of steel with walls of asbestos-covered corrugated iron.

Vulture Mill, Arizona.[1]—The special features of this new amalgamation and cyanide mill are gasoline engines for power, heavy stamps weighing 1500 lb. each, 5-ft. Australian grinding pans for regrinding, and the continuous decantation process for cyaniding using Dorr thickeners and Oliver filters. Concentrating tables are being installed to remove the concentrates from the ore coming from the lower levels of the mine as these concentrates do not give a good extraction by cyanide.

North Washington Mill at Republic, Washington.[2]—This new 1000-ton mill for cyaniding a gold ore is noteworthy since it replaces stamps with Chili mills. Briefly the system is as follows: (1) Grizzly and Blake breakers to 1 1/2 in. (2) Trommel and rolls to 1/2 in. (3) Trent Chili mills to 16 mesh. (4) Akins classifiers and tube mills to practically 200 mesh. (5) Trent agitators, disc thickeners, and Oliver filters.

Milling at Porcupine, Ontario.[3]—At the Dome mill the ore is quartz and greenstone schist, carrying free gold and iron pyrite. Much of the gold in the pyrite is free milling so that by grinding until 90 per cent. passes a 200-mesh screen, 84 per cent. of the gold is recovered by amalgamation. The flow sheet is given on page 328.

The Hollinger mill crushes in cyanide solution in stamps and tube mills and uses no amalgamated plates. The pulp is concentrated on Deister slime tables and the concentrates are ground in amalgamating pans. Tailings from these pans join the tailings from the Deister tables and go to the cyanide department which contains Dorr thickeners, Trent agitators, Moore filters and Merrill zinc dust precipitation.

Santa Gertrudis Cyanide Mill in the Pachuca District.[4]—There are two mills for treating silver ore; the old mill which uses old style Chili mills and tube mills and cyaniding and is still running, and the new 60-stamp mill. A flow sheet is given on page 329.

Gold Milling at Ouro Preto, Brazil.—Two articles have appeared, one[5] describing the Passagem mill and the other[6] the Ouro Preto mill. In both cases the gold is refractory and not easily amalgamated so that after stamping the pulp is concentrated on blanket tables and vanners. Free

[1] C. A. Dinsmore, *Mining World*, Vol. XXXV (1911), p. 645.
[2] H. W. Newton, *Mining World*, Vol. XXXIV (1911), p. 447; M. H. Joseph, *Mining and Scientific Press*, Vol. CIII (1911), p. 43.
[3] *Mining and Scientific Press*, Vol. CII (1911), pp. 363, 694; *Mining Magazine*, Vol. V (1911), p. 73; *Canadian Mining Journal*, Vol. XXXII (1911), pp. 126, 356; A. Gray, *Mining World*, Vol. XXXIV. (1911), pp. 257, 785; *Mining Journal*, Vol. XCII (1911), p. 80; L. D. Huntoon, *Canadian Mining Journal*, Vol. XXXII (1911) p. 811; P. H. Moore, *Canadian Mining Journal*, Vol. XXXII (1911) p. 560.
[4] W. Weston, *Engineering and Mining Journal*, Vol. XCII (1911), p. 109; *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911), p. 142.
[5] A. J. Bensusam, *Institution Mining and Metallurgy*, Vol. XX (1911), p. 3.
[6] R. R. Kendall, *Institution Mining and Metallurgy*, Vol. XX (1911), p. 28; *Mining Science*, Vol. LXIII (1911), p. 12.

gold is recovered from the concentrates by bateas. The concentrates residue is cyanided and the tailings classified and the sands and slimes cyanided separately. Full details are given in the original articles.

Brakpan Mill, Transvaal.[1]—Coarse ore from both No. 1 and No. 2 shafts is collected in (1). Fine ore goes through hoppers and belt-conveyor to (5).

CRUSHER STATION

1. Coarse ore bin. By five conveyors to (2).
2. Five trommels. Oversize to (3); undersize by five conveyor belts to (5).
3. Picking belts. Waste removed and delivered to six parallel conveying belts leading to main waste conveyor; also suitable material for pebbles in the tube mills is picked out and goes by car to tube mills; residue to (4).
4. Fine breakers having dust removal system. By belt conveyor to (5).
5. Main fine ore conveyor rising to top of mill. By transverse distributing belt conveyor to (6).

MILL

6. Bin holding 6000 tons. By suspended automatic feeders to (7).
7. 160 stamps, each weighing 2004 lb. Heads are 9.5×42 in. and weigh 788 lb.; stems 14.5 ft. × 4.5 in., 696 lb.; tappet 232 lb.; shoes 14 in. long, 9.25 in. diameter, 288 lb. One motor drives 10 stamps. Coarse eliminating screens are used to the back of the mortars and the mortars are of the open front type. Pulp by two Robeson-Davidson coarse sand pumps through pitch-lined launders to (8).
8. Six Caldecott diaphragm cones. Thick pulp with 27 per cent. moisture to (9); overflow to (13).
9. Six Allis-Chalmers tube mills 5.5×22 ft., fitted with Schmitt spiral feeders. Three are right hand and three are left hand. Water added to bring pulp up to 35 per cent. Six-inch silex lining. Spur gear driver at outlet end. To (10).
10. Six sets of stationary amalgamating plates with five 5×12 ft. plates in series in each set, 18-per cent. slope. To (11).
11. Six amalgam traps followed by one trap in the main launder. Pulp to (12).
12. Two Robeson-Davidson rotary pumps. To two of the classifiers (8).
13. Cyanide department containing Caldecott cones, three 60-ft. sand collecting tanks, six sand treatment vats, four slime settling tanks, six Pachuca tanks, Butters' vacuum filter plant, Merrill filter presses and refinery.

The refinery also contains three clean-up barrels for amalgam and black sands followed by batea amalgamation plate, settling tank, amalgam press and retort.

The plant is laid out so as to be enlarged to 200 stamps with 10 tube mills and other necessary additional apparatus. With 160 stamps the capacity is 60,000 tons per month and this may be increased to 80,000 or 90,000 tons.

Modderfontein B Mill.[2]—This new mill on the Rand will have 80 stamps weighing 1750 lb. each and crushing 20 tons per stamp per day through a 3-mesh screen. All ore finer than 3-mesh is screened out and goes to tube mills witohut passing through the mortars.

Special features of the mill are: (1) Absence of battery plates, (2) auxiliary cone classifier before tube mill, (3) small cone for tube mill discharge, (4) stationary amalgamating plates after the tube mill, (5) small hydraulic cone classifiers in series for returning coarse sand to tube mills, (6) hydraulic classifiers for slime separation, (7) vacuum filter

[1] H. S. Gieser and C. A. Tupper, *Mining and Scientific Press*, Vol. CIII (1911), p. 796; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 672; *Engineering and Mining Journal*, Vol. XCII (1911), p. 753; *Mining World*, Vol. XXXV (1911), p. 699; *South African Mining Journal*, Aug. 12, 1911, p. 979.

[2] *Mining Magazine*, Vol. IV (1911), p. 85.

followed by zinc dust precipitation, (8) Homestake method of cupelling the briquetted gold precipitate.

Globe and Phoenix Mill in Rhodesia.[1]—The ore is hard white quartz containing small amounts of pyrite and stibnite. The latter is particularly harmful since it slimes badly in concentration and even only 1 per cent. prevents a good extraction in cyaniding. Generally about 75 per cent. of the gold is free and there is also some rusty gold which may be recovered in amalgamating pans. A fortunate discovery was that if the slime from this ore was run direct without treatment to the dams and allowed to weather or oxidize for about 6 months and then broken up and pumped to the slime plant an extraction of 75 or 80 per cent. could be obtained. Sands cannot be weathered in this way. The new plant started in 1910 is briefly as follows, omitting bins, conveyors, etc.

1. Blake breaker. To (2).
2. Wash trommel with 1 1/2- and 1/4-in. holes. Oversize to (3), undersize to (4).
3. Picking belt where 0.5 to 1 ton massive stibnite is removed daily. Residue to (4).
4. Forty stamps with 25-mesh screens. To (5).
5. Amalgamated plates. To (6).
6. Blanket tables. Rusty gold caught and amalgamated in a barrel. Tailings to (7).
7. Classifiers. Sand to (8); slime to (10).
8. Record vanners. Concentrates to (13); tailings to (9).
9. Ten grinding and amalgamating pans. To (10).
10. Blanket table. Concentrates to (13); tailings to (11).
11. Spitzkasten. Sands to sand leaching plant; slimes to (12).
12. Weathering ponds where slime stands for 2 or 3 months and is plowed and harrowed so that it may oxidize sufficiently to go to the cyanide plant for slimes.
13. Merton roasting furnace. To (14).
14. Grinding and amalgamating pans. To (15).
15. Blanket tables to catch rusty gold and floured mercury. Tailings to cyanide plant for concentrates.

In January 1911, 6539 tons were milled with the following recoveries and losses:

Source.	Ounces per Ton Milled.	Percentage per Ton Milled.
Stamps and pans	1.447	79.86
Concentrates	0.121	6.65
Sands	0.019	1.08
Slimes	0.107	5.91
Total extracted	1.694	93.50
Concentrates residue	0.010	0.59
Sand residue	0.080	1.49
Slime residue	0.027	4.42
Total lost	0.117	6.50

The entire cost of milling including repairs and maintenance was $3.18 per ton.

Milling Practice at Komata Reefs Mine, New Zealand.[2]—The scheme of treatment is to crush by breakers and stamps to 4 mesh, regrind in

[1] H. T. Brett, *Mining Magazine*, Vol. V (1911), p. 51; *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911), p. 67; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 466.

[2] S. D. McMiken, *Mining and Scientific Press*, Vol. CIII (1911), p. 4.

tube mills, classify and leach the sand in tanks and agitate the slime in Pachuca tanks which are followed by Dorr settler and Moore filter. The original article contains full details of the plant and its operation.

Bibliography of Gold Milling

The reader is referred to the bibliography given at the end of the article on "Progress in Ore Dressing and Coal Washing" in this volume.

GRAPHITE

The production of natural graphite in the United States during 19 was approximately 3618 short tons.[1] One-third of this was amorpho graphite and graphitic shale. The production of crystalline graph was about 2395 tons in 1911, and a large number of mills were id Comparing this with the imports which for 1911 amounted to 20,7 tons valued at $1,495,729, it shows that these also have fallen off co siderably. For the latter half of 1911 there were also imported man factures of graphite amounting in value to $72,752. It is impossible give a price for graphite on account of the many grades and also t market fluctuations. Domestic crystalline graphite is quoted as l as 3 to 8 cents per pound according to grade, the average price of t entire production in the United States being 5 cents per pound f 1911 as compared with 5 1/10 cents per pound in 1910. The large producers make manufactured products so the prices depend largely the demand for these. The import values for 1911 averaged 3.62 cen per pound. The London markets at the beginning of 1912 were follows per long ton c. i. f. London:

As imported:

Ceylon LL	£17 5s. 0d. to £40 10s. 0d. per ton c. i. f. London.
Ceylon OL	£12 9s. 6d. to £41 9s. 6d. per ton c i. f. London.
Ceylon chips	£10 5s. 0d. to £31 0s. 0d. per ton c. i. f. London.
Ceylon dust	£11 10s. 0d. to £29 0s. 0d. per ton c. i. f. London.

Purified, milled and ground

Ceylon 97 to 99 per cent	£59 0s. to £63 0s. per ton f. o. b. London.
Ceylon 90 to 91 per cent	£40 0s. to £42 0s. per ton f. o. b. London.
Ceylon 80 to 81 per cent	£30 0s. to £32 0s. per ton f. o. b. London.
Ceylon 70 to 71 per cent	£27 0s. to £28 0s. per ton f. o. b. London.
American large flake	£45 0s. to £49 0s. per ton f. o. b. London.
American small flake	£35 0s. to £45 0s. per ton f. o. b. London.
Joint compound	£ 2 9s. 0d. to £2 12s. 6d. per cwt.
Paint Paste	£ 2 12s. 6d. to £2 15s. 0d. per cwt.
Paint	£ 0 6s. 3d. to £0 6s. 9d. per gallon.

Most of the graphite imported into the U. S., 13,119 tons, came fro Ceylon at an average value of 4.32 cents per pound; Mexico 3368 to average 3.77 cents; Canada 2289 tons at 1.73 cents; Korea 1256 to at .73 cents per pound; Italy 510 tons at .61 cents; Austria-Hungary tons at .8 cents; Germany 48 tons at 3.25 cents per pound.

A feature of the graphite industry is the falling off in the trade f flake graphite there being more material produced for lubricants, et than there is a demand for. The result is a low price. Crucible fla at works was six cents per pound. The production of artificial (v

[1] E. S. Bastin, advance chapter Mineral Resources, U. S. G. S., 1911.

manufactured) graphite is given in the table. It is somewhat less than in 1910, but there probably will be an expansion of this industry in the United States in 1912.

STATISTICS OF GRAPHITE IN THE UNITED STATES.

Year.	Refined Crystalline Graphite.						Amorphous Graphite Production.		Artificial Graphite. Production.	
	Production.		Imports.		Consumption (a).					
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Tons. 2000 lb.	Value.	Pounds.	Value.
1898	1,647,679	$ 82,385	30,199,680	$ 743,820	31,847,359	$ 826,205	1,200	$ 11,400	185,647	$ 11,603
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953	1,030	8,240	405,870	32,475
1900	4,103,052	164,122	32,298,560	1,389,117	36,401,612	1,553,239	1,045	8,640	860,750	68,860
1901	3,967,612	135,914	32,029,760	895,010	36,997,372	1,067,921	809	31,800	2,500,000	119,000
1902	4,176,824	153,147	40,857,600	1,168,554	45,034,424	1,322,401	4,739	55,964	2,358,828	110,700
1903	4,525,700	164,247	32,012,000	1,207,700	36,537,700	1,371,947	16,591	71,384	2,620,000	178,670
1904	4,357,927	162,332	25,350,000	905,581	29,707,927	1,067,913	19,115	102,925	3,248,000	217,790
1905	4,260,656	170,426	34,914,611	983,034	39,175,267	1,153,460	(b)21,953	80,639	4,595,500	313,979
1906	4,894,483	170,866	50,974,336	1,554,212	55,868,819	1,725,098	(b)16,853	(c)	4,868,000	312,764
1907	4,586,149	149,548	40,962,000	1,777,389	45,548,149	1,926,937	(b)26,962	138,381	6,924,000	483,717
1908	3,433,039	149,763	22,912,714	762,267	26,345,753	876,030	(b) 1,433	75,250	7,385,511	502,667
1909	5,669,899	340,194	42,532,851	1,854,459	48,202,750	2,194,653	(b) 5,096	32,238	6,870,529	467,196
1910	5,625,132	286,882	50,610,560	1,872,592	56,235,692	2,159,474	(b) 1,407	39,710	13,149,000	945,000
1911	4,790,000	256,050	41,404,000	1,495,729	46,194,000	1,751,779	1,223	32,415	10,144,000	664,000

(a) Neglecting the small re-export of foreign product. (b) Statistics of the U. S. Geological Survey. Largely graphitic shale. (c) Not reported.

Alabama.—In Alabama the production during 1911 was small. The principal producer, as heretofore, was the Ashland Graphite Company, successor to the Enitachopco Graphite Company whose quarry and mill are located 4 1/2 miles west of Ashland. A new mill is under construction. Another producer is the Allen Graphite Company located 8 miles west of Ashland. This plant was idle more than half the year and was acquired by the American Flake Graphite Co. A description of the Allen Graphite Company's mill at Graphite, Ala., is given in the "Mineral Resources of the United States," 1910, United States Geological Survey, by Edson S. Bastin.

The rock, after passing large gyratory crushers, is screened in a revolving screen, the oversize being re-crushed in smaller crushers, the product of which is added to the undersize and passed to the cylindrical rotary dryer. After drying it is crushed in rolls and screened into two sizes, one over 5-mesh which is crushed and added to that which is less than 5-mesh. It then goes to the water flotation separators, the filtered product going to 90-mesh rocking screens. After drying on floors heated by exhaust steam, it then goes to Robinson gyratory sifters separating it into three sizes which go to separate Hooper pneumatic jigs from which the middlings are returned and the tailings pass back to the water flotation process. The fines are ground in burrstone mills and then rescreened on gyratory screens and bolting reels into the final product.

California.—No Californian graphite was shipped in 1911 but some prospecting was done 50 miles north of Los Angeles. The mine belonging to the Dixon Graphite and Milling Company at Point Arena was not working. The total graphite that was produced in the State was marketed in 1901 and 1902 and amounted to only 106 tons. The owners of a mine in Sonoma County are preparing to resume work in 1912.

Colorado.—In Colorado, the Federal Graphite Company was the only one operating. Its amorphous production was shipped to the mill at Warren, Ohio, for grinding.

Georgia.—The producers in Georgia are the Emerson Brick Company and the American Chemical and Mining Company, both mining graphitic slate or amorphous graphite which is not concentrated but simply ground and shipped. The value of this material is given as $1.25 per ton f. o. b. works.

Maine.—The origin of natural graphite in Maine where there are two deposits, is given by George O. Smith[1],

Montana.—The Crystal Graphite Company continued development work and some shipments on its property near Dillon, Montana. The deposits are chiefly on a ridge between Van Camp Creek and Timber-gulch Creek near the southwestern end of the Ruby range.[2]

Michigan.—The production of this State comes from near L'Anse. Most of this material is graphitic slate used in paint manufacture and shipped by the Detroit Graphite Company. A mill is also used by the Northern Graphite Company, the material being carefully sorted and treated in pneumatic jigs. In 1911 the Detroit Graphite Company made the largest production of any season before snow prevented operation. The graphite will be hauled in sleighs to the railroad. The output in 1911 was 2700 tons. It is probable that this deposit will not be exhausted for a number of years.

New York. (By D. H. Newland.)—No noteworthy advances were recorded for the graphite industry in New York State during 1911. The production amounted to 2,510,000 lb., about the usual quantity, and represented a value of $137,750. The total for the preceding year was 2,619,000 lb., with a value of $160,700. Prices appeared to be somewhat lower; the reported average was about 5.5 cents a pound, as compared with 6.1 cents in 1910.

The American mine at Graphite, owned by the Joseph Dixon Crucible Co., continued as the leading producer. This mine has long been the largest and most successful of the kind, not only in the State but in the country as well, and may be considered the pioneer enterprise in all that

[1] *Bull.* 285, United States Geological Survey, 1906, page 480.
[2] A. N. Winchell, *Econ. Geol.*, April–May, 1911.

relates to the technology of treating the disseminated flake graphite which constitutes the principal source of domestic production. The methods of extracting and refining the graphite as developed by its management have seldom been applied elsewhere with similar results, owing in some measure undoubtedly, to the unusually favorable natural conditions found at Graphite. The ore is a quartzite carrying flakes of graphite distributed along the cleavage planes. The flakes are relatively of large size, showing the appearance of having been squeezed out by regional compression, and measure up to 1/4 in. in diameter. The average content in graphite may be placed at about 6 or 7 per cent. What is most important to the success of the milling operations is the practical absence of micaceous minerals which are more or less common in the graphite schists and quartzites of the Adirondacks. When present in any amount a high-grade graphite product cannot be expected. The Empire Graphite Co., owning mines in the town of Greenfield, Saratoga County, four miles west of Kings, was active during a part of the year.

The Saratoga Graphite Co. has lately erected a mill near Kings Station north of Saratoga Springs.

The other properties in the eastern Adirondacks that have been active during the last few years include the Conklingville mine of the Sacandaga Graphite Co. and the mine near Chilson Lake, owned by the Crown Point Graphite Co.

A small quantity of graphite has been shipped recently by the Macomb Graphite Co. from its property near Popes Mills, St. Lawrence County.

Pennsylvania.—The Rock Graphite Mining and Manufacturing Company did not operate. The plant of the American Flake Graphite Company at "Phoenix Hill" was being dismantled during 1911 and the outlook for flake graphite which is the chief production near Chester Springs, is not very good. A description of the Chester Graphite Company's mill was given in Vol. XIX of the MINERAL INDUSTRY, page 347. Pettino Bros. again operated their plant in 1911 at Byers.

Virginia.—A description of a number of localities in the Piedmont region of the Blue Ridge is given by Thomas L. Watson.[1] Some years ago the Naylor-Bruce Graphite Company opened a mine two miles west of Buck Mountain in the north of Albemarle County but it was soon abandoned.

Wisconsin.—The material produced by the Wisconsin Graphite Company is milled near Stevens Point. The product is crushed and ground in tube mills and air floated. Its chief use is as a pigment and it is worth from 1 to 2 cents per pound.

[1] Mineral Resources of Virginia, 1907.

Wyoming.—Graphite is found in Laramie, Albany, Carbon and Fremont Counties. The largest district is known at Plumbago Canyon. There is no crystalline graphite and there has been no mine worked for graphite in the State.[1]

Graphite in Foreign Countries

Africa.—Although results were anticipated from the work done by the Southern Developments Co., Ltd., during the year, no details are at hand. The property lies in the Umzimkulu River region in Natal where there are cliffs that are 500 ft. high. Test shipments were made to Europe and all samples proved of highly payable quality. The flake is too small for crucibles but is said to be suitable for all other uses and is estimated at £3 to £5 per ton in value. The same product is being sold in Johannesburg from £40 to £50 per ton. Some of the deposits of this material are reported in the *South African Mining Journal*, Feb. 25, 1911.

Austria.—The production of graphite according to O. Stutzer[2] is increasing and is greater than that of Ceylon. The output in 1910 is reported as 33,131 metric tons about one-third each being from Bohemia, Stryia and Moravia. In 1909 Austria produced 40,710 tons worth £65,815. The Ceylon production for the same year was 32,710 tons worth £665,314. From these values it can be seen that the Austrian production is of inferior grade to that of Ceylon.

Canada.—The graphite produced in Canada in 1911 amounted to 1269 long tons valued at $69,576. The exports for 1911 amounted to 16,263 cwt. valued at $43,249. The principal production comes from Quebec and during the year the Dominion Graphite Company on the Stewart property near Buckingham started operations and substantial shipments were made. Other producers of graphite that made returns in 1911 are the Bell Graphite Company, the North American Graphite Company and the Peerless Graphite Company. The first of these is said to have a capacity of five to six tons of finished product per day. Its mill, which can handle 70 tons of mill rock per day, was completed in 1910. In 1911 Quebec shipped 753,045 lb. worth $33,613.

Ceylon.—Most of the graphite used in the United States comes from Ceylon where the production has been steadily increasing from about 1830 until now it has reached 40,000 long tons per year. In 1902 the exports to the United States reached a maximum of 15,244 tons. In 1911 the exports to the United States were 12,062 tons, to Germany 8572, to Great Britain 5794, and to Belgium 3108 tons. The principal importers

[1] Mineral Resources of Wyoming, 1911. *Bull.* 1, Series B.
[2] *Zeit. f. prak. Geol.* 1910, Vol XVIII.

of Ceylon graphite are Pettinos Brothers of Bethlehem, Penna. It is estimated that 50,000 persons are employed in the industry including the mining, sorting, curing, packing and shipping. The deepest mines are located near Kurunegale. The Ceylon Blue Book for 1909 gives the relative output of the districts as follows:

District	Output in Long Tons
Ratnapura	7,351
Kegalla	4,041
Galle and Matara	5,026
Western	4,763
Kurunegale	2,432
Kandy	394
Total	24,007

A full description of occurrences and minerals is given by E. S. Bastin in "Mineral Resources of the United States for 1911."

Germany.—The graphite production comes from Bavaria which continued to be a large producer. The production of Germany in 1911 increased 50 per cent. and was 11,298 tons as compared with 7415 tons the preceding year.

India.—Most of the graphite produced in 1910, amounting to 3992 long tons, came from the Travancore mines. A small output also in 1910, namely, 259 tons, came from Vizagapatan. The latter material appears to be poor in quality, its value being stated as about $10 per ton as compared with $25 per ton for the Travancore material. The output for 1911 is not available but that for 1910 was fully one-third more than the preceding year.

Japan and Korea.—Most of the graphite that comes from Japan was imported and reexported from Korea and Ceylon. Japan had a production in 1910 of 145 tons compared with 121 tons in 1909. Altogether the average imports are 3,000,000 lb. of which two-thirds is exported.

The chief producing district in Korea (Chosen) is Northern Ping-An but the quality is not worth more than one-sixth of the price of the Ceylon graphite. According to Consul George Skidmore, Seoul, the value of exports in 1910 amounted to 56,874 tons; of this nearly all, or 55,729 tons, went to Japan. About 2000 tons were imported into the United States in 1911.

Mexico.—According to Consul Alexander V. Dye of Nogales, the exportation of graphite from Mexico for the fiscal year ending June 30, was as follows: for 1909, 1773 tons valued at $18,014; 1910, 2620 tons valued at $28,828, and for 1911, 3004 tons valued at $238,364. The graphite is the product of mines near La Colorada, Sonora, Mexico, which are owned and operated by an American company. An engineer's

report states that it occurs in beds which reach thicknesses of 9 to 10 ft., but are extremely irregular. Most of the mining is done with pick and shovel. An analysis of the average run of mine is as follows:—graphitic carbon 86.75 per cent., silica 7.60 per cent., iron, .65 and alumina 5.00 per cent. The graphite is apparently all amorphous. The graphite is simply sorted and dried in the sun and then it is ready for shipment.

WORLD'S PRODUCTION OF GRAPHITE.
(In metric tons.)

Year.	Austria.	Bavaria	Canada.	Ceylon. (a)	India.	Italy.	Japan.	Mexico. (b)	Sweden.	United States. (c)	Totals.
1897	38,504	3,861	395	19,275	61	6,650	391	795	99	450	69,350
1898	33,062	4,593	1,107	78,509	61	6,435	347	1,365	50	824	126,353
1899	31,819	5,196	1,188	29,037	1,548	9,990	53	2,305	35	1,648	80,960
1900	33,663	9,248	1,743	19,168	1,859	9,720	94	2,561	84	1,799	79,939
1901	29,992	4,435	2,004	22,707	2,530	10,313	88	762	56	1,800	74,688
1902	29,527	5,023	993	25,593	4,648	9,210	97	1,434	63	1,895	78,371
1903	29,590	3,719	660	24,492	3,448	7,920	114	1,404	25	2,053	73,435
1904	28,620	3,784	410	26,478	2,955	9,765	216	970	55	2,045	75,298
1905	34,416	4,921	491	31,134	2,361	10,572	209	970	40	1,933	87,047
1906	38,117	4,055	405	36,578	2,642	10,805	177	3,915	37	2,220	98,909
1907	49,425	4,033	525	33,027	2,472	10,989	103	3,202	33	2,080	105,889
1908	44,425	4,844	227	26,227	2,919	12,914	177	1,076	66	1,557	(d) 101,952
1909	40,710	(e)4,900	783	25,995	3,182	11,583	284	(e)1,100	26	2,572	91,135
1910	33,131	7,415	1,263	30,008	4,056	12,510	145	2,332	49	2,552	93,461

(a) The figures for 1897, 1899, 1907, and 1908 are exports; the enormous production in 1898 as reported in official government publications is not reflected in the exports of that year which amounted to 24,349 metric tons. (b) Exports. (c) Crystalline graphite. (d) Includes the production of 7,520 metric tons in Korea. (e) Estimated.

GYPSUM

The steady increase in the production of gypsum which marked the years 1908 to 1910 was not repeated in 1911. In New York, which is the largest individual producer, the output declined by about 4 per cent. The total production of gypsum during 1911 was 2,323,970 tons. The output of 1911 was shared by 17 states and Alaska. Of the total amount of crude gypsum mined in 1911, 52,880 tons were ground and sold as land plaster at $1.85 per ton; 334,600 tons were sold, without calcining, for use in portland cement, paint, etc., at $1.47 per ton; and 1,598,418 tons were converted into calcined plaster and sold at $3.67 per ton.

PRODUCTION OF CRUDE GYPSUM IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

State.	1909		1910		1911	
	Tons.	Value.	Tons.	Value.	Tons.	Value.
California, Nev., and Oregon	133,042	551,627	149,230	$695,121	(b)177,815	$650,319
Iowa	319,577	655,602	322,713	943,849	354,204	857,287
Kansas	137,697	321,921	135,088	377,222	122,579	345,203
Michigan	394,907	1,213,347	357,174	668,201	347,296	573,926
New York	403,929	1,032,650	467,339	1,153,977	472,834	1,199,596
Ohio and Va.	289,517	667,948	292,987	821,213	360,858	1,056,568
Tex. and Okla.	338,526	$771,137	351,347	941,256	288,278	773,753
Utah	(c)	.....	46,279	149,089	(d)	.....
Wyoming	43,745	132,719	(c)	.....	44,687	116,237
All other States	191,845	559,787	256,900	773,101	155,419	889,146
Total	2,252,785	$5,906,738	2,379,057	$6,523,029	2,323,970	$6,462,035

(a) Statistics of the U. S. Geol. Surv. (b) Includes Utah. (c) Included in Other States. (d) Included in Cal. and Nev.

CRUDE GYPSUM MINED IN UNITED STATES. (a)
(Tons of 2000 lb.)

1880	90,000	1892	256,259	1904	940,917
1881	85,000	1893	253,615	1905	1,043,202
1882	100,000	1894	239,312	1906	1,540,585
1883	90,000	1895	265,503	1907	1,751,748
1884	90,000	1896	224,254	1908	1,721,829
1885	90,405	1897	288,982	1909	2,252,785
1886	95,250	1898	291,638	1910	2,379,057
1887	95,000	1899	486,235	1911	2,323,970
1888	110,000	1900	594,462		
1889	267,769	1901	633,791		
1890	182,995	1902	816,478		
1891	208,126	1903	1,041,704		

(a) Statistics of the U. S. Geol. Surv. (e) Estimated.

Imports during the fiscal year ended June 30, 1911, included 342,528 long tons of crude and unground gypsum, almost entirely from Nova Scotia and New Brunswick, valued at $1.19 per ton, and 2045 long tons of ground or calcined, valued at $6.97 per ton. Under the latest tariff act, crude gypsum is taxed at 30 cents and prepared at $1.75 per ton.

Gypsum Mining in the United States

California.—Recent development of a gypsum deposit in the Maria Mountains in the eastern end of Riverside County was described by Gordon Surr[1] on the basis of a brief investigation.

The locality, 2 1/2 miles east-west, and 2 miles north-south, is situated 23 miles, by road, south of Blythe Junction, on the Santa Fe railway and 10 miles west of the Colorado River. Similar large deposits are known in the Palen Mts., 15 miles west of this locality. The deposits consist of a series of beds from 10 to 100 ft. thick, separated by limestones and shales. On the north the beds are nearly vertical, but on the south they dip 20 degrees to the north. To a maximum depth of 4 ft. the gypsum is porous, but below this level it is compact, finely crystalline and generally snow white. Such iron stains as do occur at the outcrop are found to disappear in depth. Microscopic examination of a typical sample from a depth of 5 ft. showed that anhydrite, quartz and clay were absent, and that the only impurity was 5 to 10 per cent. of calcium carbonate. Salt was entirely lacking. Calcining tests indicated that the gypsum was thoroughly suitable for making plaster.

The United States Gypsum Company, of Chicago, has secured 1000 acres in this locality and has instituted systematic exploration, by shafts and drill holes. Estimates of the available tonnage are inexact, but exceed several million tons. The locality is a desert but domestic water occurs within 5 miles. Timber and firewood are lacking, but with the building of a railroad from Blythe Junction, fuel oil could be obtained cheaply.

Nevada.—The Nevada-Douglas Copper Co., operating extensive copper properties in the Yerington district, has developed an important deposit of gypsum near its Ludwig mine. It is very conveniently situated with respect to the new Nevada Copper Belt Railroad, and shipments of 500 tons per week have been made, since early in 1912, to the Western Gypsum Co., at Reno.

New York (By D. H. Newland).—Most of the output was used for the manufacture of calcined plasters and with portland cement. There was a similar falling off in many other industries based on the production

[1] *Mining World*, April 15, 1911, p. 787.

of building and structural materials. It may also be said that the period of rapid expansion, marking the development stage of the gypsum industry, is probably over, whence progress henceforth will be slower and less constant than in the past.

The output, based on the crude rock mined or quarried in 1911, amounted to 446,794 short tons. The corresponding total for the preceding year was 465,591 short tons, showing a decrease of 18,797 short tons, compared with a gain of 87, 359 tons in 1910. With that one exception, the product of 1911 was the largest on record.

Of the total as given, about 70 per cent. was consumed by the local calcining plants operated in connection with the mines for the manufacture of plaster of Paris and wall plasters. The reports of these plants showed a production of 262,249 tons of calcined plasters with a value of $871,106. The outturn for 1910 was 250,228 tons valued at $848,340. The quantity of gypsum ground for land plaster was reported as 9959 tons valued at $18,508, against 12,597 tons valued at $28,100 in 1910. The sales of crude or lump gypsum, chiefly to portland cement works, accounted for 144,035 tons valued at $202,984, as compared with sales of 178,518 tons valued at $256,512 in the preceding year.

PRODUCTION OF GYPSUM IN NEW YORK.

	1910.		1911.	
	Short Tons.	Value.	Short Tons.	Value.
Total output, crude	465,591	.....	446,794	.....
Sold crude	178,518	$256,512	144,035	$202,984
Ground for land plaster	12,597	28,100	9,959	18,508
Wall plaster, etc., made	250,228	838,340	262,249	871,106
Total value	.....	$1,122,952	.....	$1,092,598

The output of gypsum and gypsum products, as given, was reported by 14 firms and was divided among the five counties of Onondaga, Cayuga, Monroe, Genesee and Erie. The greater quantity of rock was obtained in the western section where it found use mainly in the production of calcined plasters.

In Onondaga County there was less activity than usual. The Fayetteville Gypsum Co. produced most of the crude gypsum obtained in the county and shipped the output to New York City for calcination. The property operated by the company was the Severance quarry, near Lyndon, noted for the great thickness of the gypsum beds, which are exposed around the summit of a hill, and have a vertical extent of from 40 to 60 ft. They are worked by open-cut excavation, after first blasting

down the overlying shale and limestone, which are from 20 to 25 ft. thick. The lump gypsum is loaded on 20-ton wagons and hauled 2 miles by a traction engine to the Erie canal for shipment. The quarry formerly worked by the National Wall Plaster Co. in the same vicinity produced some gypsum which was ground in the local mills to land plaster.

The quarries at Union Springs, Cayuga County, were worked during the year by local interests, the lease under which they had been operated for several years by the United States Gypsum Co. having expired. Most of the output from this place has been used for land plaster and for portland cement. The gypsum ranges from 20 to 30 ft. thick and is worked by quarry methods.

In Monroe County, around Garbutt, the usual activity was manifest, though there was one less producer than in the preceding year. The Garbutt Gypsum Co., one of the pioneers in the district, closed down its mine and mill. The active companies were the Consolidated Wheatland Plaster Co., The Empire Gypsum Co., the Lyconing Calcining Co., and the Oatka Gypsum Co., the last named succeeding the Monarch Plaster Co. The gypsum occurs in two seams, each from 5 to 8 ft. thick, separated by from 6 to 12 ft. of limestone. Only the upper seam has thus far been attacked. The work is all underground, conducted through adits or shallow vertical shafts. About one-fourth of the output last year was marketed as crude or ground raw gypsum, the rest being converted into calcined plasters. Monroe County held second place in quantity and value of its products.

PRODUCTION OF CRUDE GYPSUM IN THE PRINCIPAL COUNTRIES. (a)
(In metric tons.)

Year.	Algeria. (b)	Canada.	France. (b)	Germany. (c)		Greece.	India.	United Kingdom.	United States.
				Baden.	Bavaria.				
1897	36,750	217,340	2,004,339	40,702	26,153	51	8,187	184,287	272,493
1898	37,337	198,864	2,115,261	28,037	25,688	83	8,390	199,174	285,644
1899	39,950	221,821	1,807,454	29,419	29,727	81	6,546	215,974	382,891
1900	42,237	228,656	1,774,492	26,381	35,484	129	4,415	211,436	439,265
1901	44,025	266,476	2,385,633	28,183	3,581	671	(d)	204,045	598,529
1902	44,975	301,165	2,185,346	33,150	31,701	*Nil.*	(d)	228,264	740,906
1903	41,550	285,242	1,998,804	29,423	30,894	94	(d)	223,426	945,285
1904	48,375	309,133	1,957,802	26,984	22,766	393	3,937	237,749	853,546
1905	34,743	395,341	1,378,145	28,823	46,247	185	4,877	259,596	982,626
1906	27,950	378,904	1,377,429	25,643	50,763	70	(e) 5,000	228,627	1,397,480
1907	26,400	431,286	1,316,567	29,153	48,975	70	(e) 5,000	239,285	1,564,061
1908	25,500	346,436	1,750,562	35,217	51,314	*Nil.*	(e) 5,000	231,980	1,694,155
1909	36,250	398,290	1,655,672	36,621	51,630	191	(e) 5,000	242,832	(f) 2,042,286
1910	60,625	481,941	1,980,804	41,078	54,397	249	6,678	259,648	(f) 2,158,756
1911	(d)	458,652	(d)	42,408	(d)	(d)	(d)	(e) 274,838	(f) 2,018,770

(a) From official reports of the respective countries, except the statistics for the United States. (b) A part of the product is reported as plaster of paris. In converting this into crude gypsum it has been assumed that the loss by calcination is 20 per cent. (c) Prussia is a large producer of gypsum, but there are no complete statistics available. The output in 1910 was 22,042 tons. (d) Statistics not yet available. (e) Estimated. (f) U. S. Geol. Surv.

The active mines in the western section were those of the United States Gypsum Co. and the Niagara Gypsum Co. near Oakfield, Genesee County, and of the American Gypsum Co., and the Akron Gypsum Co. near Akron, on the Erie-Genesee County boundary. Their output was consumed mainly in the manufacture of wall plasters by the plants located at the mines. The American Gypsum Co.,however, shipped most of its output to portland cement makers. The gypsum beds in this section are rather thin, averaging not more than 4 or 5 ft., but they are of high-grade character, well adapted for calcination. The mines are worked through vertical shafts in a manner similar to that employed in coal mining. Their equipment and management are based on the most approved methods, some of the mines being operated by electric power.

Gypsum in Foreign Countries

Canada.—The output of gypsum in 1911 was 505,457 short tons, valued at $978,863. This is a decrease of 19,789 tons in output, but an increase of $44,417 in value as compared with 1910. More than half the production of Canada comes from Nova Scotia, and only a small part of that is consumed in the Province, most of it being shipped in a crude state to New York, Boston, and other Atlantic ports. The gypsum quarries and works of Nova Scotia are fully described in the *Canadian Min. Journ.*, Jan. 1, 1912. Manitoba is another important contributor to the output of Canada.

England.—For the last 13 years, the United Kingdom has produced between 200,000 and 250,000 long tons of gypsum annually. Nottingham and Stafford Counties are credited with much the larger part of the total. The output of 1910 included 217,917 tons from mines and 37,642 tons from quarries. In 1911, the mines yielded 232,510 tons; the quarry returns are not available, but estimating them at 38,000 tons, gives a total for 1911 of 270,510 long tons.

Germany.—Complete statistics of gypsum production in the German Empire are not accessible. The figures for Baden and Bavaria are found in the accompanying table; Prussia produced 22,042 metric tons in 1910 and about 20,000 tons in 1909, the principal producers being the state-owned mines of Lüneburg and Segeberg in Clausthal. The exports of gypsum from Germany amount to nine times the quantity imported; about half the imports consist of special grades from France. Exports go to Austria-Hungary, Sweden, Belgium, and other European countries, in the order named; none come to the United States. Germany's foreign trade in gypsum is shown in the following table:

FOREIGN COMMERCE OF GERMANY IN GYPSUM.

Year.	Imports.		Exports.		Excess of Exports.	
	Metric Tons.	Value, Marks.	Metric Tons.	Value, Marks.	Metric Tons.	Value, Marks.
1909....	11,285	294,000	63,228	1,688,000	51,943	1,394,000
1910....	10,873	284,000	89,660	1,847,000	78,787	1,563,000
1911....	11,973	(e) 313,000	102,754	(e) 2,117,000	90,781	1,804,000

(e) Estimated

Characteristics of Gypsum

Beds of gypsum are associated with sedimentary rocks, such as slates, clay or shale; sometimes it is associated with salt, but not necessarily as this may have been washed away or the gypsum may have had its origin in fresh water.

Gypsite is a term applied to loose or unconsolidated deposits, formed of fine grains of gypsum and mixed very often with sand, clay, and other mixed products resulting from weathering or erosion of a gypsum bed, and its subsequent concentration by water or wind. The latter are often like sand dunes in shape, as in Arizona, New Mexico and Utah. The efflorescent type of gypsite deposit from the evaporation of solutions at the surface is represented in California.[1]

Many of these are concentrated and therefore thicker along the shorelines of lakes, but as a rule those formed from extensive lakes are thicker in the middle.

Gypsum is soluble in 415 parts of water, or less, if salt water; in crystallizing out it would retain 21 per cent. of water. Anhydrite is lime sulphate without water, and is therefore useless as a source of plaster.

The manufacture of plaster-of-Paris is based on the following properties: If gypsum is heated to 400° F., or above, all water of hydration is expelled and no addition of water will "reset" or hydrate the material. Water begins to be lost on heating from 15° F. in an air current; sometimes sufficient heat is generated in grinding to cause loss of water, thus vitiating assay results. The highest heat that can be used which will allow a reset of the plaster, is 390 °to 395° which is the temperature used in practice. Anhydrite in taking up water to form gypsum would increase in volume 60 per cent.

Common impurities in gypsum beds are clay, marl, and bitumen; less common are also quartz, dolomite, calcite, sulphur and iron pyrite. It is supposed that certain sulphate reducing bacteria, which have wide distribution, act on the moist gypsum and liberate sulphuretted hydrogen, leaving carbonate as a residue.

[1] *Bull.* 413, U. S. Geological Survey.

Most gypsum is manufactured into plaster or is used as a raw material for cement. Other usesare as a fertilizer, in paint, paper, artificial ivory, meerschaum, etc., and as an adulterant. Alabaster is a variety of gypsum that is cut into statuary, etc. In cement manufacture, either plaster or raw gypsum can be used, generally the latter if added before grinding or the former if added after thc cement is ground. The retarding action is not materially different.[1]

In the burning of cement, the sulphur of gypsum would be driven off, hence it is added subsequent to burning. Usually 2 per cent. is required; a 1000-ton plant would require about 4 tons of gypsum per day.

[1] Meade, "Portland Cement."

IRON AND STEEL

By Federick Hobart

Overproduction and too sanguine estimates of trade recovery in the latter part of 1910 left their ill effects to be worked out in 1911. The consequence was a half year of general depression and of efforts to work off accummulated stocks. Yet with all the vicissitudes of the year there was a large production; and in consumption the growth of population and of general industry began to assert itself. It still failed to come up to the capacity of the furnaces and mills, greatly enlarged as that was in 1907 and the two succeeding years. Averaging roughly, it may be said that the activity in finished steel, which is the final test, gradually increased from about 60 per cent. of the total productive capacity up to 75 per cent.; with the prospect that 1912 will open with nearly 85 per cent. employed.

It has often been said that every year in the iron trade has its parallel in the past; but we must go back 10 years or more to find a time when optimism received so severe a check. A break in the curve of slow recovery led to a feeling that there was no recovery at all, and to a failure to recognize the great underlying volume of business which was done. This was, perhaps, natural; but the statistics show that while 1911 was not up to the preceding year, there was no such break as characterized 1908, for instance. If production decreased, it was in part because of the necessity of using up the accumulated stocks of the previous year.

Iron Ores.—The production and consumption of iron ore for 7 years past are estimated as in the accompanying table:

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.

(In tons of 2240 lb.)

District.	1905.	1906.	1907.	1908.	1909(a).	1910(a).	(1911(a).
Lake Superior	34,353,456	38,522,129	42,245,070	26,014,987	42,586,869	42,442,397	32,793,130
Southern States	7,175,000	7,450,000	7,585,000	5,900,000	7,350,000	7,500,000	6,475,000
Other States	3,050,000	3,265,000	3,125,000	1,875,000	3,150,000	3,325,000	2,885,000
Total	44,578,456	49,237,129	52,955,070	33,789,987	53,086,869	53,267,397	42,153,130
Add decrease in stocks							650,000
Add imports	845,651	1,060,390	1,229,168	776,898	1,696,411	2,591,031	1,811,732
Total	45,424,107	50,297,519	54,284,238	34,566,885	54,783,280	55,858,428	44,614,862
Increase in stocks			3,750,000	750,000	521,000	950,000	
Deduct exports	208,058	265,240	278,208	309,099	455,932	644,875	768,386
Total consumption	45,216,049	50,032,279	50,256,030	33,507,786	53,806,304	53,263,553	43,846,476

(a) From *Eng. and Min. Journ.*

PRODUCTION OF IRON ORE IN LAKE SUPERIOR DISTRICT.
(In tons of 2240 lb.)

Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.
1855...	1,449	1875..	881,166	1895..	10,429,037	1904..	21,822,839	1908..	26,014,987
1860...	114,401	1880..	1,948,334	1900..	19,059,393	1905..	34,353,456	1909..	42,586,869
1865...	193,758	1885..	2,466,642	1902..	27,562,566	1906..	38,522,239	1910..	43,442,597
1870...	859,507	1890..	9,003,725	1903..	24,289,674	1907..	42,266,668	1911..	32,793,120

SHIPMENTS OF IRON ORE FROM LAKE SUPERIOR DISTRICTS.
(In tons of 2240 lb.)

Range.	1906.	1907.	1908.	1909.	1910.	1911.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Marquette.....	4,057,187	4,388,073	2,414,632	4,256,172	4,392,726	2,833,116
Menominee.....	5,109,088	4,964,728	2,679,156	4,875,385	4,237,738	2,911,174
Gogebic.....	3,643,514	3,637,907	2,699,856	4,088,057	4,315,314	2,603,318
Vermillion.....	1,792,355	1,685,267	841,544	1,108,215	1,203,177	1,088,930
Mesabi.....	23,792,553	27,492,949	17,257,350	28,176,281	29,201,760	22,093,532
Baraboo.....	128,742	76,146	122,449	82,759	91,682	115,629
Cuyuna						147,431
						32,793,130

RECEIPTS AND STOCKS AT LAKE ERIE PORTS.
(In tons of 2240 lb.)

Ports.	Receipts.			Stocks.		
	1909.	1910.	1911(a).	1909.	1910.	1911(a).
Toledo.....	1,374,224	1,225,202	493,345	332,456	433,215	661,382
Sandusky.....	11,088	.....	.....	39,557	17,728	2,471
Huron.....	243,082	197,951	223,947	477,333	375,118	344,371
Lorain.....	2,796,856	2,884,738	2,937,605	407,129	259,448	314,321
Cleveland.....	6,051,342	5,800,501	4,584,211	1,547,142	1,638,795	1,589,491
Fairport.....	1,734,277	1,516,434	666,365	867,640	839,970	636,566
Ashtabula.....	8,056,941	9,620,638	6,359,131	2,594,359	3,287,816	3,295,862
Conneaut.....	7,007,834	6,309,548	6,931,278	1,411,002	1,329,997	1,237,573
Erie.....	1,235,057	942,592	289,400	788,046	792,011	636,274
Buffalo.....	5,002.235	4,704,439	2,802.976	501,125	452,783	413,353
Detroit.....	159,889	296 412	243,292	.....	.....	.....
Total.....	33,672,825	33,498,455	25,531,550	8,965,789	9,426,681	9,131,664

LAKE SUPERIOR ORE SHIPMENTS TO END OF 1911.
(In tons of 2240 lb.)

Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.
Marquette.....	99,129,831	18.9	Gogebic....	67,739,155	12.9	Mesabi...	246,998,726	46.9
Menominee.....	79,367,967	15.1	Vermillion.	31,417,492	6.0	Baraboo.	1,087,938	0.2
						Cuyuna	147,431	

Accurate statistics are available for the Lake Superior iron ore from which nearly 80 per cent. of our pig iron is made. Dock records are carefully kept and are compiled each year by the Cleveland *Iron Trade Review.*

The decrease in stocks in 1911 is chiefly in those visible, as on the

Lake Erie docks; there are not statistics available for the exact determination of stocks in furnace yards.

The season prices of Lake Superior ore on Lake Erie docks were fixed as usual early in the year. They were 50 cents per ton below those of 1910, being as follows: Bessemer ore—base 55 per cent. iron and under 0.45 phosphorus—$4.50 per ton for Old Range and $4.25 for Mesabi; non-bessemer—base 51.5 per cent. iron—$3.70 for Old Range and $3.50 for Mesabi. These prices apply to not more than 20 percent. of the ore mined. The rest is from mines owned by the Steel Corporation and other large steel companies, and the price to them is a matter of book-keeping, the real charge being the costs of mining and transportation. To the prices given above the cost of transportation from Lake docks must be added to get the cost of the ore at furnaces; this will run from 50 cents to $1 per ton, according to the situation of the furnaces. Toward the end of the shipping season these prices were quietly cut about 25 cents a ton. There was a further reduction when the prices came to be adjusted for the season of 1912.

Prices of iron ore in the East are not controlled by any agreement or association of producers. Sales made through the year at New York mines showed a basis of 5 1/2 to 6 cents per unit of iron at mines. Newfoundland and Spanish ores sold at about 7 1/4 to 7 1/2 cents per unit of iron f.o.b. docks, Philadelphia. The furnaces using those ores, however, can generally get a low freight rate from Philadelphia, so that there is not much difference between them and domestic ores in cost at furnace.

Little or no Cuban ore is sold to outside furnaces, nearly all of it being used by the steel companies which control the mines.

In the South iron ore production is rather closely limited by the demands of the furnaces. The proximity of the mines to the points where the ore is needed and the fact that mining can continue without interruption through the year, render unnecessary the storage or accumulation of any large stocks of ore. Accordingly the decrease of about 15 per cent. in production corresponds pretty closely to the decrease in the make of pig iron. Nearly all the southern ore is mined under control of the furnace companies.

Eastern mines were fairly active through the year, though their production was to some extent limited by the lower requirements of furnaces. Most of these ores are sold on yearly contracts. The mines of the Lake Champlain district in New York were more steadily worked than any others.

Imports and Exports.—In the latter part of 1909 some good contracts were made for delivery of foreign ore to eastern furnaces. These

were not all fully carried out, partly because the supplies were not needed, and in some cases because the ores did not come up to the standard expected. Cuban ores imported go for the most part to two or three large steel companies which control mines in the island; though there is a little surplus for sale to outside users. The imports for the year, 1,811,732 tons, included 1,147,879 tons from Cuba; 194,965 from Spain; 219,238 from Sweden; 174,853 from Newfoundland and Labrador; 50,480 from Canada. The imports were second to 1910 the largest ever recorded in a year. There was a considerable fall in imports, owing chiefly to the lesser activity of the furnaces. Some large contracts for delivery of foreign ores to eastern furnaces in 1912 were made, principally for Swedish ore.

Exports are chiefly of Lake Superior ore to Canadian furnaces.

Perhaps the most important new development in the iron ore mining during 1911 was in Texas, where work progressed rapidly. The railroad connection with the mines was completed. This work has also aroused interest in other fields in the state. From the eastern field experimental shipments were made to Philadelphia and to Alabama, and the price at which ore can be mined and transported, would indicate the commercial possibility of using them. The ores in the eastern part of the state run up to 57 per cent. of iron and from 0.10 to 0.20 per cent. of phosphorus, while the picked ores in the northeastern section run 3 to 5 points higher in iron and of Bessemer grade.

Iron Ores in the United States

The Lake Superior Region (By Dwight E. Woodbridge).—The season of 1908 is the only one for 6 years in which iron ore shipments from Lake Superior did not exceed those of 1911. Total forwardings of iron ore from the Lake district for the past 10 years are as follows in gross tons:

1902	27,585,904	1907	42,266,268
1903	24,308,510	1908	26,014,987
1904	21,849,401	1909	42,586,869
1905	34,384,116	1910	43,442,397
1906	38,565,762	1911	32,793,130

Excessive production for preceding years, as well as the condition of the iron trade at the beginning of the last season, were responsible for this tremendous shrinkage. But such has been the recuperative power of the iron trade that, in spite of dock and furnace reserves of ore in greater volume at the close of 1911 than had been known, the probabilities are for a business in 1912 that shall fully equal any of the record-breaking seasons of the past. However, at the very close of 1911 indications for the following season were not such as to lead mining com-

panies to bestir themselves greatly in preparation, and the winter was dull and quiet along the Lake. Many concern had difficulty in holding their organizations, much as they might wish to do so. The cessation of operation at mines, the lessened need of development work, both in underground and open-pit properties, and the lack of money with which to do much, have compelled the scaling down of both administrative and operative forces.

Only in one direction was there noteworthy activity—in the matter of taking ore from the lands under lease to the United States Steel Corporation by the Great Northern Railway Company. The contingencies of that lease were such as to demand as large shipments as possible, that minima already paid for but not mined should be forwarded, in order that interest charges on these advances might be saved and that as large shipments as possible should be made during the remaining 3 years of the leasehold. That interest alone was no small item is evident from the fact that, at the commencement of 1911, there had been paid the Great Northern in advances on ore not yet mined, considerably more than $5,000,000, while the deficiency in tonnage moved was, of course, much in excess of 5,000,000 tons. By concentrating its efforts on Great Northern lands the Steel Corporation has taken therefrom during the year 5,500,000 tons. This is a most remarkable record; for not only must there be properties developed on Great Northern lands to permit such shipments, but there were difficulties of a physical nature —those due to contracts with other fee owners than the Great Northern, other minima aggregating enormous totals, all of which were to be met, and the requirements of furnaces, which demanded certain classes of ore, some of which could not be secured from Great Northern mines. Total shipments of ore for the year by all mines operated by the Steel Corporation, on all Lake ranges, amounted to 13,943,000 tons, of which almost 40 per cent. was under the Great Northern lease.

The shipments of ore from the Lake region are given in the tables on a preceding page. To the figures in those tables should be added 128,628 tons from the Michipicoten and 115,700 tons from the Moose Mountain range in Canada. These should be fairly included in the Lake Superior region, although they are beyond the limits of the United States.

The Cuyuna range appeared in 1911 for the first time, shipments being from Kennedy mine of Rogers-Brown Iron Co. There were 184 shipping mines in the Lake region in 1911, compared with 196 in 1910. Of the shipping mines last year 90 were on the Mesabi Range; 38 on the Menominee, 29 on the Marquette, 17 on the Gogebic, 7 on the Vermillion, 2 on the Baraboo and one on the Cuyuna.

Shipments of the Oliver Iron Mining Company, mining subsidiary of the

Steel Corporation, in 1911 were 17,282,499 tons, or 52.7 per cent. of the total. Nearly one-third of this tonnage came from mines on Great Northern leaseholds, or 5,499,759 tons. This was 1,729,759 tons more than the agreed minimum for the year; but the Corporation was still behind at the end of the year. The total minimum requirements up to 1912 are 11,250,000 tons; the total mined up to that date was 7,832,138 tons. The minimum for 1912 is 4,500,000 tons and for 1913, it is 5,250,000 tons.

The great attention given to these leases combined with the general falling off in shipments to make some remarkable changes in the older mines of the corporation on all ranges.

The abrogation of the lease, assured now for the close of 1914, means that the Corporation will mine from those lands as much as possible of the minimum of 27,000,000 tons that will then have accrued. To mine it all will require the shipment of a little more than 6,000,000 tons for each of the 3 years, or but a trifle more than was produced therefrom during 1911. Doubtless all will be mined, and the corporation will retire from the Great Northern lands with a clean balance sheet. There is speculation in financial circles as to what effect the abrogation of this lease will have upon the Great Northern holdings. No doubt the exploration of these lands and the opening and development of mines, will enhance their value, though it will eliminate the element of speculative statement as to the tonnages existing thereon, an element that has been much in evidence the last few years. The purpose of withdrawing from this lease seems to have been to diminish the force of the allegation of monopolistic character of the Steel Corporation's ore holdings, combined with the fact that the lease was a serious economic error. In point of fact, and with the broad and general view of the situation that is the proper one, the ownership or loss of these lands is a matter of little importance, when viewed in connection with the reserves of available iron ore in the United States.

Progress in Mining Methods

Progress of the year in the matter of engineering science, improvements in equipment, power, etc., has been marked. Of the items of advance in this line these stand preëminent: Electric utilization of water powers; the sinking of permanent concrete shafts, of which a dozen or more have gone down; the inauguration of scientific management and the inauguration of many plans for practical conservation of ore deposits by the dressing of lean ores and low-grade ferruginous materials.

While a first glance at the operations of the year lead one to the

opinion that abrogation of the Great Northern lease by the Steel Corporation and the reduction of ore freights from the Minnesota districts were all-important events, this is far from the truth. To the steps taken for the beneficiation of low-grade ores one should assign first place.

First in tonnage and results is the great concentrator of the Oliver Iron Mining Co., on the Mesabi range. It is true that this is not essentially a matter of the past year, for experiments looking to the washing of these lean Mesabi ores have been in progress for 6 years, and two units of the works were operated a short time in 1910. For a period of about 160 days' total working time, in 1911, the work has been in full operation with the five units, with a production of 1,900,000 tons of merchantable concentrates, or 2000 tons above the rated capacity of 10,000 tons per day. This is an admirable result, and one that shows that experiments preliminary to construction must have been most thorough. It is understood that costs of this separation of sand from the west Mesabi ores have been satisfactory, and well within expectations, and there is little doubt that, should an extension of this operation be desirable, the new works would be along somewhat similar lines. This plant has treated ore from a number of mines on the Mesabi, all operated by the Oliver company. It was built primarily for use in connection with the ores of the Canisteo lease, but those from several other mines have been treated the last year, particularly from the Walker, Hill and Holman, two of which are Great Northern leases. In passing, it may be remarked that the capacity of this works is several times that of any other iron-ore concentrating plant in the world.

Hawkins mine, belonging to the International Harvester Co., contains a large tonnage of similar sandy ores, and works of the same type are underway there now, to be ready for operation the coming season. This installation differs from the Coleraine plant in but one essential detail, which is that concentration is not to be carried quite so far, for there will be no tables for treating the fines. This new plant will have a 20-hour capacity for 2000 tons of product.

At Moose Mountains, in the Georgian Bay region, are vast deposits of magnetite of from 35 to 40 per cent. iron, with a silica-hornblende-feldspar gangue, and of schistose nature. Here a wet magnetic separator plant of 800 tons daily capacity in concentrates is under construction, using the Grondal patents. This plant is modeled after that at Sydvaranger, Lappland. The works will cost about $250,000 and are expected to produce a brick of 60 per cent. or better, low in phosphorus and of excellent furnace texture. The product to come from this plant the present year will be trifling, as but a part will be ready before late in the season. The Moose Mountain company estimates above 100,000,000

tons of ore upon its property that can be made merchantable. The cost of separation is expected to be under $2 per ton.

Still more interesting are the works under construction at the Magpie mine, Michipicoten district, by the Lake Superior Corporation. Siderite rocks occur frequently in this region, and the Magpie is a mass of concentrated siderite, carrying, say, 35 to 40 per cent. iron and 18 to 20 per cent. carbon dioxide. The located claims there, or which the Corporation has bought seven for $250,000, contain a tonnage of this rock available for calcination that may amount to 40,000,000 or more tons. Elsewhere in the same region is much more. As the result of experiments in the roasting of this rock to drive off the CO_2 a plant is being erected sufficient to treat 1500 tons per day, and it is to be in operation by the opening of navigation. The material carries sulphur to excess in the form of pyrite, most of which is driven off at the high temperature maintained in the roasters by means of producer gas. The product is a porous bessemer hematite, running about 57 per cent. iron and basic in nature, and will be made at a cost of about 40 cents.

Experiments in drying ore for the purpose of securing a considerable saving in freights and a lesser one in furnace treatment, were carried along for some months and have resulted in the erection of a drier plant at the Hollister mine, Crystal Falls district, Michigan. Hollister ore assays about 49 per cent. iron, high silica, low alumina and above the average in hygroscopic moisture, of which it has 13 to 14 per cent. A drier of the Atlas, horizontal rotating type, with capacity for 300 tons per 20 hours, has been in successful operation there during a part of the year. The drawbacks to successful operation of such works, have been met, and are classed as easily surmountable. Indeed, the figures seem to prove them less serious than a casual consideration would indicate. For instance, re-admission of moisture into the partially dried ore while *en route*, by rains, from atmospheric absorption and the like, will affect merely the exposed portions of the mass of ore, and then only to the extent of the amount of fall or the percentage of atmospheric saturation limited by the exposed area. In ore in cars or pockets this does not mean much. Hollister ore has been brought down to 6 or 7 per cent. moisture and the selling guarantees for this ore at present a marked contrast to those for prior years. It is now proposed to extend this drying process to an ore more difficult of treatment. Works to consist at once of two units, each of 40 tons per hour capacity, are under construction at the Brunt mine, Mesabi range. Brunt ore assays 47 per cent. iron, 5.7 per cent. silica, 5 per cent. alumina and more than 15 per cent. hygroscopic moisture, therefore quite clayey and wet. The

works under construction will dry 1600 tons per 20 hours. If successful as predicted this capacity will be doubled.

At the mine of the Northwestern Iron Company, at Mayville, Wis., a Grondal briquetting plant for fines has been built and is now in operation. The Mayville ores contain a large proportion of fines that have caused much loss in furnaces, and it has been the practice to wet these ores thoroughly before changing them, so that they would stay until thickened or covered. A Grondal plant of four ovens, capacity 400 tons per 24 bours. is now working on these ores. In this plant the kilns consist of long channel furnaces, through which the shaped bricks pass slowly on flat cars. These cars form a continuous, intermittently moving platform dividing each kiln into an upper and a lower compartment. Through the lower one cold air is drawn to protect the trucks from heat; along the upper one and down through the bricks and platforms, oil flames pass, agglutinizing the bricks, with an extreme temperature of 2400° F. The bricks are made in a near-by pressing plant.

The plant at Port Arthur, Ont., for the elimination of sulphur from Atikokan ores by roasting with waste furnace gas, has been in operation for a year or two. The ore is crushed coarse and roasted with a heat that drives off most of the sulphur carried in the form of pyrite, etc.

In addition to all these devices the Jones Step Process Co. has been continuing its experiments in enriching and nodulizing low-grade siliceous ores, at its Iron Mountain headquarters, and will soon open a furnace at the Kloman mine, where there are large quantities of lean, siliceous banded ores.

The significance of all these efforts to enrich otherwise valueless ferruginous material in the Lake Superior region or to improve already merchantable ores so as to effect a material saving in freights and final treatments, should not be minimized. That it has cystallized in a year of low prices and dull business in the ore trade is not so remarkable as might appear at first glance, and is deeply significant of the attitude of the ore trade toward the future. In time of peace prepare for war, is no less a political than a commercial axiom. This work is doing much to bring into the column of available reserves ores that have been classed either as future low-grade reserves or that have not been considered at all.

During the past year explorations for additional ore supplies have been more active than might have been expected, considering the practical impossibility of selling ore. But little new ore has been added to Mesabi totals, and this little was foreshadowed by work done in 1910. The tonnages near the northwest corner of T 58 R 15 have been more accurately defined, and it is now well known that a large deposit exists there, the mining of which will present some new and difficult problems in

the diverting and handling of water in quantity. The runoff from a watershed of several hundreds of square miles must be diverted before this ore can be taken out. Certain arrangements of a private nature between the Oliver Iron Mining Co. and the estate of Henry Stephens resulted, toward the end of the year, in great activity in T 59 R 15, and some addition to known ore reserves may result. Throughout the year there has been active exploration by this company on its lands along the range, lands previously explored but the results on which were, for one reason or another, considered questionable. These have resulted in minor changes in totals.

Exploration on the Vermillion, very active a year ago, has diminished. Four or five companies are working, and the prospects are excellent for some addition to known reserves in that most alluring and difficult field. On the Michigan ranges there has been little addition to former reserves, excepting around Iron River, where a great amount of work is underway, and where the tonnages known have been materially enlarged and the number of mines increased. This old district is showing a most interesting and instructive increase in ore reserves, and bids fair to continue the scene of satisfactory exploration.

On the new Cuyuna range results have been encouraging. More is known of the formation, and exploration can be done more scientifically and accurately than earlier. Estimates of tonnage differ so widely that no attempt will be made to give figures, but it is unquestionable that the tonnage of merchantable ores actually measurable is still comparatively trifling. This is in part for the reason that it is extremely difficult to connect and correlate drill holes in a district whose ore bodies dip deeply, may be narrow and short, and where no rule has yet been adduced for their appearance. An enormous amount of drilling has been done; and the formations are such that holes must be frequent to prove anything definite. The district has entered upon its career as a shipper, with the forwarding of 149,000 tons by the Kennedy mine of the Rogers-Brown Ore Co. Several concrete shafts have gone down, good plants have been installed and half a dozen mines will be shipping in 1912. Most of these are bunched in the north center and northeast part of T 46 R 29, though Kennedy mine is in T 47 R 28, Adams mine in T 46 R 28 and Barrows in T 44 R 31. It is 25 miles from Kennedy to Barrows, these being the most widely separated developments of importance. But while the known tonnage of good ores is slight, there are large possibilities in the Cuyuna. Unfortunately, much of the ore drilled the last year is manganiferous, carrying, in rare instances, as much as 30 and 40 per cent. manganic oxide. Whether these ores exist in quantity is not yet known. Were such ores located elsewhere, in the south for instance, they would be

more valuable than where they are. Many drills are at work throughout the district, and a year from now more will be known of its deposits.

Indications are now for the shipment during 1912 from this district of not far from 300,000 tons, which is expected to come from the Kennedy and Armour mines of the Rogers-Brown interests and from the Thompson of the Inland Steel Company.

Late in the year a reduction of ore freight rates from Minnesota districts to Lake Superior was made by all the railways operating there. This reduced Mesabi range costs, effective in 1912, by 20 cents and Ely rates 40 cents per ton. The Cuyuna, which had a differential in its favor of 15 cents, is now 5 cents above Mesabi rates, and the combined freight charge for a ton of Menominee range ore, delivered Lake Erie, which was formerly 55 cents under Mesabi and 40 cents under Cuyuna, is now but 35 cents under Mesabi, though still 40 cents under Cuyuna. The Cuyuna rate will soon be lowered 5 cents, or to 60 cents.

Iron-ore prices were reduced a year ago by 50 cents per ton, practically 1 cent per unit, and at the beginning of January, 1912 ranged from $4.50 for Old Range bessemers down to $3.50 for a Mesabi range non-bessemer. Notwithstanding this decline there has been a material decline, and early sales for 1912 are made at from $3.75 for Old Range bessemer to as low as $2.75 for Mesabi non-bessemer. At these figures requirements for the year were fairly well covered. This drop, though severe, is not so great as it seems, for vessel rates are materially less than before and, in the case of the Mesabi, the freight cut absorbs 20 cents of the decline in price.

Texas (By Wm. B. Phillips[1]).—During the last 2 years there has been a revival of interest in the iron ores of east and northeast Texas.

There are now three companies engaged in the prosecution of plans for developing and mining these ores on a commercial scale. Such plans as have come into public notice look toward the shipping of Texas ores from Gulf ports.

Reports have been made on the possibility of assembling these ores on the Gulf and erecting there blast-furnaces and steel works. These plans, tentative and somewhat inchoate, considered the bringing of coking coal from Alabama and West Virginia and the construction of by-product oven at a point so convenient to Galveston or Houston, or other city as to admit of the sale of the surplus gas, for example, for power and heating purposes. While it may not be doubted that a considerable and a lucrative business could be secured by shipping these ores to coastwise or to foreign furnaces, yet we are of the opinion that a much better and a

[1] Director University of Texas Bureau of Economic Geology and Technology, Austin, Texas.

more permanent business could be built up in the production of pig iron and steel in the State of Texas.

The consumption of pig iron as such in Texas is now between 250 and 300 tons a day, inclusive of all grades. In addition, there is a very large consumption of rolled iron and steel. In cotton ties, wire fencing, wire nails, material for reinforced concrete structures, etc., there is, beyond any question, a very large and an ever widening market in Texas and the Southwest generally. By far the greater quantity of such material is hauled from 500 to 1000 miles before it reaches the borders of Texas and then has an additional haul of 250 to 500 miles before it reaches its chief points of consumption within the State.

The saving in freights alone would add a considerable item to the gross revenues of an iron and steel plant operating within the State. The lowest rate from the ore fields to tide-water may be taken at $1.34 a ton, the maximum distance being something over 300 miles.

The results of careful investigations into the extent of these ore fields of east Texas and the amount and quality of the ore to be obtained have changed the situation radically. One company alone has dug more than 800 pits on but a small part of its holdings and has proved the extent and quality of the ore in areas that were previously not regarded as within commercial considerations. There are very large reserves of iron ore in east Texas that will carry 50 per cent. of iron, with less than 0.25 per cent. of phosphorus.

The Llano County iron ores carry from 60 to 65 per cent. of iron, and in certain localities the phosphorus is low enough for the production of Bessemer stock. They occur at about the same distance from tide-water as the more northerly of the east Texas ores. For the most part the Llano County ores are close-grained, hard and dense magnetites and hematites whose reducibility in the furnace is not to be compared with that of the more or less spongy east Texas ores. With the exception of one mine, which has been full of water for many years, there is no known commercial deposit of iron ore in Llano County that is within 12 miles of a railroad. In time these ores may come into the market, but for the present the chief interest centers around the east Texas ores.

Iron Ore Reserves in Michigan

Michigan is included in the Lake Superior region; but especial interest attaches to the estimate on the iron ore reserves of the State issued by the State Board of Tax Commissioners, based on data prepared chiefly by C. K. Leith. The report published by the United States Geological Survey gives a summary of this notable work. The estimates

average very close inasmuch as many of the mine reports could be taken as they were reported by the companies, with the proper allowances. Most of the estimates received from the mines were of ore actually developed above the bottom levels. Some include as probable ore, the extensions of partly developed levels. As the reserves are not treated the same by all companies, the estimates are reduced. Estimates of developed ore are accepted as turned in but the estimates of probable ore are modified to include for each mine at least one full level below present workings with areas equivalent to the lowest fully developed levels. Partially developed levels are extended to the dimensions of the nearest fully developed level. Where drill holes show ore in advance or below and within reasonable distance of the workings, this ore is added. Where geologic conditions show exhaustion or probable narrowing of the ore, this is taken into account. Further, the total shipments of the mine, persistence of the ore body, the depth, the comparison of the areas of low levels with the average of the levels above are considered. For most properties, the probability is that the ore body is *more extensive* than has been estimated. The greatest care is required in estimating the ore cut by drill holes. Some companies simply turned in the record of their drill holes whereas others made estimates in various ways including actual tonnage for distances from or between the holes on the assumption that the ore extended. As a rule no allowance has been made for ore extending beyond limiting holes and these exclusions should counterbalance too large areas extending between the holes. Where the holes were extended in a line in such a way as to make it difficult to connect the areas, a uniform method which seemed conservative, consisted of connecting the circumferences of circles described at the limiting holes, the radius being usually 75 ft. Some of the estimates made deductions of 20 per cent. for mining loss and rock and others used a larger figure. The range in the figures used for the number of cubic feet of ore per ton appear to be 7 to 18. Careful tests made by the Cleveland Cliffs Company and the Oliver Company show an average of 8 cu. ft. per ton for hard and 12 for soft, per ton, and 13 to 14 cu. ft. of low-grade ores. The Oliver Company uses 10 cu. ft. throughout in its estimates, the error in this figure being less than in the probable dimensions of the ore bodies. The figures that were taken as a standard are given as follows:

Marquette hard ores	8
Menominee ores and Gogebic ores	10
Soft ores of the Marquette, Crystal Falls, and Iron River districts	12
Low-grade high-silica ores	14

The result of the estimate has added 25 per cent. to the total tonnage as reported by the company. The following table gives the iron ore reserves by districts as given by Mr. Leith:

MICHIGAN IRON-ORE RESERVES, BY DISTRICTS.(a)

District.	Total Area of Ore in Bottom Levels of Active Mines.	Area of Ore Developed by Drills; Vertical Projection to Surface. (b)	Total Area of Iron Ore in Michigan.	Reserve in Mines.
	Square feet.	*Square feet.*	*Square feet.*	*Long tons.*
Menominee	263,465	.....	263,465	11,650,450
Crystal Falls	209,730	167,000	376,730	3,682,142
Iron River	1,065,000	984,000	2,049,000	18,297,700
Gogebic	694,000	None.	694,000	31,605,000
Swanzy	.....	3,298,000	3,298,000	2,712,737
Soft ore or "hematite," Marquette	1,045,000	7,497,740	8,542,740	37,516,800
Hard-ore mines top of Negaunee formation, Marquette.	350,000	841,065	1,191,065	17,391,335
Mines west of Michigamme	99,000	211,600	310,600	1,742,000
Total	3,726,195	12,999,405	16,725,600	124,598,164

District.	Reserve in Area Drilled.	Total Reserve.	Total Shipment.	Average Depth of Active Mines at Lowest Developed Level.
	Long tons.	*Long tons.*	*Long Tons.*	*Feet.*
Menominee	.....	11,650,450	42,616,000	1,160
Crystal Falls	1,925,000	5,607,142	15,609,000	1,070
Iron River	9,719,000	28,016,700	6,179,300	445
Gogebic	None.	31,605,000	55,027,691	1,400
Swanzy	4,138,738	6,851,475	2,420,232	.....
Soft ore or "hematite" Marquette.	25,929,000	63,445,800	.....	(c)690
Hard-ore mines, top of Negaunee formation, Marquette.	2,939,200	20,330,535	96,296,717	(c)1,041
Mines west of Michigamme	332,000	2,074,000	1,080,000	(c)220
Total	44,982,938	169,581,102	219,228,940	.....

(a) Low-phosphorus, high-silica mines not tabulated.
(b) The phrase "vertical projection to surface" signifies that the area of dipping and pitching ore bodies is stated in terms of horizontal measurement at the surface—that is, that the structure has been flattened out in making computations.
(c) Not rated according to tonnage of outputs.

The following table shows the distribution by counties of the number of iron formations and the magnetic belts in 40-acre tracks:

CLASSIFICATION AND DISTINCTION OF IRON LANDS IN MICHIGAN BY COUNTIES, IN ACRES.

County.	Measured by 40-acre Tracts.[1]				Determined by Actual Planimeter Measurement of Areas.		
	Class A.	Class B.	Class C.	Total.	Iron Formation.	Magnetic Belt.	Total Formation and Magnetic Belt.
Dickinson	11,040	4,000	6,440	21,840	4,226	3,095	7,321
Marquette	34,960	8,880	.....	43,840	16,810	2,847	19,657
Gogebic	6,720	1,720	2,320	10,760	5,449	496	5,945
Iron	11,040	10,760	4,840	26,640	1,972	12,702	14,674
Baraga	3,160	.....	.....	3,160	810	1,841	2,651
Total	66,920	25,360	13,600	105,880	29,267	20,981	50,248

[1] Lands in class A are known to be favorably located for exploration. Lands of Class B, the geologic conditions are doubtful, but the lands will probably be explored in the future. Lands in Class C are magnetic belts or lands of lean or unfavorable iron formation in which the chances for ore are remote or entirely unknown.

In these tracts an area about twice as large as the actual iron land is included.

Iron Ores in Foreign Countries

Algeria.—The iron ore deposits and statistics of production are fully given by Mr. Dussert.[1] The exports for 1911 were expected to increase considerably over the previous years.

Australia.—In Western Australia the deposits of the Murchison stand out from those thus far discovered, but are at present neglected. Other deposits working on a small scale are used as flux in copper or lead smelting. Some of the ores as in the Lalla Rookh zone (Pilbara) could be concentrated to a high-grade ore. Of such the ore of the Weld Range is another example; it assays: Iron 35.5 per cent.; silica, 43.42; phosphorus, .089; sulphur, .036; titanium, trace. Many assays of samples from all the deposits are given in *Australian Min. and Eng. Rev.*, Nov. 5, 1910.

Austria.—Iron ore activity, increased in 1910 especially in Styria, 2,627,583 metric tons were produced all told. The production of pig iron and steel required 56,140 tons manganese ore and 28 per cent. of the iron ore was imported from Hungary, Sweden, Spain, Greece and Russia.

Belgium.—The production of iron ore in 1909 was 175,280 tons of limonite and 24,430 tons of hematite. The great bulk of the material used for the iron and steel industry was imported.

The experiments with dry blast at the Differdarge I & S. Works, in Luxemburg, promise to be successful. The innovation is that of Felix & Daubine and Eugene V. Roy; the calcium chloride used absorbs the moisture and is then regenerated by a mixture of furnace gas and ore and then cooled. Twenty-four tons of calcium chloride are charged to each dying tower and it takes 4 hours for the desiccation process. There are three towers, one idle and one cooling. The installation is for 1,000,000 cu. ft. of air per hour.

Brazil (By Bradley Stoughton).—The beginnings of the iron industry in the new world were made in Brazil in the closing years of the sixteenth century and the Brazilian Government, in February 1911, followed the example of the Portuguese Government in 1597, by encouraging the establishment of an iron and steel industry, and by the more substantial aid of guarantees and bounties. Much interest has been aroused in the United States in recent years in the iron mines of Brazil, where it is known that abundant deposits exist. An unsuccessful attempt was

[1] *Annales des Mines*, Vol. 1, 1912, L. 2 and 3.

made to market the ores in the United States, and a Brazilian attempt to finance plants, either in Germany or this country, met with the same result. At the present time, however, two companies, one financed by Brazilian capital, and the other by French capital, are commencing operations by acquiring ore and coal lands, shipping facilities, plant sites, etc. The Brazilian Government has bound itself to pay a bounty on manganese ore of $3.25 per ton; steel plates and shapes $8.12; axles, bars and other finished material, $9.75 per ton, and will force the Brazilian Railway to provide facilities for transportation at special rates, it being understood that the plant for iron and steel making must be ready within 5 years. Meanwhile, American companies have obtained options on Brazilian iron ore deposits.

The *I. & C. Trades Rev.* states that an English company has also secured iron deposits covering 9000 acres and estimated to contain 60,000,000 tons. The Brazilian geologists say that 60 or 75 per cent. ores occur in 52 places in Minas Geraes. Also in Caracas there is said to be enough iron ore to supply the world for 160 years. The English corporation expects to export 3,000,000 tons of ore per annum, but are required by the government to install an electric smelting furnace that will produce 1000 tons of iron per month.

Canada.—In Ontario about the same quantity of iron ore, 230,000 tons, was raised and shipped in 1911 as in 1910. It is mainly hematite from the Helen mine, Michipicoten, owned by the Lake Superior Corporation, the output of which goes to the company's furnaces at Sault Ste. Marie. Other producing mines were Atikokan, west of Port Arthur, and Moose Mountain, northwest of Sudbury, both of which are of magnetite. At the last-mentioned mine a Gröndal concentrator is being erected with a capacity of 375 tons of finished material per day. The iron mines of eastern Ontario shipped nothing during the year, but a concentrating plant is talked of for Trenton on Lake Ontario, to treat ore from deposits on the Central Ontario Railway. The production of pig iron for the 9 months ended Sept. 30, 1911, was 22,842 tons less than in the corresponding period of 1910. The total for the year will be about 400,000 tons.

In Nova Scotia the amount of iron ore mined during the year was 53,995 tons, a slight increase over the output of 1910. The Canada Iron Corporation, at Torbrook, was the only operator, all the ore used by the Nova Scotia Steel and Coal Co. and the Dominion Iron and Steel Co., being imported. This amounted to 853,904 tons, of which 831,660 tons came from, Newfoundland, 15,734 from the United States, and 6509 from Europe. Of the ore mined at Torbrook, 200 tons were shipped, 357 tons sent to concentrator, and the remainder put on stock pile at the mine.

The total amount of ore on hand at Torkrook and Port Wade is now in round figures 110,000 tons. A modern concentrating plant is being erected by this company at Nictaux.

The Torbrook iron ore deposits and adjoining prospects are described by Howells Fréchette.[1]

Cuba.—The importance in America of the Cuban iron ore continues to increase.[2]

The imports of iron ore into the United States from Cuba for the year ending June 30, 1911 were 1,231,916 tons. The duty under reciprocity treaties is 15 cents per ton less 20 per cent. This tonnage includes also some cinder and manganiferous iron ore. The ore imported from Cuba for the calender years 1909, 1910 and 1911 is 927,774; 1,451,096 and 1,147,879 tons, the values being $2,681,028, $4,459,789 and $3,218,485, respectively. Total exports were 1,424,206 in 1910.

Chile.—The cupriferous iron beds in Chañaral as at Hundido, Tres Gracias, etc., yield important copper outputs. The industry dates back but 25 years. Near the harbor of Taltal, in the district called Norte Magnetico, the ore is high grade. Some of these deposits are magnetite and others hematite. Iron-ore beds are found in the provinces of Macama and Coquimbo, close to the maganese deposits near Nezra. Concessions are granted to a company for the remarkable deposits of Algarobo and Algarobello. The Tofo Iron mines belonging to the Corral blast-furnaces are 4 1/2 miles from the coast, near the port of Totoralillo. It is estimated with a depth of 1300 ft. that the deposit may contain 145,000,000 tons of 65 per cent. ore. The working is done in open cuts, but much ore is gathered from the surface of the ground.

China.—Industry centers around Hankow, continue to grow. The ore from the Tayeh mines was 303,000 tons, of which one-third was shipped to Japan. The exports of iron-ore to the United States have been temporarily stopped, owing to financial difficulties of the American company.

France.—The production of iron ore from the Bricy Basin in 1911 was 10,427,000 metric tons, which compares with 8,470,000 in 1910, 6,310,000 in 1909 and 4,368,000 in 1908. The increase has been at the rate of about 2,000,000 tons a year.

Honduras (By B. Stoughton).—A United States company is contemplating expending several millions of dollars to open up the famous magnetite ore mine of Agalteca in Honduras, and build a railroad to the coast, provided the report of their engineers indicates the presence of as large a deposit as is believed to be there. This "iron mountain" has

[1] *Bulletin* No 7, Miles Branch, Dept. Mines, Canada, 1912.

[2] The iron ores of Cuba are described in *The Iron Age* for August 15, 1907, April 9, 1908, and August 26, 1911, and in the "Transactions of the American Institute of Mining Engineers," of June, 1911.

been worked spasmodically, first, by the Spaniards and later by the natives.

Italy.—Prof. G. LaValle,[1] states that though a number of deposits were worked in Italy, the southern belt of iron ores is not promising. The most important occur in the Aspromonte chain in Calabria and near Messina, Sicily.

An interesting ore-loading plant exists at the Albano mines, Island of Elba.[2] The mines on the island are on the rough east coast; the Albano deposits are worked by open cuts and hand tramming or horses. In the nearby Portello district the transporting is done by mining locomotives and trains. The first district has an output of 800 tons per day of 40 per cent. ore, the second, 1400 tons per day of higher-grade ore. The ore is a hard spathic brown ironstone, crystalline in structure and frequently found in a fibrous formation with ochre veins.

Mexico.—In 1911, American capital became interested in the exploration of iron-ore indications of large extent in Mexico.

Newfoundland.—The Wabana mines are described by Brenton Symons.[3] The two companies operating what are said to be the largest low-grade iron ore deposits in North America, are the Dominion Iron and Steel, and the Nova Scotia Iron and Steel Companies. The ore is remarkable for low sulphur and low phosphorous and it is being mined for thousands of feet under the sea. The deposits are on Bell Isle in Conception Bay and for many years before opening the deposits in 1895, hunters and fishermen were troubled with red mud. The Nova Scotia company sold in 1898, a portion of its leases to the Dominion Iron & Steel Co. for $1,000,000. In the agreement made between the companies, the lower seam, was allotted to the Dominion company, and the upper seam of a far less area, to the Nova Scotia company; both beds in the 5 1/2 square miles parallel to the shore were allotted to the Dominion, while the 32 square miles of submarine areas outside that limit were to be the sole property of the Nova Scotia company. At the time of this arrangement, the prolongation of the beds seaward was to some extent problematical.

During the first year of exploitation, by reason of the slight northwest inclination of the seam, the ore was so close to the surface, that it was raised and put on board for 39 cents per ton, with a freight of 41 cents to the Sydney furnaces. Now it approaches $1 per ton. Only the two lower seams are exploited, as at present the upper beds are of no value.

The stratigraphical sequence of the iron deposits in descending order is as follows: Shales, 56 ft.; hematite, 1 ft.; sandstone, 5 ft.; hematite,

[1] *Iron and Coal Trades Rev.*, Oct. 6, 1911.
[2] C. A. Tupper, *E. and M. J.* Sept. 2, 1911.
[3] *E. and M. J.*, May 20, 1911, p. 1008.

1 1/4 ft.; sandstone, 2 ft.; hematite, 1/4 ft.; sandstone, 3 ft.; hematite, 1/4 ft.; sandstone, 2 1/2 ft.; hematite, 1 ft.; sandstone and shale, 3 ft.; hematite, 4 ft.; sandstone and shale, 46 ft.; hematite, 9 ft.; sandstone and shale, 288 ft.; hematite, 12 1/2 ft. A total of 434 ft., of which 29 ft. is hematite.

The low dip, 9 degrees, has permitted working a great area opencast.

The lower seam is worked by the Dominion Iron & Steel Co., and has a width of 9 ft. of dense homogeneous hematite of an almost unvarying quality of 50 per cent. iron. As it is 293 ft. below the upper seam and covere 800 acres within land limits, there is a considerable reserve of ore, estimated at more than 20,000,000 tons.

The upper seam, exploited by the Nova Scotia Steel & Coal Co. has a land area of 170 acres and was estimated to have contained 6,000,000 tons of ore, the larger portion of which has been raised and exported.

To reach the submarine areas (about a mile from the shore) a new slope was commenced in May, 1906, and at the close of 1908, it had crossed—at a distance of about 4000 ft.—the western boundary of the property of the Dominion company. The extent of the deposit is so great, that the daily output need only be limited by the capacity of the hoisting equipment. There is little doubt that the ore extends over at least 12 square miles and this would give for one seam, 400 million tons.

The dimensions of the excavation were 14×8 ft., all in hematite, and it was driven in three shifts of 8 hours, yielding in 24 hours 140 to 150 tons of 50 per cent. ore. The holes were drilled from 6 to 12 ft. deep by machines, and the dynamite consumed was 3/4 lb. per ton, equivalent to the removal of 5 cu. ft. per pound of explosive. The speed of advance averaged 10 ft. per diem.

After crushing, sorting raises the hematite from 50 to 52 per. cent. The daily output is about 1600 tons.

An electric plant to operate the cables on all the tramways underground as well as at surface; an efficient engine for assuring speedy traction up the new submarine slope, with new machinery for treating the ores, and a new steamer, have been purchased. A new wharf, which exacted the blasting of 8000 cu. yd. of rock from the overhanging cliff, has been finished, measuring about 120×90 ft.

The ores of the Dominion company are exported to the furnaces at Sydney, 400 miles distant, the steamers making the return voyage in 6 days. The Nova Scotia Co. ships its hematite to Atlantic ports of the United States and to Sydney, Great Britain and Holland.

To cheapen freight a new steamer, called the "Storstud," has been built at Walker-on-Tyne on the Isnerwood system of longitudinal fram-

ing. The length is 425 ft. and 58-ft. beam, with cylinders of 25 1/2, 47 and 78 in., and capacity of 10,650 tons. The value of a ton of the ore at Sydney is about $2 and the cost of mining and freight is probably about 90 cents. There is now a Government duty of 7 cents per ton payable on exportation.

Russia.—The production of iron ore in south Russia centers at Krivoi-Rog and Kertch, the first named producing 95 per cent. In 1909, owing to the mines having put out rather more than market requirements, there accumulated quite a quantity at the pits' mouths. On January 1, 1910, this accumulation of ore at the Krivoi-Kertch mines was 595,800 and 108,000 tons, which decreased to 563,400 and 81,000 tons, respectively, on July 1, 1910. Increasing demands further decreased these accumulations to 450,000 and 45,000 tons on January 1, 1911, and to 363,000 and 7200 tons by July 1 this year. The present strong demand is indicated by the output for the first half of 1911—2,466,000 tons, against 2,136,600 tons for the same period in 1910 at Krivoi-Rog, and at Kertch 138,400, against 160,200 tons. The demand was principally for ores of no less than 62 per cent. iron. The price of ore this year averages from 71 1/2 kopecks (kopeck equals 1/2 cent) to 8 kopecks per pood (pood equals 36 lb.) at the pit's moutd, this being 1 1/2 to 1 3/4 kopecks per pood more than last year.

Sweden and Norway (By Dwight E. Woodbridge[1]).—The iron ore resources of Scandinavia are comprised chiefly in the pre-Cambrian rocks of Swedish Lappland and along the Norwegian coast from latitude 66 to and beyond North Cape, as far easterly as the frontier of Arctic Russia. In addition to deposits within these general boundaries are the fields of southern Sweden, situated some 200 km. north of west from Stockholm, and a few scattered deposits of more or less value in parts of Norway.

These ore fields are of great magnitude, and many of those of Sweden are of very high grade. Usually they are magnetites, and some are highly titaniferous. The latter are unworked. Interest in Scandinavian deposits naturally centers about the mines of Kiruna (Sweden) and Sydvaranger (Norway) which are the chief producers of the two countries.

All the ore fields of Swedish Lappland in which mining is now carried on are owned by the three companies, Luossavaara-Kirunavaara Aktiebolag, Aktiebolaget Freja and Tuolluvaara Grufaktiebolag. Of these three, the first is by far the greatest in its ore reserves, and it is owned half by a private company and half by the Swedish state. Freja is the property of Austrian steelmakers and the third mentioned is owned by a group of Swedish iron works. Practically all other iron deposits of

[1] See also *E. and M. J.*, Aug. 5, 1911, and June 24, 1911.

Lappland are the property of the Swedish state. In southern Sweden where are some 280,000 square meters of iron ore area, the Grangesberg field contains 90,000 square meters the rest being scattered. Of Grangesberg itself the bulk of the ore is controlled by the same company that is chief in the north. Other mines, as Persberg and Norrberg (low phosphorus) and Blotberg and Idkerberg (high phosphorus) are in the hands of smaller companies supplying Swedish works. Of Norwegian mines Sydvaranger is owned by a Swedish company, and Dunderland by British capital. The latter has been idle for some years but is about to resume operations.

A summary of the ore reserves of Sweden shows what may be classed as known reserves amounting to about 1,450,000,000 tons, most of which is of high grade; and this figure does not include the highly titaniferous ores of Taberg and elsewhere nor those scattered deposits whose reserves have not yet been sufficiently explored to give any detailed estimates. These may contain great quantities of good ore. The reserves of low-grade magnetites in Norway, suitable for concentration, may easily foot up to a total of 1,500,000,000 tons, in addition to which are many hundred square miles of iron-bearing rocks yet unexplored.

Of the two chief operations of the peninsula that at Kiruna is largely a form of quarrying, the ore being broken and dropped through raises into chutes and drawn off into railway cars at the main railway elevation beneath. The ore deposit varies in width up to a hundred meters and there is almost no stripping of overburden from its surface. This ore extends for several miles between walls of gray and red porphyry. At Gellivara, also in the Lappland district, the ore is mined underground by a system of magazine rooms, from which the broken ore is drawn through drifts as desired. At Sydvaranger quarrying of ore masses is the system, and here there is an overburden that averages a meter or so in thickness. In southern Sweden mining is underground, and many of the operations are equipped with very up-to-date appliances and machinery.

As Sydvaranger is the latest and one of the chief operations of its kind in the world, a more extended reference is warranted. The ore passes through a crushing process that grinds it in three operations, jaw crushing, ball and tube mills, to a fineness of 200 mesh. The iron is separated from its gangue by Grondal magnetic separators, arranged in series, and the 35 to 36 per cent. ore is delivered from the separators at a grade of from 65 per cent. upward. Losses of ore in the process of separation are slight. A considerable portion of the concentrates is shipped as fines, and the remainder as briquets.

During 1911, the Sydvaranger export was 100,000 metric tons, of which half was fine concentrates, the rest briquets. This was sent to

Germany and England, the former country taking all the fine concentrates and a part of the briquets. For the present year the company has contracted the shipment of 300,000 tons, of which two-thirds is fine concentrates. It is probable that this total will be increased by later sales. For the following year the company has already sold 500,000 tons, half in briquet form.

Up to date there has been invested in the enterprise a total of 27,000,000 kr. ($7,300,000) and it is estimated that the sale of 500,000 tons a year will yield interest on the investment. Three million kroner are now being invested as part in this capitalization, and this amount will bring the yield to the 500,000 tons mentioned. The works are to be extensively increased at once, with the view of making an annual product of not far from 1,000,000 metric tons of high-grade (65 per cent.) fine concentrates and briquets. According to calculations and investigations that have but now been completed by the company, it will be possible, by the investment of 11,000,000 kr. ($2,970,000) to raise the annual output to the desired amount.

Shipments from the export fields of Sweden during the year 1911 were as follows: Kirunavaara via Narvik, 2,170,000 metric tons; Cellivara via Lulea, 1,184,000 metric tons. Of this total quantity 211,000 tons from Narvik and 40,000 tons from Lulea came to the United States. Large sales of ore from both Kiruna and Sydvaranger have recently been made to this country.

For the Grangesberg field of Sweden during 1911 the total product of the controlling mines was 848,885 tons, of which 639,000 tons were exported, the remainder being used in Sweden. Of the latter 84,500 tons were used in the works of the Domnarports Jernfabrik.

Iron and Steel Markets in 1911

The striking incidents of the year were the first break in prices in May, when the Republic Iron & Steel Co. announced its intention of cutting prices and pursuing an aggressive policy for business. This was followed by a general cut in quotations, which was not, however, sufficient to induce a full recovery. The next was the later abandonment of the long struggle to maintain prices and the virtual acceptance of the open market and free competition.

The study of the market shows that great corporations may control production, but that they cannot yet entirely control the buyer. Conversely, that the time-honored rule of "boosting the market by getting under it" has not yet lost all of its truth and efficiency.

Railroad buying was light until nearly the close of the year, when

large orders for steel cars were placed. Renewals of track were generally kept down as much as possible. Building of new roads was not especially active. The *Railway Age-Gazette* reports the construction of new road in 1911 at 3067 miles, against 4122 in 1910. The same authority puts the number of new locomotives ordered at 2850, or 937 less than in 1910; of passenger cars at 2623, or 1258 less; of freight cars at 133,117, or 8087 less. Practically all the cars built in 1911 were of steel, or had steel frames.

The letters which follow, give the history of the year at Pittsburg, the great primary market; at Birmingham, the primary market of the South; at Chicago, the great distributing center of the Northwest, and in the seaboard iron markets.

Pittsburgh (By B. E. V. Luty[1]).—The steady decline in prices of all iron and steel commodities which characterized 1910 was arrested at the close of the year, apparently chiefly through the medium of a "Gary dinner" with a number of meetings in individual branches of the trade. The year 1911 opened with considerable promise, sufficient to increase the pig-iron-producing rate of the country from about 21,000,000 tons per year at the beginning of January to nearly 26,000,000 tons in March. Chiefly for the sentimental effect which it was believed such action would have, wire products were advanced on Jan. 23, by $1 a ton, to a basis of $1.75 for nails. As there were but three advances in finished steel products during the first 10 months of the year the other two may be mentioned here: An advance of 10 cents per box on tinplates to $3.70, base, effected at an informal meeting in Pittsburgh Feb. 3, and a second advance of $1 a ton, March 6, in wire products.

During the first 2 months of 1911 sentiment was fairly optimistic and buying was on a liberal scale, for early delivery. The iron trade was pleased with the improvement in general buying and looked forward to heavier buying by the railroads as soon as the "advanced rate cases" were decided. From what source it came no one seems to know, but word was passed around that the railroads would be permitted to make about half the total advances they had scheduled. The decisions were handled down Feb. 22 and disallowed all the advances. Then a chill seized the trade. The railroads were disgruntled, but before many months it was quietly admitted that the decision was a blessing in disguise forcing the railroads to find remedies in better ways for the conditions of which they complained.

On May 24, occurred a meeting of steel-bar manufacturers in New York and on the same day chairman Topping, of the Republic Iron & Steel Co., made public announcement that his company was not satisfied

[1] 545 Oliver Building, Pittsburgh, Pa.

with market conditions, having lost business through the cutting of some of the smaller mills. Accordingly the company would adopt "a more aggressive sales policy" and a price of 1.25 cents on steel bars was announced, $3 per ton under the nominal price at the time. This constituted a formal withdrawal of the company from the price-maintenance movement. On May 29, a general meeting of the steel manufacturers occurred, Republic not being represented. After the meeting it was announced that billets and sheet bars were reduced $2 per ton, sheets $4 per ton and plates and shapes $1 per ton, the cut in steel bars being ratified. The main purpose of the meeting was not known, whether to decide upon the extent of formal cuts which should be made, re-affirm allegiance to the agreement despite Republic's defection, or to arrange matters for a quiet and orderly abandonment of the whole coöperative movement. Prices were held for a time, but June 20 a reduction of $2 per ton occurred in wire products, restoring prices which prevailed at the opening of the year and at about the same time it developed that prices in other lines were being occasionally shaded. In July the Steel Corporation sought business very actively, at cut prices, but conducted its campaign so skillfully, making but slight cuts and then very quietly, that the independents did not, apparently, at first realize the extent to which the market had been opened. By the beginning of August it was universally recognized that there was an open market on finished steel products and that all semblance of coöperation or agreement had disappeared. Perhaps the trade was slow in recognizing the fact. The investigations of the Stanley committee of the House, commenced June 1, unquestionably made the previous position untenable.

Barring the reductions specifically mentioned, steel prices did not openly decline until late in July, but from that time until the close of November there was a continued succession of declines embracing all finished steel products except rails. The declines in May and June represented an average of between $1.50 and $2 per ton in the average of finished steel products, outside of rails, while the declines from July 1 to Dec. 1 represented further declines averaging $4 to $5 per ton. This brought the average level of finished steel products outside of rails, to a point between $12 and $13 per net ton below the level prevailing at the close of 1907, which was the highest since 1902, and only about $1 per ton above the average at the lowest point in the history of the steel business, in 1897–8. Some products, particularly sheets and steel pipe, were materially lower than ever before. This price level was practically profitless. It is about $4 per ton lower than the average during the third quarter of the year, for which quarter the United States Steel Corporation reported earnings of $29,522,725, and those earnings were considerably

larger than would have been expected from a comparison of earnings reported in 1906 and 1907, with allowance for the decline in prices and the change in tonnage output.

During the first half of the year the iron and steel market was subject to adverse political influences and the defection of the Republic Iron & Steel Co. with its attendant price declines. During the next 4 months it was subject to the disheartening influence of a constant decline in prices, prompting jobbers and consumers to withhold orders until the last moment and then buy in the smallest possible quantities. The fact that after a considerable decline production actually did increase, from a minimum rate of 21,400,000 tons of pig iron a year in July to 25,000,000 tons in October, did not create any confidence. The suit against the United States Steel Corporation sent a fresh chill over the market, and then the unexpected happened.

In about 3 weeks immediately following the bringing of the suit railroads placed orders for fully 40,000 cars, against only 20,000 in the preceding 4 months, and no more than 50,000 in the whole first half of the year. Prices continued to decline during November, but bookings were larger and at the close of the month prices began to advance. This encouraged still heavier buying in December. In both months the actual tonnages entered on mill books for rolling exceeded the shipments materially, while a very large tonnage was placed under contract for first-quarter and first-half delivery. Doubtless much of the business specified was placed with the clear knowledge on the part of buyers that

AVERAGE PRICES AT PITTSBURG, 1911.

	Pig Iron.			Ferro-manga-nese.	Steel.					Nails.	
	Bes-semer.	Basic.	No. 2 Found-ry.		Bes-semer Billets.	Beams.	Plates.	Bars.	Black Sheets No. 28.	Wire per Keg.	Cut per Keg.
Jan.	$15.90	$14.19	$14.65	$40.20	$23.00	1.40c	1.40c	1.40c	2.20c	$1.71	$1.60
Feb.	15.90	14.50	14.75	39.70	23.00	1.40	1.40	1.40	2.20	1.75	1.60
Mar.	15.90	14.65	14.90	38.25	23.00	1.40	1.40	1.40	2.20	1.79	1.63
Apr.	15.90	14.65	14.90	38.70	23.00	1.40	1.40	1.40	2.20	1.80	1.70
May.	15.90	14.30	14.66	38.45	22.02	1.40	1.40	1.37	2.20	1.80	1.60
June.	15.90	13.96	14.49	38.45	21.00	1.35	1.35	1.25	2.00	1.76	1.60
July.	15.90	13.60	14.40	38.80	21.00	1.35	1.34	1.24	2.00	1.70	1.60
Aug.	15.90	13.90	14.40	38.95	21.00	1.35	1.30	1.20	1.91	1.69	1.60
Sept.	15.90	13.57	14.40	38.45	20.60	1.33	1.28	1.18	1.90	1.65	1.58
Oct.	15.44	13.41	14.31	38.95	20.00	1.23	1.19	1.13	1.85	1.64	1.55
Nov.	14.92	13.18	14.15	39.00	19.50	1.14	1.11	1.07	1.84	1.56	1.51
Dec.	15.15	13.10	13.90	41.50	19.00	1.13	1.13	1.12	1.85	1.54	1.50
Year.	15.72	13.94	14.49	39.20	21.42	1.32	1.31	1.26	2.03	1.70	1.59
1910.	17.10	15.65	15.83	42.48	25.20	1.46	1.45	1.43	2.28	1.75	1.72
1909.	17.46	16.44	16.40	44.42	24.58	1.41	1.41	1.31	2.26	1.82	1.77

Prices of pig iron, ferromanganese and billets are per ton; of steel products, per pound; of nails per keg of 100 lb.

delivery could not be made until after Jan. 1. While no definite increase occurred in production, plans for holiday closings were largely abandoned, and orders were issued for the starting of considerable idle capacity immediately after Jan. 1. The recovery in prices ran from $1 to $3 per ton in the products affected, and represented a general advance in the level of finished steel products, outside of rails, of between $1 and $1.50 per ton.

Alabama Iron Market (By L. W. Friedman[1]).—The year 1911 in pig iron and steel in Alabama was far from being satisfactory. The make was good, but the price record rather disastrous. The year opened with the market rather weak for pig iron, the average price around $11 per ton, No. 2 foundry. The sales were not active by any means and the make not pushed. There started early in the year an accumulation of iron; in other words the make was greater than the demand. Until the eleventh month of the year, the sales were a little under or near production from month to month. Then, after the quotations had dropped at least $1 per ton, some interests made concessions on the price and started a liberal buying movement. A little strength was found in the market during the first part of December and a recovery was secured up to $10.25 to $10.50 which was under that price that prevailed when the year opened. The steel market was quite active during the better part of the year. However, toward the end of the year the market slacked off and there was a cry for orders. The year witnessed a good and steady demand for fabricated steel; a single rolling mill in the Birmingham district continued in operation through the 12 months. The bulk of this product was used by Southern manufactures.

Cast-iron pipe makers melted the larger part of the iron that was consumed in the home territory. There was a little setback during the year in the pipe trade, the Dimmick Pipe Co's plant shutting down and the property changing hands. For several months there was no work done at all at this plant, this having an effect on the year's melt in the home territory. The Dimmick plant was finally taken over by the United States Cast Iron Pipe & Foundry Co., extensive repairs made and it is promised that there will be a resumption on a large scale early in 1912. There was a steady demand for cast-iron pipe in 1911, and shipments were made long distances. Great quantities of pipe from the Birmingham district went to California points while the Middle West also looked to this section of the country for much cast-iron pipe.

Toward the end of the year stove makers in the Southern territory and in Alabama in particular took on some activity and bought some pig iron. Foundries and machine shops did not do a big business through

[1] Birmingham, Ala.

the year, the consumption of iron on their part being under the previous year.

Charcoal-iron manufactures in Alabama, with three and four small furnaces in operation from time to time, had a fairly good year in 1911. Despite the low prices prevailing in foundry iron, there was a good price maintained for charcoal pig, $22 to $22.50 per ton being asked right along. There was not much accumulation of stocks.

The pig-iron production during 1911 in Alabama fell short of the make in 1910. Estimating the production in December at 150,000 tons, the make in Alabama in 1911 is to be placed at 1,763,100 tons. The figures for 1910 show a production of 1,939,147 tons. There was a decrease in the production every month in 1911 as compared to the same month in 1910, with the exception of September. The total figures for 1911 may show a small increase in the revising that will take place by the middle or latter part of January, but in round numbers there will hardly be a change.

The year closes in an encouraging way with large sales. In the latter part of the year the Tennessee Coal, Iron & Railroad Co. was prominent as a seller of foundry iron, though most of its active furnaces ran on basic pig for the Ensley steel works.

The Chicago Iron Market (By E. Morrison).—Considered as a whole, the year 1911 was disappointing to Chicago dealers in pig iron and finished products.

In January pig iron sold lightly. The line between 1910 and 1911 was not marked by any change in buying conditions; melters bought in general for about 3 months ahead, preferring to contract for small lots—a car load to 500 tons—often and to make frequent inquiries for larger tonnage that showed their close watching for a return in the market.

Northern No. 2 pig iron sold for $15.50 to $16; Southern No. 2 for $11 to $11.50, Birmingham, for $14.35 to $14.85, Chicago. Lake Superior charcoal iron brought $18 to $18.50. Railroad supplies were in light demand and structural-material sales were confined to a few bridges. Plates, sheets and bars were in light demand. Buyers and sellers were in a waiting mood.

February saw the relation between production and consumption so improved as to cause the blowing in of four local furnaces and a demand of about 50 cents more for Southern iron for last-half than for early delivery. The founders, however, increased their buying only slightly. Rails and track supplies became in better demand and the spring contracts for structural material showed improvement. Plate specifications increased in volume of material. Sheets had a better sale. Bars improved to the extent of becoming in fairly good demand for agricultural-

machine needs. In the middle of the month the International Harvester Co. bought 100,000 tons, on deliveries extending into the third quarter, of Northern and Southern pig iron, the price being said to be slightly below $10, Birmingham.

In March, pig iron became stronger, Northern in particular strengthening after a sale of 10,000 tons to a Milwaukee foundry. Prices, however, did not advance on coke foundry irons, and Lake Superior charcoal dropped off to $17.50 to $18. Both buyers and sellers were still in a waiting mood. Sales of railroad material and structural materials became better; plates, sheets and bars also showed improvement, but production continued large for the demand.

April saw pig iron in very light demand and unchanged conditions continuing in the finished-materials market. Plate, sheet and bar mills were running at about half capacity.

May showed no revival in the demand for pig iron; the sales were chiefly in car-load lots. Founders bought very often but could not be induced to increase their contract time. Structural material became very quiet; the railroads bought reluctantly; plates, bars and sheets sagged in price.

June was a dull month in every line and the reduction in price of sheets had no stimulating effect. Everywhere buying was for immediate use. In July, sales of pig iron increased slightly, particularly of Southern, but the minimum price remained $10, Birmingham. Lake Superior charcoal dropped to $16.50 to $17. In finished lines only structural material had any fair demand.

August continued dull, with an effort by the Southern interests to obtain $10.50, Birmingham, but consumers refused to pay more than $10, at which price they obtained plenty of tonnage. Structural material sold well; other finished materials were in light demand and prices were slightly cut. In September, the demand for pig iron dropped back to car-load lots, with $10, Birmingham, on prompt shipments and $10.50 on more extended deliveries the rule. Prices of steel bars were cut very severely and remained low for the rest of the year.

IRON AND STEEL PRICES IN CHICAGO.

Material.	1909.		1910.		1911.	
	Highest.	Lowest.	Highest.	Lowest.	Highest.	Lowest.
Lake Superior Charcoal	$20.00	$19.50	$20.00	$18.00	$18.50	$16.50
Northern No. 2 Foundry	19.50	16.50	19.50	16.00	16.00	14.00
Southern No. 2 Foundry	19.85	15.35	18.85	15.10	15.85	14.35
Bar Iron	1.60c.	1.30c.	1.65c.	1.35c.	1.35c	1.15c.
Structural Material(a)	1.78c.	1.40c.	1.88c.	1.58c.	1.63c	1.30c.

(a) Beams and channels, 3 in. to 15 in., and angles 3 in. to 6 1/4 in. or heavier.

Soft steel bars ranged from 1.58 cents in January to 1.25 cents in October, the price strengthening toward the end of the year. Plates sold at 1.58 to 1.63 cents at the beginning of the year and sank to 1.33 cents at the end. Sheets declined from 2.38 cents in January to 2.03 cents in December, for No. 28 black.

October developed a little better condition for pig iron, but the demand was still light and the efforts of Southern interests to advance prices from $10, Birmingham, met with failure. All finished lines were dull. November saw pig iron very weak, with the end of the year bringing a slackening of purchases of finished products and generally low prices for them.

December brought a general improvement. Several large purchases of pig iron were made, Southern bringing $10 to $10.50 and Northern $14 to $14.50 on first-quarter deliveries, while Lake Superior charcoal iron held to $16.50 to $17. The railroads began to buy more freely and good sales were made of plates and bars.

The Seaboard Iron Market (*Special Correspondence*).—Philadelphia and New York are the central points of the seaboard market; Philadelphia being a primary market for a considerable district in Eastern Pennsylvania, while New York is a central point for a large contracting business. The eastern markets are large buyers of foundry iron, and also take a considerable quantity of basic pig, but comparatively little bessemer iron. They are also buyers of steel billets and of finished iron and steel, being especially the chief markets for structural material.

In pig iron the seaboard markets opened the year rather quietly. Buying, as in the latter part of 1910, was generally limited to immediate needs, few foundries caring to purchase far ahead. Stocks in foundry and mill yards were generally small and were kept so during the greater part of the year; users of iron, as a rule, left the burden of carrying stocks to the blast-furnaces. This plan necessitated frequent purchases and at times gave the market an appearance of greater activity than total bulk of the transactions actually warranted. An effort was made to keep up prices of foundry iron, with some success during the first quarter; but by April the weight of unsold stocks in furnace yards and the pressure of Southern iron began to be felt, and general concessions were made, which could not afterward be recovered. By June the general level of foundry iron was 50 to 75 cents a ton lower than the moderate prices which were asked in January. Forge iron dragged throughout the first part of the year, and in fact it was a buyers' market in this grade from January to November. It was impossible to ask advances, under the prevailing conditions of large stocks and idle capacity. In July some good buying by pipe foundries gave a little boost to the market, but it

did not last long enough to have any great effect. Buyers continued to take mainly small lots for early delivery, and were inclined to haggle over small reductions in prices.

With all these adverse conditions the business done reached a pretty good total, and the unsold stocks at the opening of the year were gradually reduced. Nevertheless business had to be characterized as rather uncertain and no important change came until November, when many people began to think that the bottom had been reached, and that an upward turn might be expected. This feeling brought about increased buying and the placing of contracts for the first quarter of 1912. Some sales

PRICES OF STEEL AND IRON AT TIDEWATER.

	Dec. 29, 1909.	Dec. 28, 1910.	Dec. 30, 1911.
g Iron:			
No. 2 X foundry, Philadelphia, ton	$19.00	$15.50	$14.85
No. 2 Southern, New York, ton	18.75	15.50	14.75
Basic, Philadelphia, ton	18.75	14.75	14.25
illets:			
Open-hearth, seaboard, ton	30.50	24.50	22.40
nished Iron and Steel:			
Refined iron bars, tidewater, lb	1.65c.	1.35c.	1.28c.
Steel bars, tidewater, lb	1.66c.	1.56c.	1.31c.
Tank plates, tidewater, lb	1.71c.	1.56c.	1.31c.
Beams and angles, tidewater, lb	1.71c.	1.56c.	1.31c.
Wire nails, tidewater, keg	$2.05	$1.90	$1.65
d Material:			
Old iron rails, Philadelphia, ton	$20.50	$17.00	$16.50
Old car wheels, Philadelphia, ton	17.50	13.00	12.50
Heavy steel scrap, Philadelphia, ton	17.00	12.50	12.25

ere made covering the first half, but users of iron for the most part were ow to give up the plan of short buying. Moreover, a special drive was ade to sell Southern iron, and this was a weight upon the market. For ome months $10 Birmingham had been the accepted rate for No. 2 oundry, but at this time some considerable orders were placed at $9.75 nd even $9.50 for early delivery. The sales then made relieved the nmediate necessities of sellers, and a restoration to $10, with $10.25 for 912 business, soon followed. The year closed with prices generally a hade firmer on all classes of pig iron.

Cast-iron pipe makers were generally busy during the year, and had a ood trade, though many contracts were taken at low figures. The low rices stimulated city and public utility work, and sales of pipe were arge and steady.

In finished material structural steel was easily the leader in the astern markets through the greater part of the year, and it was not until October or November that other lines caught up fully. The reduction in rices which followed the break made by the Republic Iron & Steel Co. n May brought some additional business but it was not until the controlled

prices were definitely abandoned and the open market became general that sales began to increase. As has been the case several times, the large fabricating companies began to take contracts at a rate which involved cheaper steel some time before the general break in prices actually came, and this kept structural steel in the lead. On the fall movement, however, the buying of other materials gained rapidly and then there was a general disposition to take advantage of the lower prices. Small building was stimulated and the purchases of material for this class of work reached a large total. The small buyer is on the lookout for low prices, and has more than once helped out the market in a notable way, since the aggregate of his purchases may be large.

Perhaps the most important of these was for the bridge over the East River and approaches for the New York Connecting R. R., which involves some 65,000 tons of steel. The new subways in New York, terminal and transit improvements in Philadelphia and proposed improvements in Baltimore will require large quantities of steel.

Iron and Steel Prices

The accompanying table, based largely on the records of the American Iron and Steel Association, gives the range of prices of leading articles for 50 years past; it is well worth attention.

IRON AND STEEL PRICES FOR FIFTY YEARS.
(In tons of 2240 lb., except nails, which are in kegs of 100 lb.)

Year.	Pig Iron.		Bar Iron, Best.	Rails.		Nails.		Year.	Pig Iron.		Bar Iron, Best.	Rails.		Nails.	
	No. 1 F'dry.	Bessemer.		Iron.	Steel.	Cut.	Wire.		No. 1 F'dry.	Bessemer.		Iron.	Steel.	Cut.	Wire.
	$	$	$	$	$	$	$		$	$	$	$	$	$	$
1862	23.92	.....	70.42	42.38	.....	3.47	.....	1887.	20.93	21.37	43.59	.....	37.08	2.30	3.15
1863	35.24	.....	91.04	41.75	.....	5.13	.....	1888.	18.88	17.38	39.67	.....	29.83	2.03	2.55
1864	59.22	.....	146.46	76.88	.....	7.85	.....	1889.	17.76	18.00	38.30	.....	29.25	2.00	2.49
1865	46.08	.....	106.46	126.00	.....	7.08	.....	1890.	18.41	18.87	41.25	.....	31.78	2.00	2.51
1866	46.84	.....	98.13	98.63	.....	6.97	.....	1891	17.52	15.95	38.38	.....	29.92	1.86	2.04
1867	44.08	.....	87.08	86.75	166.00	5.92	.....	1892.	15.75	14.37	36.79	.....	30.00	1.83	1.70
1868	39.25	.....	85.63	83.13	158.46	5.17	.....	1893.	14.52	12.87	33.53	.....	28.12	1.44	1.49
1869	40.61	.....	81.67	78.88	132.19	4.85	.....	1894.	12.66	11.38	26.88	.....	24.00	1.08	1.11
1870	33.23	.....	78.96	77.25	106.79	4.40	.....	1895.	13.10	12.72	28.09	.....	24.33	1.56	1.69
1871	35.08	.....	78.54	72.25	102.52	4.52	.....	1896.	12.95	12.14	27.22	.....	28.00	2.36	2.54
1872	48.94	.....	97.63	70.38	111.94	5.46	.....	1897.	12.10	10.13	24.73	.....	18.75	1.47	1.46
1873	42.79	.....	86.43	85.13	120.58	4.90	.....	1898.	11.66	10.33	23.93	.....	17.62	1.31	1.45
1874	30.19	.....	67.95	76.87	94.28	3.99	.....	1899.	19.36	19.03	43.75	.....	28.12	2.21	2.60
1875	25.53	.....	60.85	58.75	68.75	3.42	.....	1900.	19.98	19.49	48.12	.....	32.29	2.46	2.76
1876	22.19	.....	52.85	47.75	59.25	2.98	.....	1901.	15.87	15.93	40.38	.....	27.33	2.29	2.41
1877	18.92	.....	45.55	41.25	45.58	2.57	.....	1902.	22.19	20.67	43.53	.....	28.00	2.29	2.15
1878	17.67	.....	41.24	35.25	42.21	2.31	.....	1903.	19.92	18.98	39.59	.....	28.00	2.36	2.13
1879	21.72	.....	51.85	33.75	48.21	2.69	.....	1904.	15.57	13.76	33.17	.....	28.00	2.01	1.96
1880	28.48	.....	62.04	41.25	67.52	3.68	.....	1905	17.88	16.36	41.89	.....	28.00	2.00	1.93
1881	25.17	.....	58.05	49.25	61.08	3.09	.....	1906.	20.98	19.54	43.23	.....	28.00	2.13	1.98
1882	25.77	.....	54.51	47.13	48.50	3.47	.....	1907.	22.40	19.43	40.87	.....	28.00	1.86	1.82
1883	22.42	.....	44.24	45.50	37.75	3.06	.....	1908.	17.24	17.23	39.87	.....	28.00	1.83	1.99
1884	19.81	.....	38.45	.....	30.75	2.39	.....	1909.	17.46	16.40	39.42	.....	28.00	1.82	1.77
1885	17.99	.....	36.59	.....	28.52	2.33	.....	1910.	17.10	15.83	31.50	.....	28.00	1.75	1.72
1886	18.71	18.96	38.08	.....	34.52	2.27	.....	1911.	14.49	15.72	26.80	.....	28.00	1.72	1.69

Pig Iron Production for 100 *Years*

The accompanying table, is compiled from the records of the American Iron and Steel Association.

PIG IRON PRODUCTION FOR 100 YEARS.

1820.....	20,000	1858.....	629,548	1876.....	1,868,961	1894.....	6,657,388
1828.....	130,000	1859.....	750,650	1877.....	2,066,594	1895.....	9,446,308
1829.....	142,000	1860.....	821,223	1878.....	2,301,215	1896.....	8,623,127
1830.....	165,000	1861.....	653,164	1879.....	2,741,853	1897.....	9,652,680
1831.....	191,000	1862.....	703,270	1880.....	3,835,191	1898.....	11,773,934
1832.....	200,000	1863.....	846,075	1881.....	4,144,254	1899.....	13,620,703
1840.....	286,903	1864.....	1,014,282	1882.....	4,623,323	1900.....	13,789,242
1842.....	215,000	1865.....	831,770	1883.....	4,595,510	1901.....	15,878,354
1846.....	765,000	1866.....	1,205,663	1884.....	4,097,868	1902.....	17,821,307
1847.....	800,000	1867.....	1,305,023	1885.....	4,044,526	1903.....	18,009,252
1848.....	800,000	1868.....	1,431,250	1886.....	5,683,329	1904.....	16,497,033
1849.....	650,000	1869.....	1,711,287	1887.....	6,417,148	1905.....	22,992,380
1850.....	563,755	1870.....	1,665,179	1888.....	6,489,738	1906.....	25,307,191
1852.....	500,000	1871.....	1,706,793	1889.....	7,603,642	1907.....	25,781,361
1854.....	657,337	1872.....	2,548,713	1890.....	9,202,703	1908.....	15,936,018
1855.....	700,159	1873.....	2,560,963	1891.....	8,279,870	1909.....	25,795,471
1856.....	788,515	1874.....	2,401,262	1892.....	9,157,000	1910.....	27,303,567
1857.....	712,640	1875.....	2,023,733	1893.....	7,124,502	1911.....	23,649,344

Imports and Exports

The accompanying tables show the values and—as far as possible the quantities—of the imports and exports of the United States.

UNITED STATES EXPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Pig iron.....	49,025	49,221	83,317	73,844	46,696	61,999	127,385	120,799
Billets, blooms, etc.....	314,324	237,638	192,616	79,991	112,177	104,862	58,230	234,767
Bars.....	55,472	51,870	88,102	98,654	46,103	87,960	125,606	141,033
Rails.....	414,845	295,023	328,036	338,906	196,510	299,540	353,180	420,874
Sheets and plates.....	55,204	75,034	110,654	122,696	104,993	180,048	274,516	372,373
Structural steel.....	55,514	83,193	112,555	138,442	116,878	90,830	146,721	223,493
Wire.....	118,581	142,601	174,014	161,228	136,167	149,341	171,928	229,316
Wire-rods.....	20,073	6,514	5,896	10,653	7,412	20,142	22,869	22,641
Nails and spikes.....	45,112	47,756	59,491	56,826	38,906	48,055	61,201	77,833
Pipe and fittings.....	.....	.....	.....	176,832	114,371	162,140	155,728	197,597

IRON AND STEEL EXPORTS AND IMPORTS, UNITED STATES.
TOTAL VALUE. (*a*)

	1906.	1907.	1908.	1909.	1910.	1911.
Exports.....	$172,555,588	$197,036,781	$151,113,114	$157,680,331	$201,271,903	$249,656,411
Imports.....	34,827,132	38,789,992	19,957,261	30,571,542	38,907,119	28,995,600
Excess, exports:	$137,728,456	$158,246,789	$131,155,853	$127,108,789	$162,364,784	$220,660,811

(*a*) Including machinery.

The increase in value of exports in 1910 over 1909 was 27.5 per cent.; in 1911 it was 24 per cent. over 1910. The exports were, in 1911, the largest ever reported in a single year. Part of the gain was due to the

steady extension of our foreign trade, chiefly by the agencies of the Steel Corporation; and part was due to the large shipments of material for the Panama Canal works.

UNITED STATES IMPORTS OF IRON AND STEEL.

(In tons of 2240 lb.)

	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Pig iron	599,574	79,500	212,465	379,828	489,440	92,202	174,988	237,233	148,459
Billets, blooms, etc.	261,570	10,807	14,637	21,337	19,334	12,112	19,913	46,578	29,207
Scrap iron and steel	82,921	13,461	23,731	19,091	27,687	5,090	63,504	72,764	17,272
Bars	43,393	20,905	37,298	35,793	39,746	19,672	19,206	43,692	26,730
Rails	95,555	37,776	17,278	4,943	3,752	1,719	1,513	.....	.....
Wire-rods	20,836	16,206	17,616	17,999	17,076	11,208	10,544	20,374	15,483
Tinplates	47,360	71,304	65,740	56,983	57,773	58,320	62,593	66,640	14,099

Two incidents in the export trade deserve special mention. One is the large increase in billets, which was chiefly in the form of flat billets or tinplate bars, the steel corporation having secured large contracts for the supply of such material in Great Britain. The other was the complete reversal of the course of the tinplate trade. Imports of tinplates fell from 68,640 tons in 1910 to 14,099 in 1911; while exports rose from 12,445 tons in 1910 to 61,466 in 1911. The reason for this change was that American manufacturers succeeded in capturing most of what is known as the "drawback trade," that is, the tinplate used in boxing or casing oil and other goods for export, a drawback being allowed on foreign tinplate used for such purpose. The competition is open, and until 1911, the Welsh makers had a very large part of the business.

United States Steel Corporation

Toward the close of 1911 the Gary plant was completed as far as construction had been authorized. The sheet bar mill of the Indiana Steel Co. was completed June 29, the two 12- and one 10-in. merchant bar mills about Sept. 1; the 60-in. plate mill on Nov. 4, 1911. Seventy by-product coke ovens went into operation April 12, and four similar batteries during 1911. The third open-hearth unit with 14–65 tons furnaces was placed in operation early in 1912. Of the new sheet plants the American Bridge Co. commenced operations during the year.

On the coast of the purchase the Risdon Iron and Locomotive Works will make a supply center. The Duluth steel plant progressed and $1,437,518 was expended on terminal railroads, and $1,037,969 on coal property in Illinois and Indiana.

In order to insure to the subsidiary companies operating blast furnaces in the Pittsburg, Valley and Cleveland districts, a proper supply of coke

the H. C. Frick Coke Co. acquired from the Pittsburg Coal Co. and the Monongahela River Consolidated Coal & Coke Co. on July 1, 1911, a number of coking coal properties.

The foreign export business was 18.2 per cent. of the total shipments of finished steel. Costs of manufacture are not given, but exclusive of cement and other by-products the total receipts are approximately $64.92, the net earnings $11.07 and the dividends paid $5.33 per ton.

During 1911, $33,141,000 in bonds of subsidiary companies were sold in payment for construction and maturing issues. Deducting bonds and mortgages paid off the bonded debt increased $23,917,641. The inventory valuations were $153,483,589 an increase of $10,650,204 in 1911. Gross receipts in this year were $88,812,585 less than in 1910.

For permanent pension funds $663,018 was set aside from the accumulated undivided surplus, in addition to $1,000,000 previously set aside.

PRODUCTION OF THE U. S. STEEL CORPORATION.
Including Tennessee Company in 1907, 1908, 1909 and 1910.

	1906.	1907.	1908.	1909.	1910.	1911.
Iron Ore Mined—	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
From Marquette Range	1,442,290	1,170,496	830,087	899,002	830,094	560,685
From Menominee Range	1,874,680	1,625,358	1,021,598	1,359,415	1,386,465	1,105,044
From Gogebic Range	1,465,375	1,425,457	1,078,025	1,312,701	1,801,480	1,264,734
From Vermillion Range	1,794,186	1,724,217	927,206	1,066,474	1,338,110	1,182,075
From Mesabi Range	14,068,617	16,458,273	11,272,397	16,968,592	17,910,366	14,581,530
In Southern Region	.....	1,576,757	1,533,402	1,824,863	1,981,301	1,239,563
Total	20,645,148	23,980,558	16,662,715	23,431,047	25,245,816	19,933,631
Coke Manufactured	13,295,075	13,544,764	8,169,931	13,590,112	13,649,578	12,120,212
Coal Mined, not used in making coke	1,912,144	3,550,510	3,008,810	3,089,021	4,850,111	5,290,671
Limestone Quarried	2,227,436	3,201,222	2,186,007	4,715,708	5,005,087	4,835,703
Blast Furnace Products—						
Pig iron	11,058,526	11,234,447	6,810,831	11,436,570	11,645,510	10,593,726
Spiegel	150,044	130,554	74,716	80,942	102,260	66,435
Ferro-manganese and Silicon	58,807	57,794	48,861	100,838	83,628	34,736
Total	11,267,377	11,422,795	6,934,408	11,618,350	11,831,398	10,744,897
Steel Ingot Production—						
Bessemer ingots	8,072,655	7,556,460	4,055,275	5,846,300	5,796,223	5,055,696
Open-hearth ingots	5,438,494	5,786,532	3,783,438	7,508,889	8,383,146	7,697,674
Total	13,511,149	13,342,992	7,838,713	13,355,189	14,179,369	12,753,370
Rolled and Other Finished Products for Sale—						
Steel rails	1,982,042	1,879,985	1,050,389	1,719,486	2,118,473	1,568,028
Blooms, Billets, Slabs, Tinplate Bars	1,096,727	761,195	551,106	675,614	682,364	874,474
Plates	836,399	894,364	312,470	729,790	929,020	630,512
Heavy Structural Shapes	620,823	587,954	313,733	658,516	656,797	547,186
Merchant Steel, Skelp, Hoops, Bands	1,240,548	1,338,833	577,591	1,290,970	1,527,506	1,221,606
Tubing and Pipe	1,025,913	1,174,629	654,428	1,013,071	868,550	863,670
Rods	111,488	126,095	93,406	139,149	133,722	118,302
Wire and Products of Wire	1,399,717	1,481,226	1,275,785	1,607,689	1,490,318	1,613,754
Sheets—Black, Galvanized and Tinplate	1,112,542	1,070,752	770,321	1,024,985	1,082,787	1,079,046
Finished Structural Work	643,622	719,887	403,832	530,766	589,228	518,399
Angle and Splice Bars and Joints	176,730	195,157	84,669	190,226	235,998	160,855
Spikes, Bolts, Nuts and Rivets	70,233	67,991	40,252	72,076	71,326	60,386
Axles	181,913	189,006	24,057	68,366	101,066	52,046
Sundry Iron and Steel Products	79,736	77,463	54,893	138,956	246,840	167,984
Total	10,578,433	10,564,537	6,206,932	9,857,660	10,733,995	9,476,248
Spelter	28,884	31,454	28,057	27,853	26,777	28,333
Copperas (Sulphates of Iron)	21,933	24,540	26,411	33,582	33,684	28,381
Universal Portland Cement	2,076,000	2,129,700	4,535,300	5,786,000	7,001,500	7,737,500

Coke made from by-product ovens increased from 2,008,473 tons in 1910 to 2,629,006 tons in 1911.

The total gross sales and earnings were $615,148,840 and the net earnings for the year $104,305,466.

Pig Iron

The production in 1911 was greater than that of any year prior to 1906; but it was exceeded in every year since that, except in the disastrous period of 1908. While exact figures on consumption are not to be had, the closest possible estimate puts the stocks of unsold or unused pig iron in makers' hands at the close of 1910 as 2,100,000 tons, an increase of 1,700,000 tons during that year. In 1911 these stocks were decreased by something like 1,250,000 tons, standing at about 850,000 tons at the end of the year.

Number of Furnaces

The whole number of furnaces in blast on Dec. 31, 1911, was 231, against 212 on June 30, 1911, and 206 on Dec. 31, 1910. The number of furnaces idle on Dec. 31, 1911, including furnaces being rebuilt, was 235, gainst 261 on June 30, 1911, and 268 on Dec. 31, 1910. During the last six months of 1911 the number of furnaces actually in blast during a part or the whole of the period was 275, against 297 in the first half of the year. In the last half of 1910, 332 were active, against 374 in the first half.

The classification according to uses for which the iron was intended is as follows:

PIG IRON PRODUCTION OF THE UNITED STATES.

(In tons of 2240 lb.)

Kind of Iron.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Foundry and forge	5,837,174	5,714,492	6,397,777	4,307,734	6,386,833	6,352,379	5,256,815
Bessemer pig	12,407,116	13,840,518	13,231,020	7,216,976	10,557,370	11,245,642	9,409,107
Basic pig.....	4,105,179	5,018,674	5,375,219	4,010,144	8,250,225	9,084,608	8,520,029
Charcoal.....	352,928	433,007	437,397	249,146	376,003	396,507	278,676
Spiegel and ferro.	289,983	300,500	339,348	152,018	225,040	224,431	184,717
Total.....	22,992,380	25,307,191	25,781,361	15,936,018	25,795,471	27,303,567	23,649,344

Included in the bessemer pig in 1911 is 282,460 tons special low-phosphorus iron, and this was the only increase in any class of iron, except ferromanganese. The make of ferromanganese in 1911 was 74,482 tons, 3106 tons more than in 1910. Spiegeleisen decreased, however, the production in 1911 being 110,235 tons, 42,820 tons less than in the previous year. Included in charcoal iron are small quantities

of pig iron and of ferrophosphorus, ferrotitanium, ferrovanadium and other alloys made in the electric furnace.

The notable point in the classification is the decrease in the proportion of bessemer pig and the increase in basic iron.

The producing states west of the Mississippi are Missouri, Colorado, Washington and California. In 1911 Pennsylvania remained the leading state, making 41.5 per cent. of the total. It was followed in order by Ohio, Illinois, Alabama and New York.

PRODUCTION OF PIG IRON BY STATES.
(In tons of 2240 lb.)

States.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Massachusetts and Connecticut	15,987	20,239	19,119	13,794	18,388	16,582	9,649
New York	1,198,068	1,552,659	1,659,752	1,019,425	1,733,675	1,938,407	1,562,756
New Jersey	311,039	379,390	373,189	225,372	294,474	264,781	40,663
Pennsylvania	10,579,127	11,247,869	11,348,549	6,987,191	10,918,824	11,272,323	9,806,834
Maryland	322,096	386,709	411,833	183,502	286,856	326,214	255,816
Virginia	510,210	483,525	478,771	320,458	391,134	444,976	293,642
Alabama	1,604,062	1,674,848	1,686,674	1,397,014	1,763,617	1,939,147	1,712,443
N. Car., Geo. & Texas	38,699	92,599	55,825	24,345	26,072	14,725	1,200
West Virginia	298,179	304,534	291,066	65,551	228,282	174,661	291,472
Kentucky	63,735	98,127	127,946	45,096	86,371	100,509	95,202
Tennessee	372,692	426,874	393,106	290,826	333,845	397,569	324,648
Ohio	4,586,110	5,327,133	5,250,687	2,861,325	5,551,545	5,752,112	5,310,310
Illinois	2,034,483	2,156,866	2,457,768	1,691,944	2,467,155	2,675,646	2,108,002
Michigan	288,704	369,456	(a)436,507	(a)348,096	(a)964,289	(a)1,250,103	1,163,932
Wisconsin & Minn	351,415	373,323	322,083	148,938	348,177	307,200	276,807
W. of Miss. River	407,774	413,040	468,486	313,071	382,767	428,612	395,968
Total	22,992,380	25,307,191	25,781,361	15,936,018	25,795,471	27,303,567	23,649,344

(a) Includes Indiana.

PRODUCTION OF PIG IRON BY DISTRICTS.
(In tons of 2240 lb.)

	1906.	1907.	1908.	1909.	1910.	1911.
N. England, N. Y. and N. J.	1,952,288	2,052,060	1,258,661	2,046,537	2,219,770	1,613,068
Pennsylvania	11,247,869	11,348,549	6,987,191	10,918,824	11,272,323	9,806,834
Ohio, Ill., Mich., Wis. & Minn.	8,226,778	8,467,045	5,050,303	9,331,166	9,986,061	8,859,051
Maryland	386,709	411,833	183,502	286,856	326,214	255,816
Southern States	3,080,507	3,033,388	2,143,290	2,829,321	3,071,587	2,718,607
West of the Mississippi	413,040	468,486	313,071	382,767	428,612	395,968
Total	25,307,191	25,781,361	15,936,018	25,795,471	27,303,567	23,649,344

PIG IRON PRODUCTION ACCORDING TO THE FUEL USED.
(In tons of 2240 lb.)

Fuel Used.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Coke(a)	20,964,937	23,313,498	23,972,410	15,331,863	24,721,037	26,257,978	23,141,093
Anthracite & coke	1,674,515	1,535,614	1,335,286	318,741	682,383	628,579	212,548
Anthracite alone		25,072	36,268	36,268	16,048	20,503	17,027
Charcoal	352,928	433,007	(b)437,397	(b)249,146	(b)376,003	396,507	278,676
Charcoal and coke							
Total	22,992,380	25,307,191	25,781,361	15,936,018	25,795,471	27,303,567	23,649,344

(a) Under coke furnaces are included the very few which use raw bituminous coal. It may be assumed that 99 per cent. of this class of iron was made with coke. (b) Includes a small quantity made by the electric furnace.

CONSUMPTION OF PIG IRON IN THE UNITED STATES.
(In tons of 2240 lb.)

	1907.	1908.	1909.	1910.	1911.
Production	25,781,361	15,936,018	25,795,471	27,298,545	23,823,046
Imports	489,440	92,202	174,988	237,233	143,000
Decrease in stocks	.....	.....	.....	.....	1,250,000
Total	26,270,801	16,028,220	25,970,459	27,535,778	25,216,046
Exports	73,844	46,696	61,999	127,385	119,000
Increase in stocks	.....	.....	.....	1,700,000	.....
Approximate consumption.	26,196,957	15,981,524	25,908,460	25,708,393	25,097,046

The consumption for 1911 was approximately 594 lb. per capita, being 45 lb. less than in the preceding year.

Raw bituminous coal is used by only two or three small furnaces; they are included under coke. The very large decrease in anthracite iron is due to the fact that the furnaces using that fuel are chiefly merchant stacks in eastern Pennsylvania which can compete profitably with the Central West and the South only when prices are at a comparatively high level; and they are, therefore, usually the first to quit in slack times.

Wrought Iron

The sphere of rolled iron has constantly narrowed, and the actual production in America has been on the decrease since about 20 years ago. Statistics published by the American Iron and Steel Association show the decrease in the number of puddling furnaces and in the production of rolled iron. These tables are reproduced in the article by Bradley Stoughton on metallurgy. It is to be observed that a large proportion —probably from one-half to three-quarters—of this tonnage is made by the busheling and rerolling of wrought iron scrap, and is not the new product of the puddling furnace.

In 20 years as shown by the table on page 441, the number of puddling furnaces decreased practically one-half. At the same time the product deteriorated as is explained. The tonnage per furnace, it will be noted by comparing the figures with the annual production given on page 442, has materially increased, whereas the total tonnage of wrought iron produced has fallen off about one-fifth. The most notable fact is that 20 years ago 45 per cent. of the total rolled material was wrought iron, to-day the latter is only 8 per cent.

Steel

The total production of steel in the United States, in 1911, was 9.2 per cent. less than 1910.

The make of steel last year, as shown in the tables below, was again marked by an important increase in the proportion of open-hearth steel. Another noteworthy fact is that the make of steel in 1911 exceeded that

STEEL PRODUCTION FOR 13 YEARS.
(In tons of 2240 lb.)

	Acid.								Basic.		Total.
	Converter.		Open-hearth.		Crucible, etc.		Total.		Open-hearth.		
	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.
1899	7,586,354	71.3	866,890	8.1	106,187	1.0	8,559,431	80.4	2,080,426	19.6	10,639,857
1900	6,684,770	65.6	853,044	8.4	105,424	1.0	7,643,238	75.0	2,545,091	25.0	10,188,329
1901	8,713,302	64.7	1,037,316	7.7	103,984	0.7	9,854,602	73.1	3,618,993	26.9	13,473,595
1902	9,138,363	61.2	1,191,196	8.0	121,158	0.7	10,450,717	69.9	4,496,533	30.1	14,947,250
1903	8,592,829	59.1	1,094,998	7.5	112,238	0.8	9,800,065	67.4	4,734,913	32.6	14,534,978
1904	7,859,140	56.7	801,799	5.8	92,581	0.7	8,755,520	63.2	5,106,367	36.8	13,859,887
1905	10,941,375	54.6	1,155,648	5.8	111,196	0.6	12,208,219	61.0	7,815,728	39.0	20,023,947
1906	12,275,830	52.5	1,321,653	5.6	141,893	0.6	13,739,376	58.7	9,658,760	41.3	23,398,136
1907	11,667,549	49.9	1,269,773	5.5	145,309	0.6	13,082,631	56.0	10,279,315	44.0	23,361,946
1908	6,116,755	43.7	696,304	4.9	69,763	0.5	6,882,634	49.4	7,140,425	50.9	14,023,247
1909	9,330,783	39.0	1,076,464	4.5	130,302	0.5	10,537,549	44.0	13,417,472	56.0	23,955,021
1910	9,412,772	36.1	1,212,180	4.6	177,638	0.7	10,802,599	41.4	15,292,329	58.6	26,094,919
1911	7,947,849	33.6	912,718	3.8	99,897	0.4	8,960,464	37.8	14,685,932	62.2	23,675,501

MAKE OF ACID AND BASIC STEEL.
(In metric tons.)

	1909.		1910.		1911.	
	Acid.	Basic.	Acid.	Basic.	Acid.	Basic.
United States...	10,706,149	13,632,152	10,975,431	15,537,006	8,860,567	14,685,932
Germany	564,802	11,485,032	542,616	13,155,992	571,254	14,308,665
Total.....	10,270,951	25,117,184	11,518,047	28,692,998	9,431,821	28,994,597

STEEL PRODUCTION OF UNITED STATES AND GERMANY.
(In metric tons.)

Kind of Steel.	1909.		1910.		1911.	
	United States.	Germany.	United States.	Germany.	United States.	Germany.
Acid converter.....	9,480,076	151,148	9,563,376	171,108	7,947,849	187,359
Basic converter.....	.....	7,517,451	.....	8,030,571	.....	8,640,164
Total converter.....	9,480,076	7,668,599	9,563,376	8,047,679	7,947,849	8,827,523
Acid open-hearth.....	1,093,687	311,812	1,231,575	252,148	912,718	383,895
Basic open-hearth.....	13,632,152	3,967,581	15,537,006	5,125,421	14,685,932	5,668,501
Total open-hearth.....	14,724,839	4,279,393	16,768,581	5,377,569	15,598,650	6,052,396
	109,072	84,069				
Crucible.....	23,314	17,773	124,260	83,202	99,897	78,760
Electric, etc.....	.....	.....	56,220	36,158	29,105	60,654
Total.....	24,338,301	12,049,834	26,512,437	13,698,638	23,675,501	15,019,333
Proportions steel to pig iron.	92.9	93.3	95.6	92.6	93.6	96.8

of pig iron by 46,157 tons. This happened only in one or two European countries—as in Italy—where a few large works make steel from imported pig iron. A large part of the heavy stocks of pig iron, estimated at more than 2,000,000 tons at the close of 1910 was used for conversion into steel in 1911. During a large part of the year there was an abundant and relatively cheap supply of steel scrap, which led to the use of an usually large proportion of scrap in the open-hearth furnaces. This second fact is confirmed by the fact that the decrease in basic open-hearth steel, as compared with 1910, was proportionally less than that in basic pig iron.

Rails.—Rails rolled from purchased blooms, crop ends and seconds, and rerolled and renewed steel rails are included in the following table from the American Iron & Steel Association. Included in the total for 1911 are 205,409 tons of girder and high T steel rails for electric and street railways.

Included in the miscellaneous rails rolled in 1911 are 462 tons of rails rolled from electric steel, and 6538 tons of rerolled steel rails which the makers were unable to classify as bessemer or open-hearth rails. Of the total production of steel rails in 1911, 2,780,795 tons were rolled from ingots made by the makers and 113,761 tons were rolled from purchased ingots or blooms, crop ends, seconds or renewed or rerolled rails.

The production in 1906 was the largest ever reported, 3,977,887 tons. The proportion of bessemer rails to the total fell from 51.8 per cent. in 1910 to 40.3 in 1911, while that of open-hearth rails increased from 48.2 to 59.4 per cent. Almost all of the open-hearth rails in 1911 were rolled from basic steel, and all were rolled by the producers of ingots. Rails from electric steel were reported in 1911 for the first time.

RAIL PRODUCTION IN THE UNITED STATES.

(In tons of 2240 lb.)

Material.	1909.	1910.	1911.	Section.	1909.	1910.	1911.
Bessemer steel	1,806,621	1,884,442	1,138,633	Under 45 lb.	255,858	260,709	218,758
Open-hearth steel	1,255,961	1,751,359	1,676,923	45 to 85 lb.	1,041,184	1,275,339	1,067,696
Iron and miscell.	.....	230	7,234	Over 85 lb.	1,765,540	2,099,983	1,536,336
Total	3,062,582	3,636,031	2,822,790	Total	3,062,582	3,636,031	2,822,790

Foreign Countries

The iron and steel industries were generally active abroad, and most foreign countries showed gains in production. The tables herewith give statistics of all the more important countries on this continent and in Europe.

STEEL PRODUCTION OF AUSTRIA-HUNGARY.
(In metric tons.)

Steel.	1910.			1911.		
	Austria.	Hungary.	Bosnia.	Austria.	Hungary.	Bosnia.
nverter	735	34,139	.....	360	34,058	.....
onverter	297,500	.....	.....	293,750	.....	.....
earth	1,170,138	600,152	33,539	1,268,907	675,477	35,508
d	14,372	182	.....	.....	.....	.....
e	16,275	1,311	.....	30,655	1,326	.....
;	19,672	356	.....	21,706	1,261	.....
l steel	1,518,692	636,140	33,539	1,615,378	712,122	35,508
d (wrought) iron	65,260	17,533	.....	91,286	11,463	.....
l	1,583,952	653,673	33,539	1,706,644	723,585	.....

PIG IRON PRODUCTION IN BELGIUM.
(In metric tons.)

	1905.	1906.	1907.	1908.	1909.	1910 (a).	1911.
y iron	98,170	101,430	100,020	76,190	89,960	91,830	51,070
ron	206,309	226,900	226,430	127,630	156,590	161,210	90,950
ig	1,006,641	1,103,130	1,101,490	1,002,620	1,386,800	1,550,460	1,951,100
l	1,311,120	1,431,460	1,427,940	1,206,440	1,632,350	1,803,500	2,103,120

From *Eng. and Min. Journ.*

FOREIGN TRADE OF BELGIUM.
(In metric tons.)

	Imports.				Exports.			
	1908.	1909.	1910.	1911.	1908.	1909.	1910.	1911.
n	481,124	477,311	687,001	693,235	119,095	19,362	14,695	11,610
ht iron and	270,273	360,255	360,422	358,411	941,374	1,177,693	1,406,190	1,498,104
l	751,397	837,566	1,047,423	1,051,646	1,060,469	1,197,055	1,420,885	1,509,714

PIG IRON PRODUCTION IN CANADA.
(In tons of 2240 lb.)

	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
ry and forge	.....	146,698	130,120	84,979	114,951	149,580	153,626	224,768
ner	.....	149,203	165,609	154,910	112,811	169,545	221,494	186,274
	.....	172,102	246,228	341,257	335,410	357,965	365,090	413,303
l	270,942	468,003	541,957	581,146	563,172	677,090	740,210	824,345

STEEL PRODUCTION IN CANADA. (a)

	1910.				1911.			
	Converter.	Open-hearth.	Special.	Total.	Converter.	Open-hearth.	Special.	T
Ingots	223,266	580,334	.....	803,100	209,817	651,676	740	8
Castings	.....	18,681	.....	18,681	.....	13,982	.....	
Total	223,266	599,015	.....	822,281	209,817	665,658	740	8

(a) Reported by Statistical Section, Mines Department, long tons.

PIG IRON PRODUCTION IN FRANCE.
(In metric tons.)

	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1
Foundry	.....	635,672	591,275	651,700	695,527	749,247	760,622	...
Forge	.....	705,691	741,571	673,885	543,067	538,053	556,767	...
Bessemer	.....	160,411	149,971	122,046	118,121	118,002	104,966	...
Basic	.....	1,530,671	1,784,726	1,988,343	1,949,107	2,172,718	2,549,908	...
Special irons	.....	44,267	51,489	152,975	85,328	54,085	60,196	...
Total	2,999,787	3,076,712	3,319,032	3,588,949	3,391,150	3,632,105	4,032,459	4,4

IRON AND STEEL PRODUCTION IN FRANCE.
(In metric tons.)

	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1
Wrought iron	554,632	669,841	747,900	687,249	563,745	519,200	501,100	
Steel ingots	2,080,554	2,210,284	2,436,322	2,766,773	2,727,617	3,034,571	3,506,497	3,(
Finished steel	1,482,708	1,442,071	1,454,456	2,261,217	1,894,022	2,043,022	2,684,000	2,(

PRODUCTION OF PIG IRON IN GERMANY.
(In metric tons.)

	1905.	1906.	1907.	1908.	1909.	1910.	1
Foundry iron	1,905,668	2,108,684	2,259,416	2,254,644	2,491,919	2,968,810	2,(
Forge iron	827,498	854,536	786,113	635,228	652,306	644,992	
Steel pig	714,335	943,573	1,034,650	934,940	1,099,779	1,327,196	1,(
Bessemer pig	425,237	482,740	471,355	361,472	412,118	471,366	
Thomas pig	7,114,885	8,088,534	8,494,226	7,627,227	8,261,538	9,338,961	9,(
Total	10,987,623	12,478,067	13,045,760	11,813,511	12,917,653	14,793,325	15,(

PRODUCTION OF STEEL IN GERMANY.
(In metric tons.)

	1908.		1909.		1910.		1911.	
	Acid.	Basic.	Acid.	Basic.	Acid.	Basic.	Acid.	B
Converter ingots	374,100	6,510,754	151,148	7,517,451	171,108	8,030,571	187,359	8,(
Open-hearth ingots	224,211	3,969,595	311,812	3,967,581	252,148	5,125,421	383,895	5,(
Special steels	107,719	.....	101,842	.....	119,390	.....	139,414	....
Total (a)	706,030	10,480,349	564,802	11,485,032	542,646	13,155,992	710,668	14,(

(a) Includes direct castings.

GROWTH OF CONSUMPTION OF IRON IN GERMANY.
(In metric tons.)

	1900.	1907.	1908.	1909.	1910.	1911.
Pig iron production	8,520,541	13,045,760	11,813,511	12,917,653	14,793,325	15,652,004
Imported as pig	827,095	607,729	399,661	318,938	383,039	412,501
Imported as steel, etc.(a)	338,980	459,060	344,583	479,513	376,884	434,672
Total supply	9,886,616	14,112,549	12,557,755	13,716,114	15,553,248	16,499,177
Exported as pig	190,505	385,766	421,611	644,935	933,456	1,003,607
Exported as steel, etc. (a)	2,118,772	4,706,587	4,930,399	5,032,653	5,828,243	6,508,440
Total exports	2,309,277	5,092,353	5,352,010	5,677,588	6,761,699	7,512,047
Consumption	7,377,339	9,020,196	7,205,745	8,038,526	8,791,549	8,987,130
Consumption per head, kg.	131.1	145.1	114.4	125.8	136.0	136.9

(a) Reduced to terms of pig iron at the rate of 1 ton steel = 1.33 tons pig iron.

PRODUCTION OF IRON AND STEEL IN RUSSIA.
(In metric tons.)

	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Iron ore	4,218,600	5,272,300	4,050,000	4,580,000	4,400,000	4,450,000	4,750,000	4,595,000	5,101,000
Pig iron	2,486,610	2,978,325	2,125,000	2,350,000	2,768,220	2,748,000	2,871,332	2,740,000	2,865,000
Steel ingots	2,410,938	2,811,948	1,650,000	1,763,000	2,076,000	2,341,000	2,477,000	2,350,000	2,519,000

PRODUCTION OF IRON AND STEEL IN SPAIN.
(In metric tons.)

	1903.	1904.	1905.	1906.(a)	1907.(a)	1908.(a)	1909.(a)	1910.	1911.
Pig iron	380,284	386,000	383,100	387,500	385,000	375,000	389,000	367,000	353,500
Wrought iron	53,288	53,177	52,250	57,100	53,200	51,000	49,500	43,250	40,170
Bessemer steel	105,263	93,100	113,664	116,200	115,500	111,500	113,250	101,250	96,720
Open-hearth steel	94,379	100,659	124,200	135,400	131,600	128,000	113,750	118,750	131,510
Total steel	199,642	193,759	237,864	251,600	247,100	239,500	227,000	219,500	225,230

(a) Estimated.

SWEDISH PRODUCTION AND EXPORTS.
(In metric tons.)

	1909.			1910.			1911.		
	Pig Iron.	Wrought Iron.	Steel.	Pig. Iron.	Wrought Iron.	Steel.	Pig Iron.	Wrought Iron.	Steel.
Production	443,000	116,900	(a)310,600	604,300	150,500	(b)468,600	633,800	146,700	(c)458,200
Exports	105,700	21,000	189,500	143,500	106,700	171,100	158,400	90,600	169,300

(a) Includes 63,400 tons bessemer and 247,200 openhearth. (b) Includes 97,900 tons bessemer and 370,700 tons openhearth. (c) Includes 93,800 tons bessemer and 364,400 tons openhearth steel.

UNITED KINGDOM, PIG IRON AND WROUGHT IRON.
(In tons of 2240 lb.)

	1909.	1910.	1911.
Pig iron	9,664,394	10,216,745	9,718,638
Wrought or puddled iron	1,129,412	1,118,893	1,191,499

PRODUCTION OF STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1902.	1903.	1904.	1905.	1906.	1907	1908.
Open-hearth	3,083,288	3,124,083	3,245,346	3,879,748	4,554,936	4,663,489	3,817,103
Bessemer	1,825,779	910,018	1,781,533	2,009,712	1,907,338	1,859,259	1,478,539
Total	4,909,067	4,034,101	5,026,879	5,889,460	6,462,274	6,522,748	5,295,642

ACID AND BASIC STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1909.			1910.			1911.		
	Acid.	Basic.	Total.	Acid.	Basic.	Total.	Acid.	Basic.	Total.
Open-hearth	2,763,158	1,385,250	4,148,408	3,016,830	1,578,536	4,595,366	3,131,218	1,869,254	5,000,472
Converter	1,111,042	622,178	1,733,220	1,138,103	641,012	1,779,115	887,767	573,373	1,461,140
Total	3,874,200	2,007,428	5,881,628	4,154,933	2,219,548	6,374,481	4,018,985	2,442,627	6,461,612

EXPORTS AND IMPORTS OF UNITED KINGDOM.
(Values.)

	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Exports	£60,524,755	£75,256,655	£88,448,689	£78,914,315	£77,589,552	£91,544,552	£90,612,551
Imports	13,128,270	13,486,724	12,527,157	12,236,417	16,128,959	19,917,125	23,622,122

Note.—Exports and imports include values of machinery hardware and new ships.

EXPORTS, UNITED KINGDOM.
(In tons of 2240 lb.)

	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Pig iron	981,891	1,662,820	1,947,925	1,295,767	1,136,369	1,205,863	1,204,319
Wrought iron	183,406	200,182	211,771	172,072	170,189	141,747	138,089
Sheets	407,021	442,414	469,329	390,281	494,826	597,117	617,557
Plates	204,503	275,045	300,590	207,278	167,797	250,867	267,691
Rails	546,644	460,328	433,638	435,739	571,524	485,693	370,543
Steel shapes, etc.	151,809	226,230	338,716	275,022	305,530	369,020	347,828
Tin plates	354,951	374,802	405,329	403,007	439,804	483,020	484,355
All other kinds	891,290	1,040,379	1,059,068	921,810	932,024	1,208,578	1,088,727

PIG IRON PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1902	1,335,000	1,102,910	325,076	2,427,427	8,402,660	24,500	2,597,435
1903	1,355,000	1,299,211	269,665	2,827,668	10,085,634	28,250	2,486,610
1904	1,369,500	1,307,399	274,777	2,999,787	10,103,941	27,600	2,978,325
1905	1,372,300	1,310,290	475,491	3,077,000	10,987,623	31,300	2,125,000
1906	1,403,500	1,431,160	550,618	3,319,032	12,478,067	30,450	2,350,000
1907	1,405,000	1,427,940	590,444	3,588,949	13,045,760	32,000	2,768,220
1908	1,650,000	1,206,440	572,123	3,391,150	11,813,511	112,924	2,748,000
1909	1,958,786	1,632,350	687,923	3,632,105	12,917,653	207,800	2,871,332
1910	2,010,000	1,803,500	752,090	4,032,459	14,793,325	215,000	2,740,000
1911	2,095,000	2,103,120	837,575	4,410,856	15,280,527	235,000	2,865,000

PIG IRON PRODUCTION OF THE WORLD.—(*Continued.*)
(In metric tons.)

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1902	330,747	524,400	8,653,976	18,003,448	615,000	44,342,579
1903	380,284	506,825	8,952,183	18,297,400	625,000	47,113,730
1904	386,000	528,525	8,699,661	16,760,986	633,000	46,069,501
1905	383,100	531,200	9,746,221	23,340,258	655,000	54,054,783
1906	387,500	552,250	10,311,778	25,706,882	650,000	59,074,861
1907	385,000	603,100	10,082,638	26,193,863	556,900	60,680,014
1908	403,500	563,300	9,438,477	16,190,994	550,000	48,640,479
1909	389,000	443,000	9,818,916	26,108,199	550,000	61,217,064
1910	367,000	604,300	10,380,723	27,636,687	525,000	65,860,084
1911	353,500	633,800	9,874,620	24,027,733	535,000	63,251,731

STEEL PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1901	1,142,500	526,670	26,501	1,425,351	6,394,222	121,300	2,230,000
1902	1,143,900	776,875	184,950	1,635,300	7,780,682	119,500	2,183,400
1903	1,146,000	981,740	181,514	1,854,620	8,801,515	116,000	2,410,938
1904	1,195,000	1,069,880	151,165	2,080,554	8,930,291	113,800	2,811,948
1905	1,188,000	1,023,500	403,449	2,210,284	10,066,553	117,300	1,650,000
1906	1,195,000	1,185,660	515,200	2,371,377	11,135,085	109,000	1,763,000
1907	1,195,500	1,183,500	516,300	2,677,805	12,063,632	115,000	2,076,000
1908	2,025,182	1,065,500	598,183	2,727,717	10,480,349	537,000	2,341,000
1909	1,969,538	1,370,000	766,795	3,034,571	12,049,834	661,600	2,471,000
1910	2,154,832	1,449,500	835,478	3,506,497	13,698,638	635,000	2,350,000
1911	2,475,437	1,537,000	880,278	3,668,678	15,019,333	646,500	2,519,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1901	122,954	269,897	5,096,301	13,689,173	405,000	31,440,869
1902	163,564	283,500	5,102,420	15,186,406	412,000	34,972,497
1903	199,642	317,107	5,114,647	14,756,691	418,000	36,298,414
1904	193,759	333,522	5,107,309	13,746,051	415,000	36,148,079
1905	237,864	340,000	5,983,691	20,354,291	426,000	43,900,648
1906	251,600	351,900	6,565,670	23,772,506	420,000	49,635,998
1907	247,100	443,000	6,627,112	23,733,391	405,000	51,273,340
1908	239,500	427,100	5,380,372	14,247,619	300,000	44,359,522
1909	227,000	310,600	5,975,734	24,338,302	325,000	53,499,974
1910	220,000	468,600	6,476,791	26,512,437	315,000	58,622,773
1911	228,230	458,200	6,565,321	24,054,309	325,000	58,377,286

Recent Changes in the Metallurgy of Iron and Steel

By Bradley Stoughton

Some notable events of 1911 in the American iron and steel industry are: the cancellation of the Hill ore lease by the U. S. Steel Corporation; an increasing number of discoveries of new ore bodies in the Lake Superior region; important developments of Central and South American ore deposits and the probable beginnings of an iron and steel industry in South America; increasing treatment of American ores, such as crushing, washing, calcining, roasting and sintering; improvements in blast drying

processes; removable openhearth furnace ports; an increased production of alloy steels and greater attention paid by American iron and steel masters to the improvement of the quality of their product.

Treatment of Lake Superior Iron Ores.—Extensive washing of Mesabi iron ores was carried on during 1911, and the crushing of the large ores has been practised on a more extended scale, while more recent innovations are drying, calcining and roasting. The Gröndal concentration process is in use at Moose Mountain, in Canada; and the Jones step process is being operated in an experimental scale at Koman Mine. The Gröndal briquetting process is practised at Mayville, Wisconsin.

Agglomeration of Iron Ore and Flue Dust.—The increasing number of known iron ore bodies in America which would be made available for smelting by agglomerating, either with or without desulphurizing, crushing or concentrating, has awakened much interest in the various agglomerating processes. A nodulizing process has been in successful operation for some years at different localities, but its cost and the somewhat compact and glazed character of the product, which resists ready attack in the blast-furnace, has militated against its universal adoption where applicable. Briquetting and sintering processes have since been tried experimentally, and several installations were made in 1911; in February, 1912, a symposium on this subject was held by the American Institute of Mining Engineers, and much valuable data presented and published, from which much of what follows is abstracted. The ideal agglomerated iron-bearing product for smelting would have the following characteristics:

1. Be porous and permeable by gases at low pressure;
2. Yield easily to reduction by carbon monoxide;
3. Cheap to make;
4. Retain an unglazed surface of exposure;
5. Have the strength to resist at least 2000 lb. per square inch crushing force;
6. When it does break, break in lumps with no dust;
7. Able to fall 20 to 30 ft. on to a hard substance without breaking, and especially without forming dust;
8. Resist exposure to dampness and steam without softening or crumbling;
9. Be uniform in size and in surface exposed;
10. Resist a strong red heat without decrepitating.

Each of the processes so far adopted in America have all of these characteristics in greater or less degree, but some excell in one or another of the different qualities desired. The processes naturally divide themselves into two classes:

A. Briquetting processes.
  a. Using binders.
  b. Not using binders.
B. Blast-roasting processes.

The briquetting processes originate chiefly in Germany, where many of them are in successful operation, while the blast-roasting process—Dwight-Lloyd process—is an American invention. The product of briquetting processes, even the porous, is not as permeable to gases as the more spongy product of the sintering processes, with the exception of the material from the Gröndal process, which is first briquetted and then sintered. In short, the briquetted product is compact in form, as distinguished from the open structure of the sintered material. Where binders are used, the porosity is reduced by the filling up of interstices by the cement which holds the particles together. Consequently, none of the briquetting processes using 5 to 10 per cent. of binder have found extensive employment; on the other hand, it is urged against the sintered product that its surface is covered with at least an incipient glaze which resists the attack of furnace gas. The process which uses no binder whatever—Ronay process—employs very heavy briquetting pressure—5000 to 15,000 lb. per square inch—which reduce the porosity of the product, and increases the cost of installation. This process has found favor, however, for briquetting metallic chips for melting in the cupola. The three processes which have so far met with the most promising success in America are: the Schumacher process, the Gröndal process, and the Dwight-Lloyd process. Each of these produces a porous material, which yields easily to reduction and has the requisite strength and ability to resist dampness, steam, strong red heat, and to form very little dust. None have been tried a sufficient length of time under comparative conditions to prove which excells in these respects, but the Schumacher product is a compact brick, while the Gröndal and Dwight-Lloyd products have a spongy, open structure. It is claimed, however, that the surface of the two latter products has an incipient glaze. In answer to this argument, B. G. Klugh says:[1] "The typical structure of the Dwight & Lloyd sinter is similar to that of a very porous sponge. The cell walls of the larger walls in the sinter are very thin, but are nevertheless highly porous, due to the presence of many microscopic pores. . . . . It is thus evident that even if Mr. Vogel's statement that the surfaces of the sinter are glazed, be true, this condition would not affect the reducibility of the sinter, since the glazed surface would be perforated at innumerable points by microscopic pores." It would appear that the Schumacher process is cheaper to operate than the Dwight-Lloyd process, and both are cheaper than

[1] *Bulletin*, Am. Inst. Min. Engs., May, 1912, p. 543.

the Gröndal process. In respect to this latter point, N. V. Hansell says: [1] "In calculating the technical economy by which any process must stand or fall, it must be recognized that this cannot be determined by the briquetting cost alone. . . . . For the preparation of a briquette which is easily reducible with a low consumption of coke, and the use of which increases the output of the blast-furnace, one can afford to pay more. . . . . It is the rapid reduction of the (Gröndal) briquettes in the blast-furnace that has made them so popular in Sweden."

The operation of the processes is described as follows:

The Schumacher process employs flue dust as the cement or binder, Dr. Schumacher having discovered that the addition of a very small amount of certain agents—0.25 to 2.00 per cent. of magnesium chloride, or of calcium chloride, for instance—will release a latent binding power in this material so that it will set into a hard mass in 15 to 60 minutes. Some samples of flue dust have sufficient binding power to cement as much as three times their weight of fine ore, and thus a valuable, but useless, by-product of the furnace may be employed to improve the condition of the ore for the furnace. The flue dust produced in America has been estimated at 2,500,000 to 3,500,000 tons per annum. At the Cambria Steel Plant, the warm flue dust and ore are mechanically mixed together, and with a 30 per cent. solution of calcium chloride. The product is then fed into one of two hydraulic presses and subjected to a pressure of either 5500 or 12,500 lb. per square inch. The chemical reaction produced under the influence of the pressure and the so-called "catalytic" action of the small amount of calcium chloride, causes the bricks to heat up so that they come from the press hot and steaming. They are set aside and are then ready for charging into the furnace. The value of the coke normally contained in the flue dust used is said to equal the total cost of the briquetting operation.

In the Dwight-Lloyd process "the finely divided iron-bearing material to be sintered is intimately mixed with the amount of carbon"—about 8 per cent.—"requisite to produce the sinter. This mixture is then moistened and deposited upon the machine in a uniform layer, the upper surface of which is ignited by a small flame of intense local heat, and the combustion of the intermixed carbon is effected in a progressively downward course by a current of air flowing in the same direction through the permeable mass. The energy from the combustion of each particle of carbon is expended and utilized directly upon the adjacent particles of iron-bearing material, agglomerating them into a coherent sinter."

For the Gröndal process the ore must contain enough water to main-

[1] *Bulletin*, Am. Inst. Min. Engs., May, 1912, p. 561.

tain its brick form after pressing; it is subjected to a moderate pressure and then piled on cars and pushed through a long furnace in which it is subjected to a maximum temperature of 1400° C. In the Gröndal process, as in the Dwight-Lloyd process, the ore is desulphurized to practically any desired extent.

Thin-lined Blast-furnace.—David Baker describes the different types of thin-lined blast-furnace construction, and suggests that the thin-lined furnace has not been so successful in America as in Germany, chiefly because of the greater loss of heat, due to the American type of construction.[1] In the German thin-lined furnace, the shells are constructed of heavy cast-iron segments, whereas, in the American type, steel plates of 1 in. to 1 1/8 in. thickness are provided, which are cooled on the outside by water. In one type of cooling, the water circulates from pocket to pocket rivetted on the outside of the shell; in a second type, water is supplied to the different pockets at a rate sufficient to neutralize what is lost by evaporation, and in the third type the outside is sprayed by water, so that a uniform film flows down the sides.

New Uses for Blast-furnace Gas.—Probably there is no subdepartment of iron and steel manufacture in which improvements effecting economies in operation have been more notable and far reaching than in the improved use of blast-furnace gas. Several of these improvements have originated as an indirect result of better cleaning of the gas for its use in gas engines, which has also facilitated and improved the care and operation of the hot-blast stoves, conducting pipes, etc. Where by-product coke ovens are operated in conjunction with a blast-furnace plant, the blast-furnace gas may be used for burning in the coke-oven flues, thus making available a larger proportion of the richer coke-oven gas for more valuable purposes. For heating openhearth furnaces, blast-furnace gas has not sufficient calorific power when used alone; we may estimate it at about 85 to 95 b. t. u. per cubic foot, as compared with 130 to 145 b. t. u. for good coke-oven gas, and 970 to 1010 b. t. u. for natural gas. Another deficiency of blast-furnace gas for radiation heating (openhearth, annealing, reverberatory, and similar furnaces) is its lack in luminosity. But by mixing some coke-oven gas with blast-furnace gas, it may be brought to the desired degree of calorific power and luminosity, and thus employed economically. It is stated by Dr. Rudolph Buck, who cites many experiments made in Germany in this connection, that a mixture having 170 to 213 b. t. u. per cubic foot gives satisfactory results.

Improvements in Blast-drying Apparatus.—The cost of installation

[1] David Baker in *The Iron Age*, December 28, 1911.

of processes for the desiccation of blast-furnace air has been reduced at a Welch blast-furnace by dispensing with the use of brine and refrigerating with liquid ammonia direct. The strong liquor from the condensor apparatus is evaporated by the latent heat in the waste steam and thus the ammonia is used over and over again. At a Luxemburg blast-furnace, refrigeration is dispensed with altogether, and the air is dried by means of calcium chloride, which is said to reduce the cost of installation 70 per cent. The calcium chloride is charged with moisture after about 6 to 8 hours' use, and it is then regenerated by heating by means of the flue gases from the boilers and the hot-blast stoves. A triplicate installation of calcium chloride apparatus affords one unit always available for drying purposes.

In the *Iron Age* of Jan. 4, 1912, James Gayley describes a two-stage system of refrigeration which is more economical in construction and operation than the original method of passing the air over coils of pipe within which brine at a low pressure circulates.

Turbine Blowing Engines.—J. E. Johnson suggested the use of turbine blowers, employing waste steam from the steam-blowing engines of blast-furnaces, and supplying the steam engines with air at low pressure, which is by them raised to high pressure necessary for blowing, thus bettering the conditions of operation of both types of apparatus, increasing their efficiency and reducing their cost. Although this combination was not tried at a furnace, the turbine was introduced in simple form at at least three American blast-furnaces, with the result of more uniform blowing conditions and decreased cost of blowing. This innovation is doubtless of far-reaching importance in mechanical equipment of blast-furnace plants, although the significance of it seems to be slow in obtaining recognition.

The Growth of the Slag-cement Industry.—The manufacture of cement from iron blast-furnace slag has grown from 32,000 barrels in 1900 to 7,000,000 barrels (equivalent to over 1,000,000 gross tons) in 1910. Thus a waste product now supplies nearly 10 per cent. of the production of an important industry. It might be thought that the possibility of making an increase in the lime content of a slag pay for itself in the cement industry, and the frequently reported depreciation of the grade of the American iron ores, would result in a large increase in the production of cinder, and more especially an increase in the use of flux. On the contrary, however, there has been but little change in the amount of limestone used per ton of iron made in the last 10 years, which averages about 1200 lb.

Developments in Iron Foundry Practice.—A new method of treating borings, turnings and drillings of cast iron, wrought iron and steel, which

form an important by-product of every machine shop and are melted only with difficulty and large waste in a cupola or air furnace, is to briquet them into solid masses which do not disintegrate during passage through the furnace. Several installations of the Ronay process of briquetting have been made. In this process the chips are agglomerated by means of pressure alone. A pressure of 35 lb. per square inch is used and a few moments must be allowed for it to exert its effect. The cost is said to be about 50 cents per ton, and, as metallic chips can be bought for $6 or so per ton, and the briquettes are nearly equal to pig iron or heavy steel scrap costing $10 to $14 per ton, the process would seem to have a bright future.

Other foundry innovations are the melting and casting of iron, steel or other alloys *in vacuo*, for the purpose of getting rid of occluded gases and blow holes, the chilling of cast iron by means of an air blast instead of by means of a metal chill incorporated in the mold, the use of permanent molds for brake-shoes, and the use of the Hooper continuous pouring spout for distributing metal to as many different ladles as desired from air furnaces and openhearth furnaces without closing the tap hole, thus saving time, heat, oxidation of the metal, drip of metal, and the use of large ladles. Professor B. Osann discussed in *Stahl und Eisen*, Vol. XXXI, p. 1219, that little understood subject of the foundry drying oven, and the principles governing the operation are set forth in a clearer manner than has previously been accomplished, as far as we know.

Decreasing Wrought-iron Production and Quality.—The oft-quoted statement that the invention of the bessemer and openhearth processes would drive the wrought-iron industry out of existence has long since ceased to worry the iron manufacturer; nevertheless, the sphere of rolled iron has constantly narrowed, and the actual production in America has been on the decrease since about 20 years ago. The following statistics of the number of puddling furnaces are compiled from data published by the American Iron and Steel Association:

Nov., 1887	4882	April, 1898	3889
Nov., 1889	4914	Nov., 1901	3251
Jan., 1892	5120	June, 1904	3161
Jan., 1894	4715	Nov., 1907	2635
Jan., 1896	4408		

The following figures of production of wrought iron are from the same sources.

It is to be observed that a large proportion—probably from one-half to three-quarters—of this tonnage is made by the busheling and re-rolling of wrought-iron scrap, and is not the new product of the puddling furnace. This circumstance is especially important because it means the abundant use of an inferior product through a misconception. Busheled

PRODUCTION OF ROLLED IRON.

	Rolled Iron, Gross Tons.	Percentage of Total Rolled Iron and Steel.
1888	2,153,263	46.5
1889	2,309,272	44.1
1890	2,518,194	42.8
1904	1,760,084	14.7
1905	2,059,990	12.2
1906	2,186,557	11.2
1907	2,200,086	11.1
1908	1,238,449	10.5
1909	1,709,431	8.7
1910	1,740,156	8.1

scrap almost inevitably contains masses of intermixed steel scrap, which renders the product less able to resist rusting than either wrought iron or steel alone; and yet it is the belief in the the superiority of iron to resist rusting that is the chief factor in keeping alive its manufacture to day, as evidenced by the circumstance that the field of wrought iron has narrowed more and more to the production of forms that are to be subjected to corrosive influences; namely, roofing sheets, boiler plates and pipes. Whatever one may think of the quality of newly puddled iron in this respect, we believe the unreliability of busheled scrap will not be questioned.

Openhearth Furnace Changes.—The tendency to increase the size of openhearth furnaces continues, and recent installations have been in units of 100 tons capacity, instead of the previous 50 and 75 ton sizes. There is also a further increase in the use of water-cooled ports. A recent German innovation deals with changeable openhearth furnace ports somewhat similar in its effect upon the openhearth operation to Alexander L. Holley's improvement of the bessemer converter, by supplying interchangeable bottoms.

The ports of openhearth furnaces must be maintained fairly constant in size and form, in order that they may direct the gas and air efficiently and uniformly throughout the operation. The brick work at these points, however, suffers more damage from heat than almost any other part of the furnace, and is consequently comparatively short lived, because of the intense temperature to which it is subjected by the outgoing gases. Water-cooling and other devices have been installed in order to increase the endurance of this part of the furnace with noteworthy success. This new invention permits of the removal and replacement of the ports of the furnace without closing down for more than 5 hours, which can easily be accomplished during the customary week-end shutdown. This makes the life of the openhearth furnace practically independent of the life of the ports. In this way basic openhearth furnaces whose life might normally be 350 to 500 heats, have been made to last for 1000 heats with three changes of ports.

Another German improvement is in openhearth furnace ports.[1]

The advantage of this innovation has already been proven in practice, and its efficiency is due to the circumstance that, as the ports burn back, no change is made in the location with respect to the charge in which the gas and air meet and combustion takes place, so that the working of the furnace is regular throughout its entire life.

Still another German improvement is the melting of ferromanganese for recarburizing in an electric furnace and adding it molten to the steel ladle. A gain is made in the uniformity of analysis and of deoxidation, and a saving of about 35 per cent. of the manganese, which alone is said to be more than double in value the cost of melting.

Electric Smelting of Iron Ores.—That electric smelting of iron ore under special conditions of high priced coke and cheap power is an economical and commercial possibility was still further indicated in 1911 by the installation of a new plant in Sweden, and by continued success of the former plants in Sweden and California. At the September Meeting of the American Electrochemical Society, some further details were given in regard to this process and certain innovations were described, notably improvements in electrodes, whereby one is able to use a greater part of the stump. One of these methods consists of an iron sheath forming a frame around the electrode, and another is the use of screw joints whereby the stump of an old electrode may be used for the end of a new one. This improvement is said to reduce the consumption of electrodes from 10.28 kilos per ton of pig iron produced, to 5.5 kilos. At the new Trolhattan furnace, which has been in continuous operation since Aug. 4, 1911, having been first started in Nov., 1910, the consumption of electricity is said to be in the proportion of 5.05 tons of pig iron produced per kw. year. It is said that five additional furnaces are in the course of erection in Sweden and Norway, as a result of the experience with the Domnarfvet and Trolhattan furnaces. An innovation in the new type of operation is an increase in the pressure of the tunnel head gases which, it will be remembered, are taken off from near the top of the furnace and blown against the vault of the smelting crucible to cool it; another novelty is charging the ore near the middle of the shaft. By this means the efficiency of carbon reduction has increased and the carbon dioxide in the escaping gases is brought to over 30 per cent.

Combination Electro-openhearth and Electro-bessemer Furnaces.—The chief function of electric refining processes from the industrial standpoint is doubtless the super-refining of steel which has already been purified as far as is commercially practical in furnaces using cheaper sources of heat—namely, openhearth or bessemer furnaces, and perhaps even

[1] *Stahl und Eisen*, Vol. XXXI, No. 28.

puddling and Bell-Krupp furnaces, as the electric processes develop. Substantially all of the 50,000 tons of electric steel produced in America in 1910 was made by purification in bessemer or openhearth furnaces, followed by further refining in electric furnaces, this super-refining consisting virtually in the removal of the last traces of phosphorus, and of sulphur, gases and entangled solid particles of oxidized metals. Realizing this, the Homestead steel works has been operating for some time an openhearth furnace to which have been added two graphite electrodes, which can be brought into play for heating purposes in a manner similar to the action of the ordinary Heroult steel furnace, after the customary openhearth process of purification is completed, thus avoiding the transfer of metal from one furnace to another. An unconfirmed consular report from England states also that a firm in Sheffield is installing a double-ended bessemer converter, in one part of which the metal can be purified by the ordinary bessemer process, and then transferred to the opposite end by merely tilting the converter. In this further end, are situated electrodes by means of which electric current is brought to heat the charge by arc and resistance, thus maintaining the metal at the desired temperature while the super-refining is carried on. The general report for the year 1911 is that electric refining in the three chief siderurgical countries—America, Germany and England—is progressing at a rate as rapid as could be reasonably expected in view of the large cost for installation and operation, and the financial conditions in the industry at present. One German plant has even gone so far as to produce regularly ingots by purification complete from pig iron to finished steel in the electric furnace.

In America the great bulk of steel produced by electricity is made with the greatest possible saving in electric power; namely, by carrying on a preliminary purification, either in a bessemer or openhearth furnace, and then giving the metal a super-refining in the electric furnace. In Europe about one-half the furnaces in operation are used for complete refining of a charge of pig iron, or pig iron and steel scrap, and the other half are used for super-refining subsequent to a process not using electric heat.

A New Induction Furnace Lining.—As a result of experiments carried on in Kladnov, Austria, patents have been granted to the Gröndal-Kjellin Co., Ltd., for a combination basic furnace lining, which is said to increase the life of the lining from an average of 49 charges when using a Veit magnesite, to about 500 charges.[1]

Electric Furnace for Heating, Rolling and Forging.—One of the great factors contributing to the advantage of using an electric furnace is its

[1] *Iron Age*, Nov. 30, 1911, p. 1171.

ability to heat in a non-oxidizing atmosphere. Further advantage has now been taken of this characteristic by an American forging plant which has installed an electrically heated furnace for heating previous to forging. Since the loss of metal by oxidation in this operation is often as great as 5 per cent. of the weight of the piece, and greater than that when the pieces are small and have to be heated many times, so that occasionally so much metal may be lost that the piece will not fill the die where drop forgings are being made, the advantage of reducing oxidation to a minimum can be readily appreciated. The accompanying advantage of such a furnace is that it may be occasionally used for annealing, in which its unexcelled control of the temperature would tend to improve the quality of the product.

Welding of Blow-holes in Ingots.—In *The Journal of the Iron and Steel Institute*, No. 1, 1911, and No. 1, 1912, J. E. Stead contributes the results of experiments carried on with the welding of blow-holes in ingots, from which he draws the important conclusion that, under the ordinary heating and rolling treatment, to which honeycombed ingots of steel are subjected, internal small cavities or blow-holes do become perfectly welded up provided there is an absence of sulphide segregations, and that, even when these segregations are present, as they are not in continuous lines, but occur only at intervals, the clean metallic faces between them come into contact and weld together.

It seems also justifiable to conclude that surface blow-holes, which become oxidized on their walls during the heating and rolling of the ingot, do become more or less completely welded. The conditions favorable to this welding are: A sufficiently high temperature, and maintenance of the steel at that temperature for a long enough period after the cavities have been closed, to admit of the carbon in the adjacent steel being afforded the opportunity to reduce the oxide scale.

Power Requirements of Rolling Mills.—Samuel S. Roberts, Testing Engineer, Carnegie Steel Company, has devised an apparatus for ascertaining the relation between heavy imposed loads and limited speed variations, together with economical operation of the prime mover of rolling mills. With such an instrument it is possible to make a graphical analysis of the power measurements with corresponding speed determinations of the rolling mill, with the services of but one man.

Improvement of the Quality of Steel

Case-hardening Steel

Solubility of Carbon in Iron.—Ruff and Goecke publish in *Metallurgie* for July 22, 1911, an account of some discoveries they have made in the

solubility of carbon in iron. They heated carbon in contact with iron in an electric furnace, to temperatures ranging from 1100 to 2500° C. (2012 to 4532° F.) and determined the amount of carbon absorbed. On plotting their results in graphic form, there is seen to be a rise in solubility with increase in temperature up to 6.7 per cent. of carbon, which corresponds to the chemical compound Fe_3C, and which was obtained at 1837° C. After this point there is a temporary decrease in the rate of absorption with increasing temperature, but carbon continues to be dissolved until, at 2220° C., there is 9.7 per cent. absorbed, corresponding to the formula Fe_2C. This is the maximum amount dissolved; with further increase in temperature less carbon was absorbed.

Dr. Giollitti publishes in *The Journal of the Iron and Steel Institute*, No. 2, 1911, a description of a new process for the case-hardening of steel by means of compressed gases, whereby the speed of the operation and the uniformity of the treatment is said to be materially increased, while the difficulties frequently met with in carburizing alloy steels are said to be materially reduced.

Production of Alloy Steels.—One reason for the increase in the use of alloy steels is the increased production of automobiles, shown by the following estimate by an authority on the subject:

	Cars Built.	Value.
1912	210,000	$321,930,000
1911	140,000	175,000,000
1910	185,000	242,000,000
1909	82,000	98,400,000
1908	55,400	83,100,000
1904	20,100	40,200,000

A feature of the automobile situation in America is the increasing demand for motor cars for exportation which is estimated for the 10 months ending in October, 1911 to have been over $15,000,000.

The following statistics of the production of alloy steel ingots and castings, presented by the American Iron & Steel Association, show not only great improvements of this high-grade industry at the present time, but also the large increase during 1910, which bids fair to be again exceeded when the 1911 figures are published.

PRODUCTION OF ALLOY STEELS.

Alloy.	Tons.	Per cent.
Titanium	326,316	57.5
Nickel	106,707	18.8
Nickel-chrome	52,021	9.1
Chrome	23,550	4.2
Manganese	19,360	3.4
Vanadium	9,049	1.6
Chrome-vanadium	6,903	1.2
Nickel-chrome-vanadium	8,760	1.5
Miscellaneous	15,153	2.7
Total	567,819	100.0

Included in miscellaneous are 3163 tons of tungsten steel. Of the total reported, 296,298 tons were made by the bessemer process; 296,135 were basic openhearth; 44, 394 tons acid openhearth; 20,392 tons crucible, electric, etc. The total includes 29,357 tons of direct castings, chiefly of manganese nickel and chrome alloys. As compared with 1909 there was an increase of 385,839 tons, chiefly in titanium steel.

It is doubtful whether the term "alloy steels" is really applicable to the steels which have been subjected to a super-refining by vanadium or titanium. Nevertheless, the vanadium and titanium steels are included in these special classes for convenience. In this connection it is worthy of note that nearly 196,000 tons of rails, out of a total production of 6,334,000 were treated with titanium in the United States, and the indications are that double this amount will be so treated during 1911. An illuminating discussion was carried forward in *Iron Age* as to the advantages of using titanium for super-refining with or without aluminum.

The use of titanium and vanadium has increased materially during 1911, and the price at which vanadium has been sold, which has been a great drawback to its larger employment, has been lowered from about $6 per pound of contained vanadium, to less than $3 per pound. Titanium, which had already proved its importance and efficacy in rolled steels, has now admitted advantages in chilled cast iron and is finding increased employment as a cleanser in all types of openhearth and bessemer steels.

A very important decision in the field of alloy steels is that of the United States Circuit Court of Appeals in sustaining the decision of the lower court by dismissing the suits brought by the Friedrich Krupp Actien Gesellschaft against the Midvale Steel Co. for the infringement of armor-plate patents.

LEAD

The production of lead in the United States for the year 1911, as reported by the various refiners,[1] was 400,958 tons as compared with 392,704 tons in 1910. As stated in Vol. XVIII of THE MINERAL INDUSTRY these figures give the actual production of the various smelteries which treat lead ore and also those which produce lead as a by-product. They represent the refinery production of lead in final marketable form. It appears that there was a small decrease in the output of what we class as desilverized, which was offset by the increased production of the Kansas, Missouri and Illinois refiners. Certain of the latter now desilverize a portion of their lead, but adhering to the old industrial classification, we include under desilverized only the production of Omaha, Chicago, Pittsburg, New Jersey and the Pacific Coast. There is also more or less scrap lead worked over which tends to increase the production figures a small amount, but this is not included in the figures here given, as it is handled separately by dealers in junk and scrap material.

The production of lead by the mines increased in 1911 more than the refinery statistics show, a rather large supply of lead ore having been accumulated at the Tooele smeltery, in Utah.

The refinery statistics indicate that the output of lead in Missouri experienced another large increase in 1911, while that of the Coeur d'Alene was probably about the same as in the previous year.

METALLURGICAL PRODUCTION OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Domestic Origin.						Foreign Origin.		Grand Total.
	Desilverized.	Antimonial	S. E. Mo.	S. W.Mo.	Miscel.	Total.	Desilverized.	Antimonial	
1906...	220,095	7,434	100,492	16,528	980	345,529	67,441	2,686	415,656
1907...	213,383	9,614	108,510	17,833	790	350,130	76,016	(a)	426,146
1908...	174,650	13,109	113,103	18,014	*Nil.*	318,876	94,992	(a)	413,868
1909...	211,499	10,392	126,784	20,489	*Nil.*	369,164	89,681	2,338	461,183
1910(b)	217,490	9,098	145,387	20,729	*Nil.*	392,704	90,473	4,892	488,069
1911...	211,041	8,916	155,008	25,993	*Nil.*	400,958	89,487	4,929	495,374

(a) Entered under domestic antimonial. (b) From *Eng. and Min. Journ.*

In the statistics given in the accompanying table, one smeltery in Kansas is included in "Southwest Missouri" and two smelteries in Illinois are included in "Southeast Missouri," as most of their ore comes

[1] Through *Eng. and Min. Journ.*

from these districts. The production of soft lead from the Mississippi valley was 179,911 tons in 1911 as compared with 166,116 tons in 1910. The total increase in the production of soft lead therefore was about 13,000 tons from the Mississippi valley, while there was a decrease of 4000 tons in desilverized lead. The Southeast Missouri district shows an increased production of 8528 tons and the Joplin district 5267 tons more than in 1910. Owing to more exports and also to less quantity of lead being produced from foreign material smelted in bond, the stocks in bonded warehouses were much decreased.

A development of some significance in the lead industry was the action of the International Smelting and Refining Co. in adding lead furnaces to its plant at Tooele, Utah. The company has undertaken to enter the lead smelting field in competition with the American Smelting and Refining Co. and its agents have been active in securing contracts for lead ore from the various mining districts of the West.

STATISTICS OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Produced from Domestic Ores.				Imported in Ores and Bullion. (c)	Total Production and Imports.	Exported in all Forms. (c)
	Desilverized.	Soft. (a)	Antimonial. (b)	Totals.			
1898....	169,364	50,468	8,643	228,475	89,209	348,845	78,168
1899....	171,495	40,508	7,377	217,085	76,423	317,196	74,944
1900....	221,278	47,923	9,906	279,107	114,397	425,824	100,288
1901....	211,368	57,898	10,656	279,922	112,471	458,033	100,026
1902....	199,615	70,424	10,485	280,524	107,715	458,456	82,228
1903....	188,943	78,298	9,453	276,694	106,407	418,601	81,971
1904....	200,858	90,470	10,876	302,204	112,852	415,056	84,142
1905....	205,665	105,623	11,186	322,474	98,378	420,852	59,741
1906....	220,095	118,000	10,120	348,215	84,134	432,349	47,323
1907....	213,383	127,133	9,614	350,130	79,815	429,945	51,502
1908....	174,650	131,117	13,109	318,876	112,074	430,950	76,857
1909....	211,499	147,273	12,730	371,502	114,181	485,683	86,077
1910....	217,490	166,116	13,990	397,596	108,602	506,198	69,786
1911(d).	211,041	181,001	13,845	405,887	89,951	495,838	101,227

(a) Since 1904 a large part of the so-called soft lead was desilverized, but this (being of Missouri origin) has been included in the old classification. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin. (c) Includes "pigs, bars and old." (d) From *Eng. and Min. Journ.*

IMPORTS OF LEAD IN ORE, BASE BULLION, PIGS, BARS AND OLD. (a)
(In tons of 2000 lb.)

Source.	1903.	1904.	1905.	1906.	1907.	1908. (c)	1909. (c)	1910. (c)	1911.(c)
United Kingdom....	776	247	795	4,926	217	(b)	(b)	(b)	(b)
Germany.....	705	366	125	1,003	228	(b)	(b)	(b)	(b)
Other European.....	226	83	59	1,961	3,461	(b)	(b)	(b)	(b)
Canada.....	9,600	8,952	8,182	9,257	6,663	644	2,063	60	68
Mexico.....	93,068	102,903	87,584	66,756	68,767	107,369	104,120	98,535	83,925
South America.....	1,948	290	1,577	158	442	(b)	915	3,289	2,383
Other Countries.....	83	11	56	74	63	1,273	3,507	3,233	943
Total.....	106,406	112,852	98,378	84,135	79,814	109,286	110,605	105,117	87,319

(a) Refined lead, i.e., in pigs, bars and old, is a small part of the total. It was in 1901, 604 tons; in 1902, 2,529 tons; 1903, 3,023 tons; 1904, 8,724 tons; 1905, 5,720 tons; 1906, 11,763 tons; 1907, 9,277 tons; 1908, 2,759 tons. (b) Included in other countries. (c) Figures do not include import of pigs, bars and old. With these the total imports for 1908 were 112,045 tons; 114,182 tons in 1909, 108,602 tons in 1910, and 89,952 tons in 1911.

Missouri continues to hold first place in the lead-producing columns of the United States. Colorado's small increase in lead production is offset by its zinc production, which started to increase in 1910.

PRODUCTION OF CRUDE LEAD BY STATES.
(In tons of 2000 lb.)

State.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Arizona	1,424	1,986	2,884	2,200	1,867	1,304	1,399	4,392
California	155	110	432	850	490	865	1,038	529
Colorado	49,290	(*f*)57,856	(*f*)52,992	47,332	26,707	26,413	34,711	28,730
Idaho	103,411	(*h*)107,000	(*i*)121,584	111,697	98,394	97,137	99,840	116,524
Kansas	(*n*)	(*n*)	(*n*)	1,800	2,400	1,500	1,343	2,500
Missouri	92,119	(*g*)102,500	115,103	123,613	125,216	146,829	160,642	173,787
Montana	3,454	2,097	2,485	2,005	2,309	1,331	1,795	2,492
Nevada	1,779	2,096	1,669	3,400	3,676	4,086	2,119	1,076
New Mexico	1,295	1,170	640	1,900	611	1,275	1,730	1,381
Oklahoma	.....	.....	.....	400	1,000	3,000	1,860	1,950
Utah	53,647	42,746	56,260	54,738	43,995	65,975	57,158	54,538
Wisconsin	.....	.....	1,753	3,500	3,486	2,745	3,400	3,944
Other States	(*k*)630	(*k*)2,695	(*m*)943	1,204	600	629	177	912
Undistributed	.....	.....	.....	.....	2,026	3,062	1,429	1,760
Zinc Smelters	.....	.....	.....	1,320	1,290	2,796	1,988	3,511
Total	307,204	320,256	356,745	355,959	314,067	358,947	370,629	398,026

Figures for 1904 and 1905 are by the U. S. Geological Survey giving production of "merchant lead." Figures for 1906 and subsequent years are from smelters' reports to the *Eng. and Min. Journ.* (*f*) Report of State Commissioner of Mines. (*h*) Partly estimated. (*i*) Report of State Inspector of Mines less allowance of 5 per cent. for loss in smelting. (*k*) Includees production of Alaska, Oregon, So. Dakota, Washington, Georgia, Tennessee, Virginia, Kentucky and Texas. (*m*) Includes production of Illinois, Iowa, Kentucky, Tennessee and Washington. (*n*) Included with Missouri.

Imports and Exports.—The imports of lead in 1911 as reported by the Bureau of Statistics were 17,843 short tons in ore, valued at $661,615; 69,476 short tons in base bullion valued at $2,791,080, and 2632 tons of refined lead valued at $190,543; a total of 89,952 short tons as compared with 108,607 in 1910, and 114,182 in 1909.

DELIVERY OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

	1907.	1908.	1909.	1910.	1911.
Supply:					
Production desilverized	289,399	269,642	301,180	307,963	300,528
Production soft lead	127,133	131,117	147,273	166,116	181,001
Production antimonial lead	9,614	13,109	12,730	13,990	13,560
Imports foreign refined lead	9,277	2,759	3,576	3,485	2,632
Stock, domestic lead, Jan. 1	4,000	(*e*)50,000	(*e*)75,000	(*e*)94,430	(*e*) 80,000
Foreign in bond, Jan. 1	5,691	12,897	18,462	12,695	(*e*) 13,000
Total supply	445,114	479,524	558,221	598,679	590,721
Deductions:					
Re-exports of foreign	51,424	76,857	86,077	69,786	101,227
Exports of domestic lead	55	*Nil.*	6	*Nil.*	*Nil.*
Stock, domestic, Dec. 31	(*e*)50,000	(*e*)75,000	(*e*)94,430	(*e*)80,000	(*f*) 80,000
Foreign in bond, Dec. 31	12,897	18,462	12,695	(*e*)13,000	(*e*) 2,000
Total deductions	114,376	170,319	183,208	162,786	183,227
Delivery	330,738	309,205	(*e*)351,000	(*e*)435,893	407,494

(*e*) Partly estimated. (*f*) Assumed.

The greatest imports came from Mexico, but this amounted to only 85,101 tons in 1911 as compared with 98,535 in 1910 and 107,369 short tons in 1908.

LEAD PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Australasia. (c)	Austria. (a)	Belgium (a)	Canada (a)	France. (a)	Germany (a)	Greece. (a)	Hungary (a)	Italy. (a)
1898	1,997	10,340	19,330	14,477	10,920	132,742	19,193	2,305	24,543
1899	4,951	9,736	15,700	9,917	15,981	129,225	19,059	2,166	20,543
1900	7,014	10,650	16,365	28,648	15,210	121,513	16,396	2,030	23,673
1901	3,963	10,161	18,760	23,537	21,000	123,098	17,644	2,029	25,796
1902	4,956	11,264	19,504	10,411	18,817	140,331	14,048	2,243	26,494
1903	7,417	12,162	22,263	8,226	23,258	145,319	12,361	2,057	22,126
1904	8,056	12,645	23,470	17,241	18,800	137,580	15,186	2,104	23,475
1905	2,677	12,968	22,885	25,391	24,100	152,590	13,729	2,146	19,097
1906	2,919	14,846	23,765	24,580	25,614	150,741	12,308	1,925	21,268
1907	25,324	13,598	27,450	21,660	24,800	164,079	13,814	1,468	22,978
1908	22,382	12,669	35,650	19,593	26,112	164,079	15,892	1,544	26,003
1909	21,057	12,941	40,306	23,295	26,927	167,920	14,948	(e)1,590	22,133
1910	23,962	(e)13,100	40,715	14,967	20,226	159,851	(e)16,710	(e)2,077	(e)14,495
1911	19,352	(d)15,000	(d)30,800	10,673	(d)23,000	161,287	(d)14,300	(e)2,000	(d)16,700

Year.	Japan. (a)	Mexico. (a)	Russia. (a)	Spain. (a)	Sweden (a)	United Kingdom. (a)		United States.	Totals. (f)
						Foreign Ores.	Domestic Ore.		
1898	1,705	71,442	241	198,392	1,559	23,239	25,761	207,271	777,314
1899	1,989	84,656	322	184,007	1,606	17,571	23,929	196,938	738,296
1900	1,877	63,827	221	172,530	1,424	10,738	24,762	253,204	770,082
1901	1,806	94,194	156	169,294	988	19,639	20,361	253,944	806,370
1902	1,644	106,805	225	177,560	842	9,113	17,987	254,682	816,926
1903	1,728	(b) 94,181	106	175,109	678	14,900	20,278	256,188	818,307
1904	1,803	(e) 95,010	90	185,862	589	6,888	20,155	278,634	847,588
1905	2,272	(b)101,196	700	185,693	576	7,517	20,977	290,472	884,986
1906	4,305	(b) 73,699	907	185,470	753	6,984	22,691	323,567	896,342
1907	3,067	(b) 76,158	520	(d)185,800	813	10,880	24,850	322,854	939,321
1908	2,910	(b)127,010	523	188,062	277	11,480	21,336	284,858	960,380
1909	(e)3,337	118,186	(d) 790	179,993	166	(e)12,000	22,822	(g)334,832	1,003,243
1910	(e)3,845	120,662	(d)1,200	(e)151,975	(e)355	(e)10,500	21,668	(g)355,183	971,492
1911	(d)3,500	124,605	(d)1,000	(d)171,000	(e)500	(e) 9,000	(e)18,100	(g)363,829	984,646

(a) From official reports of countries unless otherwise denoted. (b) Exports. (c) Pig lead smelted in New South Wales and Queensland (d) As reported by Metallgesellschaft, Frankfurt am Main. (e) Estimated. (f) The totals may be high on account of duplications which cannot be eliminated. (g) From *Eng. and Min. Journ.*

LEAD MARKETS IN 1911

New York.[1]—Conditions in this market have not changed from those of previous years; the largest producer and seller of this metal maintains the market as steady as possible and no wide fluctuations occur. The largest consumers obtain their requirements on average prices, so that they are forced to enter the market only at times when an unexpected expansion in business compels them to increase their consumption of lead on account of insufficient regular supplies under contract. Production

[1] *E. and M. J.*, Jan. 6, 1912.

of the metal has remained on about the same level as during last year, and as no new mines of consequence have been discovered in this country, any increased smelting and refining capacity would not in itself mean an increased supply of lead.

From the first of the year until the beginning of October no change was made in the quotations of the largest producer, which were 4.425 cents, East St. Louis, and 4.50 cents, New York, for desilverized lead. Only at such times when lead produced in Missouri accumulated and was offered very freely were these quotations shaded by outside sellers; so at the end of March and the beginning of April lead was offered in St. Louis down to 4.225 cents and could, therefore, be sold in the New York market at about 4.375 cents.

When the usual spring business did not develop, the lead which had been held for the expected improvement was pressed for sale, so that at the beginning of June outside quotations were lowered to 4.20 cents and 4.35 cents, St. Louis and New York respectively. The belated spring business, however, came during June, and this demand quickly took care of the supplies that were pressing on and demoralizing the market. From the end of July and all through the month of August there was no difference between the quotations of the largest and outside sellers.

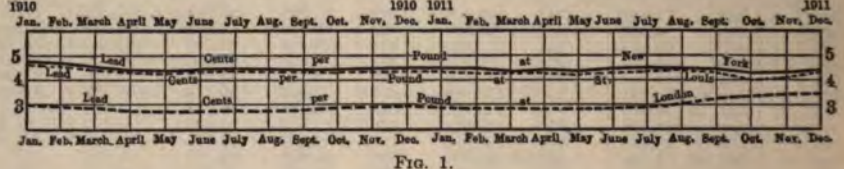

FIG. 1.

The market eased up somewhat during the month of September. At the beginning of October the leading interest suddenly reduced its price by $3 per ton which was quickly followed by an additional cut of $2 per ton, and the outsiders promptly followed suit. At this lower level, which was maintained until the middle of November, a very heavy demand developed and large purchases were made. In consequence thereof, prices were advanced to the extent of 10 cents twice in quick succession during the latter part of November, thus establishing a price of 4.375 cents, East St. Louis, and 4.45 cents, New York. Business continued satisfactory, and the year closed on that basis. The fluctuations in price are shown graphically in the adjoining diagram, Fig. 1.

London.—English brands rule 5s per ton higher than Spanish lead. The slight fluctuations are given in the table. For details and causes the reader is referred to *Engineering and Mining Journal*, Jan. 6, 1912.

AVERAGE MONTHLY PRICE OF LEAD PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*
1898...	3.65	3.71	3.72	3.63	3.64	3.82	3.95	4.00	3.99	3.78	3.76	3.76	3.78
1899...	4.18	4.49	4.37	4.31	4.44	4.43	4.52	4.57	4.58	4.58	3.70	4.64	4.47
1900...	4.68	4.68	4.68	4.68	4.18	3.90	4.03	4.25	4.35	4.35	4.58	4.35	4.37
1901...	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.15	4.33
1902...	4.00	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.069
1903...	4.075	4.075	4.442	4.567	4.325	4.210	4.075	4.075	4.243	4.375	4.218	4.162	4.237
1904...	4.347	4.375	4.475	4.475	4.423	4.196	4.192	4.111	4.200	4.200	4.200	4.600	4.309
1905...	4.552	4.450	4.470	4.500	4.500	4.500	4.524	4.665	4.850	4.850	5.200	5.422	4.707
1906...	5.600	5.464	5.350	5.404	5.685	5.750	5.750	5.750	5.750	5.750	5.750	5.900	5.657
1907...	6.000	6.000	6.000	6.000	6.000	5.760	5.288	5.250	4.813	4.750	4.376	3.658	5.325
1908...	3.691	3.725	3.838	3.993	4.253	4.466	4.477	4.580	4.515	4.351	4.330	4.213	4.200
1909...	4.175	4.018	3.986	4.168	4.287	4.350	4.321	4.363	4.342	4.341	4.370	4.560	4.273
1910(a	4.700	4.613	4.459	4.376	4.315	4.343	4.404	4.400	4.400	4.400	4.442	4.500	4.446
1911...	4.483	4.440	4.394	4.412	4.373	4.435	4.499	4.500	4.485	4.265	4.298	4.450	4.420

(a) From *Eng. and Min. Journ.*

AVERAGE MONTHLY PRICE PER 2240 lb. OF LEAD AT LONDON. (a)
(In pounds sterling.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1898....	12.508	12.362	12.650	13.062	13.700	13.437	12.950	12.800	12.800	13.050	13.412	13.100	12.983
1899....	13.375	14.350	14.150	14.375	14.146	14.283	14.385	14.733	15.267	16.179	17.096	16.883	14.933
1900....	16.296	16.542	16.612	16.733	16.900	17.225	17.533	17.633	17.667	17.596	17.229	16.233	16.987
1901....	15.925	14.667	13.379	12.421	12.275	12.342	12.150	11.692	11.954	11.600	11.267	10.533	12.521
1902....	10.567	11.617	11.508	11.596	11.600	11.271	11.233	11.121	10.892	10.746	10.717	10.754	11.262
1903....	11.304	11.708	13.225	12.404	11.800	11.437	11.383	11.146	11.167	11.108	11.108	11.179	11.579
1904....	11.558	11.592	12.037	12.254	11.754	11.521	11.667	11.737	11.787	12.187	12.892	12.775	11.983
1905....	12.875	12.462	12.296	12.658	12.762	13.000	13.608	13.958	13.950	14.679	15.337	17.050	13.719
1906....	16.850	16.031	15.922	15.959	16.725	16.813	16.525	17.109	18.266	19.350	19.281	19.609	17.370
1907....	19.828	19.531	19.703	19.975	19.688	20.188	20.350	19.063	19.775	18.531	17.281	14.500	19.034
1908....	14.469	14.250	13.975	13.469	12.938	12.600	13.000	13.375	13.125	13.375	13.538	13.156	13.439
1909....	13.113	13.313	13.438	13.297	13.225	13.031	12.563	12.475	12.781	13.175	13.047	13.125	13.042
1910....	13.650	13.328	13.063	12.641	12.550	12.688	12.531	12.513	12.582	13.091	13.217	13.197	12.920
1911....	13.009	13.043	13.122	12.889	12.984	13.260	13.530	14.260	14.744	15.332	15.821	15.648	13.970

(a) The statistics for 1897–1905 are from the report of the Metallgesellschaft, Frankfurt-am-Main. Those for subsequent years are from the *Engineering and Mining Journal.*

White Lead in 1911[1]

The consumption of all paints in 1911 was appreciably below that of 1910, the reduction being estimated at about 25 per cent. This is attributed chiefly to the high cost of linseed oil, the price of which averaged higher than for any year since the era of high values following the Civil War. While the difference between this average and that of the last 10 years would not represent a material addition to the cost of painting an ordinary house, the effect of practically doubling what had come to be regarded as a normal price for oil was to make property owners defer painting to such an extent as to cause the deficiency in consumption already noted. This effect extended to the ready-made paints in the same degree, by reason of the advance in prices which manufacturers were compelled to make. With a prospect for lower prices of oil, as a result of a larger crop of flaxseed, both here and in the foreign producing coun-

[1] *E. and M. J.*, Jan. 7, 1912.

tries, notably in the Argentine, the outlook for 1912 is regarded as more encouraging.

Lead Carbonate.—In addition to the retarding influence of the high cost of oil, the white-lead industry has had to meet a growing competition in the lower grades of paint made from lithophone, a chemical product containing zinc and barium. This pigment seems to possess some advantages in the manufacture of cheap paints over mechanical combinations of white lead, zinc and barytes or gypsum. It cannot be used in combination with lead, because the presence of the zinc sulphide causes the lead to turn black, and it does not appear to be a serious competitor of white lead, except in the cheapest grade of paints.

The steadiness of pig lead which was practically unchanged in price from late in November, 1910, until the early part of October, 1911, was reflected in white lead which was without change from the list in force from the close of 1910 until the last of October, 1911, when a reduction of 1/4 cent was made in the price of dry lead, following a decline of the same fraction in pig lead. This brought the price down to 5 cents, at which contracts with paint manufacturers and other large consumers were made, covering a good share of their requirements for 1912. The advances of pig lead covering 20 cents of the previous decline of 25 cents per cwt., which occurred within a month after white lead had been reduced, made these contracts favorable for the buyers, and correspondingly disastrous to the sellers, who had had no opportunity to accumulate any considerable stock of lower-cost metal.

The reduction in the price of dry white lead left the margin between the pigment dry and in oil wider than has prevailed for many years, and with an accumulation of stocks in the hands of some of the corroders, the temptation to break prices in the hope of getting a portion of the stock into the hands of distributors proved irresistible and following some special inducements by one of the smaller Western companies came a break of 1/2 cent in the card price by another company which was speedily followed by all corroders. This made the closing price 6 3/4 cents for lead in oil in packages of 100 lb. and upward, with a differential on the smaller sizes, with dry lead at 5 cents to large buyers.

Lead Oxide.—Red lead was irregular in price during 1911; the bulk of the business was at 6 1/2 cents for large packages and some at 6 1/4 cents. There was even more irregularity in the price of litharge, on which there is a keener competition than on red lead. Large consumers were paying 5 3/4 cents during the early part of 1911 but during the later months they were able to buy as low as 5 1/4 cents and at the close that appeared to be the quotable price to carload buyers, and large packages in less than

carload lots were not quoted above 5 1/2 cents. As a whole the oxide business is not on a profitable basis to the manufacturer.

PRODUCTION OF LEAD PIGMENTS IN THE UNITED STATES.

Year.	Red Lead.		White Lead. (a)		Litharge.		Orange Mineral.	
	Short Tons	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897...	7,798	$744,709	105,804	$9,522,360	8,591	$773,190	477	$76,320
1898...	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200
1899...	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200
1900...	10,098	1,050,192	96,408	9,910,742	10,462	1,067,124	825	100,650
1901...	13,103	1,448,550	100,787	11,252,653	9,460	979,586	1,087	224,667
1902...	11,669	1,262,712	114,658	11,978,172	12,755	1,299,443	867	138,349
1903...	12,300	1,385,900	112,700	12,228,024	12,400	1,326,800	1,000	168,000
1904...	13,938	1,672,569	126,336	13,896,913	12,487	1,248,691	1,125	168,681
1905...	16,269	1,919,767	122,398	12,068,443	12,643	1,422,616	1,000	120,000
1906...	13,693	1,874,448	123,640	15,234,990	13,816	1,890,050	2,927	421,488
1907...	13,370	1,778,717	111,409	12,254,297	14,769	1,624,553	815	123,917
1908...	11,358	1,156,282	116,628	10,515,315	12,254	1,231,206	393	43,157
1909...	15,800	1,438,197	131,643	12,652,638	13,391	1,266,903	530	68,003
1910 (b)	16,116	1,482,672	134,276	13,024,762	13,659	1,283,940	541	70,330

(a) The output of "sublimed white lead," a mixed sulphate and oxide of lead, is not included in 1904–10. (b) Estimated.

IMPORTS OF LEAD PIGMENTS INTO THE UNITED STATES

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1897.....	1,386,070	$46,992	1,101,829	$48,988	60,984	$1,931	1,486,042	$67,549
1898.....	682,449	25,780	506,739	24,334	56,417	2,021	795,116	37,745
1899.....	776,197	30,479	583,409	30,212	55,127	3,614	1,141,387	58,142
1900.....	549,551	25,532	456,872	28,336	77,314	2,852	1,068,793	61,885
1901.....	485,466	19,369	384,671	21,226	49,306	1,873	977,644	52,409
1902.....	1,075,839	37,833	506,423	25,320	88,115	2,908	997,494	49,060
1903.....	1,152,715	40,846	453,284	24,595	42,756	1,464	756,742	36,407
1904.....	836,077	30,115	587,383	33,788	44,541	1,500	766,469	37,178
1905.....	704,402	26,553	597,510	34,722	117,759	4,139	628,003	31,106
1906.....	1,093,619	50,741	647,636	41,233	87,230	3,737	770,342	42,519
1907.....	679,171	35,959	584,309	37,482	90,475	4,386	615,015	37,799
1908.....	645,073	28,155	540,311	30,451	96,184	3,327	485,407	26,645
1909.....	760,179	30,428	694,599	39,963	90,655	3,740	496,231	27,562
1910.....	822,289	32,750	686,052	38,919	48,693	2,252	600,461	32,199
1911.....	1,163,533	46,170	741,071	46,213	24,662	1,196	504,734	28,515

Lead Mining in the United States

Arizona.—The lead production in Arizona was more than three times increased. This is due more particularly to the increased output of the Warren district of Cochise County and also to an increased production from the Cerbat Range in Mohave County.

In 1910 the principal producers were as follows:

	Tons of Lead.
Copper Queen Consolidated, Bisbee	348
Herschel M. and M. Co., Tombstone	30
Tombstone Consolidated Mines Co	22
C. O. D. M. Co., Mohave County	13
Needles Mining and Smelting Co	72
Union Basin M. Co	16
Penn Arizona Mining Co	33
Total	534

California.—In California there was no increase in the lead production and it was all obtained from Inyo County. The Needles Smelting Company draws its own supplies chiefly from Arizona.

LEAD PRODUCTION OF COLORADO.(a)
(In tons of 2000 lb.)

County.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Clear Creek..	1,726	1,981	1,631	1,439	1,832	1,205	1,495	1,228	1,956
Hinsdale.....	230	521	446	442	470	82	50	124	60
Lake.....	18,177	23,590	26,424	23,918	17,032	7,169	7,061	6,364	8,119
Mineral.....	4,300	6,673	5,940	7,443	6,490	4,119	4,526	4,066	3,501
Ouray.....	1,675	1,022	2,674	2,861	1,803	1,516	1,354	2,021	2,002
Pitkin.....	16,635	9,441	10,987	8,781	6,957	3,713	6,315	6,704	5,090
San Juan....	3,485	4,644	3,223	2,070	6,213	5,133	4,921	5,444	3,140
Others.....	4,529	5,901	6,531	6,038	5,696	5,918	6,638	3,332	8,891
Total.....	50,757	53,773	57,856	52,992	46,493	28,855	32,360	35,283	32,759

(a) As reported by the State Commissioner of Mines.

Colorado (G. E. Collins).—No great development of lead can be looked for in Colorado[1]. The largest aggregate production of lead is made in the San Juan, particularly at Silverton. Here the chief individual producing mines are the Iowa, and the Silver Lake, which is now being operated again on a small scale by the owners. Considerable bodies of mixed lead-zinc ore have been opened up in the Silver Ledge. At Telluride the Alta is a large producer of lead ore.

Aspen continues to produce a large tonnage of low-grade silver-bearing lead ore. The same may be said of Creede, where the various claims along the Amethyst vein are actively producing lead-zinc concentrating ores, and where a project is under way to sink below the tunnel level. The principal lead resources of the state are intimately associated with zinc, and usually also with pyrite. Such mixed-sulphide ores are capable of making a great aggregate production, the chief difficulty being that most of the individual mines are small, and therefore do not warrant the provision of the elaborate separating plant necessary to produce marketable products. Large orebodies of this character exist in the Iron Mask and other mines at Red Cliff; and the Wellington mine at Breckenridge may also be cited as a case where complex lead-zinc ores are being treated successfully. Other important deposits occur at Rico, at the Mary Murphy, in Chaffee County, and many other localities scattered throughout the state. At Bonanza a deep tunnel is being driven to unwater and develop in depth the Rawley mine, which promises to become a considerable factor in the lead output.

Idaho.[2]—The mining industry of the Coeur d'Alene during 1911 en-

[1] *E. and M. J.*, Jan. 6, 1912.
[2] Abstract from State Mine Inspector Robert N. Bell's report, for 1911.

joyed a period of quiet prosperity. The lead production totalled 275,000,000 lb. None of the producers dropped out and an increased production was made from the Hercules Mining Co. and a slightly increased production from the Federal Mining & Smelting Co. This, together with the production from the Stewart Mining Co., makes a greater yield than for 1910. The zinc production was materially increased. Dividend disbursements in 1911 were not far from $2,400,000 for the mines of the district. At no time were there less than 2200 miners at work.

The Bunker Hill and Sullivan Mine continues to be the largest producer in the Coeur d'Alene. Its daily output is twelve to fourteen hundred metric tons of concentrating ore and is now largely from the levels driven from an inclined shaft sunk below the Kellogg tunnel. The famous Foot Wall Fracture now forms the hanging wall of the largest and richest ore stopes in the mine. This fault fissure is a secondary ore channel. On account of the extensive new ore disclosures the company is looking forward to equipping for deep work. The company now also owns the Last Chance Mine formerly belonging to the Federal Mining and Smelting Company. The compromise between the two companies stipulates certain ore resources above a certain level to be mined by the old company. During the year electric haulage and other improvements were installed. At the portal of the Arizona tunnel a sorting plant was erected and the life of the mine has been materially increased. The Senator Stewart mine continued a steady output of concentrating ore between 400 and 500 tons per day. This ore is shipped to the Mammoth Mill at Wallace. During the past year a new double drum hoist was installed and also a long cross-cut tunnel started from Deadwood Gulch. The rate of driving this tunnel has been 350 to 400 ft. per month. The Caledonia mine has been operated steadily for the past 2 years and its tunnel which be will 2800 ft. in length is nearing completion. This mine has the record for high grade silver ore and during the past year one carload from the five hundred level ran 486 oz. of silver, 16 per cent. copper, and 7 per cent. lead. During the year an ore body making on a fault of slight displacement was developed 90 ft. in width; sorted, it ran 40 per cent. lead, 2 per cent. copper and 2 oz. of silver for each unit of lead. Quantities of concentrating ore are stored for future milling. At Mullan the Morning Mine has put much ore in sight during the past year. Iron and zinc sulphide in the lower levels, however, are associated so as to make the separation difficult. The mill has a capacity of 1200 tons per day and experiments are being made to avoid the losses which are due to the complexity of the ore at depth. The annual report of the Federal Mining and Smelting Co. for the year ending Aug 31, 1911, shows net earnings of $1,241,115, an increase

of $497,307 over 1910, accounted for chiefly by the Morning and Mace mines. It is expected that the earnings from Omaha ground leased to the Caledonia Co. will cease in 1912 on account of exhausted ground. Although the Macquisten tube at the Morning Mill earns $1500 per month, or a profit of $1 per ton of ore treated, the saving was not as much as expected in lead, and was largely due to zinc. The cost of the unit was $35,000. Only 50 per cent. of the lead is recovered while other mines recover 80 per cent. At all the properties 784,600 tons of ore were mined in 1911, yielding 118,315 tons of concentrates, averaging 44.18 per cent. lead, 27.9 oz. silver per ton and 531 tons of zinc concentrate averaging 46.9 per cent. zinc.

Ore in sight for milling was 863,340 tons. At the Wardner the metal contents is 3 per cent. less than before, the reserves being practically unchanged. This mine is shipping 550 tons per month of first class ore obtained by sorting. The tonnage in sight will last 18 months. At the Mace the milling ore in sight decreased 10 per cent., and there was a decrease of 16 per cent. in metal contents of first class ore and concentrates. The reserves will last 13 months. At the Morning the reserves have decreased one-third but the quantity of first class ore is 16,000 greater. Among other lead mines operating regularly in 1911 are the Standard Mammoth, the Hecla and the Hercules.

Southwestern Missouri (Jesse A. Zook[1]).—Lead shipments were 3000 tons in excess of 1910.[2] The lead production was less variable than in most preceding years. The advent of the Webb City smelter into the field had no immediate effect on prices as local producers owned a large amount of stock, and from their production the bulk of the smeltery's needs was drawn. The average monthly shipment was 3685 tons. The new smelter is located on the Guinn land north of Webb City in the heart of the sheet-ground district. The plant has a capacity for handling 500 tons of ore per week. A fuller description of the Joplin district, which is principally a zinc producer, is given under zinc.

The district fell off slightly in the production of zinc blende and silicate concentrates but gained in the output of lead concentrates. The production was 250,000 tons of blende concentrates, 20,000 tons of silicate, and 45,500 tons of lead concentrates. The active competition for lead ore, stimulated by the opening of the new smelter, is credited with the increase in lead ore production.

[1] Abstract *Engineering and Mining Journal*, Jan. 6, 1912.
[2] See under "Zinc," for table showing Joplin lead production, by districts

PRICE OF LEAD ORE AT JOPLIN.
(Per ton of 2000 lb.)

Year.	1902.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Highest.....	$50.00	$60.50	$62.00	$80.00	$87.00	$88.50	$66.00	$60.50	$58.00	$64.00
Average.....	46.10	54.12	54.80	62.12	77.78	68.90	55.75	54.58	51.98	56.76

[1] See under "Zinc," for table showing Joplin lead production, by districts.

PRODUCTION OF LEAD ORE IN THE JOPLIN DISTRICT.
(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1896.....	27,721	1900..	29,132	1904..	34,362	1908..	38,533	.....	.....
1897.....	30,105	1901..	35,177	1905..	31,679	1909..	43,659	.....	.....
1898.....	26,687	1902..	31,625	1906..	39,189	1910..	44,037	.....	.....
1899.....	23,888	1903..	28,656	1907..	42,065	1911..	45,500	.....	.....

SOUTHEASTERN MISSOURI IN 1911[1]

BY H. A. WHEELER.

The southeastern Missouri lead district produced about 155,000 tons of pig lead in 1911 which had a value at St. Louis of approximately $13,430,000. This magnificent production maintains the prestige of this district as the world's greatest lead producer. St. Francois County has been the leader of this district since the discovery of the deeper or disseminated lead about 40 years ago at Bonne Terre, and in 1911 contributed about 97 per cent of the output.

It seems quite probable that Washington County and also Jefferson County are underlain by the disseminated deposits, like those in St. Francois and Madison Counties associated with the old shallow diggings.

The large mills are drawing heavily on the reserves of St. Francois County, and it is going to require energetic prospecting to open up new orebodies to maintain the present heavy production than has been pursued since the panic of 1907. Important interests were beginning to prospect in Washington County just prior to the panic, but since then no further drilling has been attempted. Some quiet investigations were made by Bostonians last year but it was directed towards property investments rather than development work.

Madison County, at the southern end of the lead belt, made the smallest output in many years, as only one property was operated. The North American Lead property, at Fredericktown, formerly controlled by Columbus, Ohio, capitalists, was taken over by the bond-holders under a foreclosure sale and is now held by the Dominion Nickel Co. of Sudbury, Canada. As it changed in recent years from lead to a pro-

[1] Abstract from article *E. and M. J.*, Jan. 6, 1912.

ducer of copper, nickel and cobalt, with lead as a by-product, it will probably be kept closed, to prevent disturbing the nickel market. The Catherine mine was examined under a lease and preparations are being made to reopen this property, as also the adjoining Fleming land, on a royalty basis.

The St. Joe company had a successful year in 1911, and through its affiliation with and control of the output of the Doe Run Lead Co., its smeltery at Herculaneum is the largest producer of pig-lead in the country. After struggling for years with bad roads, poor teams and wretched service at a continuous loss, this company built a 13-mile narrow-gauge road and solved the problem of changing an annual deficit into a profit; at present it is about 64 miles long, including branches.

The old or No. 2 mill at Bonne Terre was continuously operated throughout 1911 as also the No. 3 mill at Leadwood, each of which has a capacity of 1500 tons per day. They were supplied by seven mines located at Bonne Terre (3), Flat River, Gumbo, Huntington and Leadwood. The smeltery at Herculaneum is now equipped with four Dwight-Lloyd sintering furnaces for roasting the concentrates, which have replaced the Savelsberg pot-roasting system. A MacDougal furnace has been erected for roasting matte, which replaces a Pierce turret roaster.

No increase has been made in the land holding of the St. Joe company, which totals about 20,000 acres, of which only about 160 acres have been exhausted. The usual 6 per cent. dividends were paid in 1911 on the $10,000,000 issued stock, which brings the total paid to $7,958,357.

In 1911 the Doe Run Lead Co., produced about 32,000 tons of lead. This was due to its new, or No. 3, mill at Rivermines being increased from 2000 to 3000 tons per day. The old, or No. 1 mill at Doe Run, of 1200 tons capacity, was operated on ore from the new or No. 12 shaft at Doe Run, and also from the No. 3 shaft at Flat River. The old No. 2 mill on the former Columbia property has not been operated since the erection of the No. 3 mill.

This large production was obtained from six mines, which are situated on spurs of the M. R. & B. T. R. R., as follows: Nos. 1 and 4 shafts at Flat River, No. 3 at Rivermines (formerly Central Station), No. 6 at Elvins, No. 8 at Mitchell, No. 9 at Columbia (Red Onion) and No. 12 at Doe Run. The No. 12 shaft is equipped with electric-driven machinery, driven by a 300-kw. generator, which is direct-connected with a low-pressure turbine. The latter is supplied with the exhaust from two noncondensing corliss engines that drive the No. 1 or Doe Run mill.

The electric-power station at River mines supplies current for operat-

ing the six shafts and the 3000-ton mill in the Flat River district. It is equipped with four Snow twin-tandem 4-cycle gas engines of 800-h. p. that furnish 900,000 kw. per month. Recently two 500-h. p. independent Westinghouse continuous gas producers were installed. The No. 3 mill was increased from a daily capacity of 200 tons to 300 by adding two Hancock jigs and four sets of rolls, with the necessary auxiliaries. The plant at Doe Run was originally equipped with a small smeltery, but since the St. Joe company built the large smeltery at Herculaneum, the outputs of both properties have been smelted there. The concentrates were roasted at Doe Run until last year in a battery of Freiberg roasting furnaces, but these have been abandoned. The company continued to pay 6 per cent. dividends on its capital of $6,578,200 ($10,000,000 authorized) during 1911 and the total dividends paid to date aggregate $2,941,858.

The Desloge property made the largest output in its history. The mill was enlarged to a daily capacity of 1500 tons. Although four Flintshire or small reverberatory furnaces have always been operated, and notwithstanding that a small water-jacket shaft-furnace and three Freiberg roasters were subsequently added, most of the concentrates are smelted at either the large plant of the Federal Lead Co. at Alton, Ill., or the National Lead Co.'s smeltery at Collinsville, Ill., where the Scotch hearth and bag process are employed. The production was derived from three mines, or Nos. 3, 4 and 6 shafts and shipped over its own railroad to the mill, the latter being on Big River. Although No. 4 shaft is on Big River and the mine is working beneath it, the mine makes little water and is one of the dryest in the district.

The National Lead Co., under its local operating company, the St. Louis Smelting & Refining Co., worked steadily throughout the year.

The important subsidiary of the Guggenheim Exploration Company known as the Federal Lead Co. continued as the largest individual producer in the district, which position it has maintained since the completion of the Central mill. The latter is the largest in the district, as it has a capacity of 4000 tons per day.

The Potosi Mines Co. is the new company which acquired 25,000 acres 2 years ago about Potosi, in Washington County, which has produced large amounts of shallow lead and "tiff" or barite.

The Mine la Motte property was the only producer last year in Madison County. It was operated under the Pittsburg management that has controlled it since its meteoric exploitation by Dougherty & Albers several years ago. It was under consideration by one of the large St. Francois County operators last year, but thus far no changes have occurred in the title. It has a large acreage, said to be in excess of 35,000, but much

of it is timber, granite and sandstone land that does not contain the Bonne Terre or disseminated-lead-bearing limestone. It has a 500-ton mill that is supplied by five mines that are less than 150 ft. deep. It formerly operated its own smeltery, but in recent years the concentrates have been sold in the St. Louis market.

Montana.—The output of lead in Montana in 1911 increased nearly one-half, principally on account of larger shipments from Beaverhead, Cascade, Jefferson and Lincoln Counties. It is notable that in this state the production of zinc is also increasing. During 1911, a new plant was installed at the Barker mine were a shaft was sunk to a depth of 400 ft. The company has shipped about 51 cars of ore, averaging $41 per ton, to the American Smelting and Refining Co.'s plant at Helena. The 50-ton concentrator which was erected started on November 15, 1911 and is working successfully. It consists of one Blake crusher, two sets of rolls, three jigs and three Wilfley tables. There are other properties in the Barker district which formerly were large producers of lead-silver ores, and it is hoped that with good managment and proper development, some of these mines may be made productive again.[1]

Nevada.—Principally on account of the idleness of the Richmond Eureka, the lead production in this state fell off. The greatest output was derived from lead zinc ores and Clark County supplied most of the lead and nearly all of the zinc production.

Oklahoma.—The lead production of Oklahoma has been about the same for the last 3 years. In the Arbuckle Mountain region, one mill was completed and another was being built at the close of the year. At Peoria, the oldest lead-zinc camp in the State, a few miles from the Kansas and Missouri lines, the ore so far mined has been produced from shallow diggings seldom over 18 ft. deep. There were only 3 tons of galena and 65 tons of silicate produced in 1910, and less in 1911 (60 tons of calamine worth $1500.) Lack of transportation facilities is the great drawback. The Quapaw district had four steam concentrators out of 12 existing plants in operation during 1911. The production for 1911 was 132 tons of galena and 1411 tons of sphalerite.

The Miami camp is 4 miles north of the City of Miami. The mining is troubled considerably with water and also with bitumen in the crevices and pockets in the rocks. The water is heavily charged with hydrogen sulphide which makes much trouble for the men who work in it. The southern end of this camp has practically been worked out but a mill is to be erected on the Carson-Dobson lease and shafts are to be sunk on the Thirty Acre Mining Company's lease and the Chapman-Lennon lease. The runs to the west are worked by a number of mills,

[1] J. R. Villars, *Engineering and Mining Journal*, Feb. 3, 1912.

among which the Consolidated and the Southern Queen produced concentrates. The production of the Miami camp in 1911 was 2365 tons galena and 7840 tons of sphalerite.

Utah.—The Utah production of lead increased 8 per cent. most of it coming from the Park City district. The Bingham district produced as usual but the Tintic district fell off. At the Little Bell mine in the Park City district, the minerals are galena and pyrite. The shipments approximate 21 per cent. lead, with 76 oz. of silver, 0.05 oz. of gold and 2.25 per cent. copper. The connection has been made at the 900 ft. level with the 700 ft. level of the Daly West, thus securing ventilation and drainage. The Daly West is also furnishing compressed air for the operation of this property. Considerable ore has been accumulated and a mill with 100-ton capacity per day has been built.[1]

The Bingham production increased from the Utah Apex, the Highland Boy and Yampa.

The Utah Apex developed important bedded deposits of silver-lead ore, which were opened in December, 1910. The Parvenu tunnel orebody was followed 400 ft. on the strike, 100 to 150 ft. in width, with a known thickness of 90 ft. at one point and an average thickness where opened of 60 ft. Up to September, 28,000 tons of first-class ore from this deposit were shipped and for the balance of the year 200 tons per day were mined, or a total of 24,000 tons. During this period 115 tons of ore and concentrates were shipped daily. At the beginning of September the ore reserves were 143,000 tons, of which ore developed amounted to 83,000 tons and partially developed to 60,000 tons. Mining was discontinued November, 1910 and resumed April, 1911, during which time work was devoted to development. A mining profit of $121,299 was made for the 6 months.

The Utah Consolidated Mining Co., during 1911 shipped to the International Smelting and Refining Co., 7793 tons of lead ore averaging 22.164 per cent. lead, .655 per cent. copper, 5.41 oz. silver, .0854 oz. gold, also 512 tons of concentrates assaying 21.98 per cent. lead, .609 per cent. copper, 5.04 oz. silver, .079 oz. gold. The lead concentrates were produced at the Winnamuck mill, Bingham, from 2524 tons of second-class ore. Important new lead-ore bodies were opened extending 300 ft. along the contact and 3 ft. average width. The ore reserves, as far as lead ore is concerned, are 23,785 tons averaging 15 per cent. lead, 1.04 per cent. copper, 3 oz. silver, .023 oz. gold, 12,000 tons of second-class ore averaging 6.8 per cent lead, 1.10 per cent. copper, .35 oz. silver, .035 oz. gold.

[1] Edward R. Zalinski, *Engineering and Mining Journal*, June 3, 1911.

Lead Mining in Foreign Countries

Australia.—The only refinery in Australia working at the present time, is that of the Broken Hill Proprietary Company at Port Pirie, South Australia. This refinery contains five units, each consisting of copper softening furnace, antimony softening furnace, two desilverizing kettles, refining furnace and molding kettle, two double retort furnaces, eight cupel furnaces, and one reverberatory furnace for treating retort drosses. A complete description of this plant is given by W. Poole,[1]

(F. S. Mance[2]).—The value of the output from the silver-lead mines

LEAD PRODUCTION IN NEW SOUTH WALES. (a)
(In tons of 2240 lb.)

Year.	Pig Lead.		Silver-lead Ore and Concentrates.	
	Quantity.	Value.	Quantity.	Value.
1909	15,476	£186,073	269,306	£1,484,641
1910	21,195	248,561	317,697	1,685,704
1911	17,276	209,784	338,469	2,265,669

(a) According to the official statistics of New South Wales.

of New South Wales for the year 1911 constitutes a record. The chief contribution is supplied by the lead-zinc mines at Broken Hill. Each

SILVER-LEAD PRODUCTION OF NEW SOUTH WALES.

Year.	In Bullion.		In Concentrates Exported.	
	Silver.	Lead.	Silver.	Lead.
	Oz.	Tons.	Oz.	Tons.
1908	6,484,288	103,371	5,499,381	69,501
1909	3,717,016	64,821	6,867,775	90,307
1910	5,916,323	94,818	7,608,336	85,035
1911	5,731,468	94,966	8,797,677	111,795

year witnesses the attainment of greater excellence in milling practice, with a corresponding improvement in recoveries, and a reduction in working costs, and this despite the fact that the ore is being mined from deeper levels, and is of poorer grade, while wages have been materially increased. The output would have been still greater if more skilled labor could have been obtained.

The lode in the bottom levels of the South mine is maintaining its great size, although at the 1070-ft. level the work so far done showed

[1] Director School of Mines, before the Sydney Univ. Engineering Society.—Abstract *Mines and Minerals*, Feb. 1912.
[2] *E. and M. J.*, Jan. 6, 1912.

that the ore-body is more irregular than at the 970-ft. level. In the North mine a fine body of ore is being followed at the 1100-ft. level. Owing to the relocation of the lode which was previously missed, the British mine has been given a new lease of life. In the parent mine, the Proprietary, there were no new developments in the lower levels; the old fire area in blocks 13 and 12 is being attacked at the 300- to 500-ft. levels. The Central mine (Sulphide Corporation) possesses good ore reserves, and has furnished a splendid output. The large low-grade lode which is being opened up by the South Extended Co. gives promise of a successful future for the mine. It is estimated that the ore reserves at Broken Hill in sight at present are sufficient for 9 years. From the Yerranderie field, N. S. W., high-grade silver-lead ore had been supplied in considerable quantity.

In Tasmania the cessation of smelting operations by the Tasmanian Smelting Co. affected the output from the mines at Zeehan during the first half of the year, but owing to the timely intervention of the Government, the smelteries were again put into commission, and during the second half of the year operations proceeded with marked energy. The success of the exploratory work carried on at the North Mount Farrell mine at depth has stimulated mining at this center. The silver lead ore produced in 1911 was worth $1,232,981, as compared with $1,204,828 in 1910.

In Queensland the Mungana mines were still the chief producers of silver-lead ore, the bulk of the output being drawn from the Girofla mine. Operations at the Lady Jane mine have been considerably retarded by the condition of the workings which were affected by the creep and the fire.

Western Australia.—The lead ores were first found in 1848 on what is now known as the Geraldine River. Up to 1900, 33,617 tons, valued at £364,514, were produced; since then to 1908, 417 tons, valued at £2034. The ores are found in ancient crystalline schists, traversed by basic dykes, with which the ores are associated. Another occurrence is in the Ashburton district at Uaroo in sedimentaries, namely quartzites and phyllites. The quartz reef yielded 1224 tons of galena valued at £10,863 up to 1910.

Austria.—There were 22,841 tons of lead ore produced in 1910 and 15,476 tons of pig lead, most of which came from Carinthia. The state mines near Przibram are nearing exhaustion and it has been decided to shut them down for good.

Belgium.—In 1909, five smelters produced lead and silver, the metric tons of lead produced being 40,306 and the silver 271,270 kilos. In 1910 the pig lead produced was 88,499 tons and 264,699 kilos silver.

British Columbia (E. Jacobs).—A combination of adverse conditions

appears to have caused the total production of lead in 1911 to be smaller than in any other year since 1903. The largest individual decrease was that of the St. Eugene, which in 1910 produced nearly 15,800,000, lb. out of a total for the whole province of 34,652,000 lb.; its output for 1911 was approximately 35,000 tons of ore from which 5300 tons of concentrate were produced, containing 5,882,000 lb. lead and 136,000 oz. of silver.

On the other hand, the Rambler-Cariboo, Slocan, made an increase larger than any single decrease, and it may reasonably be expected to do even better in 1912, for it has comparatively large reserves of ore the stoping of which will shortly be commenced, following completion of a concentration mill and railway transportation facilities.

In both East and West Kootenay, much development work was done in preparation for an enlarged output in the future. Deep development in several Slocan mines proved the occurrence at considerable depth of ore of good grade and in quantity.

Development work was in progress at the Sullivan, Molly Gibson, and on some claims near Salmo, Nelson mining division, on which there is a large surface showing of lead carbonate ore.

At Silverton, Slocan Lake, the Van-Roi Mining Co.'s concentrator was completed last spring and has since treated about 30,000 tons of silver-lead-zinc ore, making silver-lead and silver-zinc concentrates; the Silverton Mines, Ltd., put in one unit of the Elmore vacuum process plant at its Wakefield mill, to treat zinc middlings from the jigs; and the Standard Silver-Lead Mining Co. built a 100-ton concentrating mill, also near Silverton, and commenced operations with it in November. In the coast district, satisfactory results were achieved at the Britannia mill in concentrating ore containing chalcopyrite. The Portland Canal Mining Co. increased the capacity of its concentrating mill to 75 tons per day, and made a good average saving in two classes of concentrate—iron with gold, and lead with both silver and gold.

The Standard mine[1] at Silverton has attracted much notice of late, owing to the opening of large shoots of galena ore of good grade. Development work on a systematic plan has been in practically continuous progress since 1904. The "big oreshoot" was cut by the lower levels in 1909–10.

The Slocan Star Mines, Ltd., is to acquire and operate the Slocan Star and Rabbit Paw groups, near Sandon, B. C.

Germany.—In Prussia three districts, namely the Rhineland Westphalia the Rhineland and the Silesia, the latter producing lead as a by-product, had a lead ore production in 1910 amounting to only 133,955

[1] John Vallance, Western Branch Canadian Mining Institute, May 18, 1911.

metric tons. Metallic lead was produced mainly by two works in the Breslau district of Silesia and seven others in the Bonn district. They produced together 132,171 metric tons. Together with other productions from the Hartz Mountains and the Dortmund district, the total production of Prussia was 150,276 metric tons of pig lead in 1910. There were also 3076 metric tons of litharge produced. The total production of Germany in 1911 included 140,154 metric tons of lead ore, 161,287 tons of pig lead and 3687 tons of litharge.

In Saxony, during the last 25 years, the fall in prices affected the Freiberg district, where works were bought up by the Government to avoid collapse in 1886. In 1910 subsidies amounting to 852,565 marks had to be given to keep the industries going. The Government has announced that, at the end of 1913, the mines will have to be shut down. The lead produced in 1910 was 4554 metric tons pig, and 2846 tons of manufactures.

The following table gives the imports and exports for the last 6 years.

GERMAN IMPORTS AND EXPORTS OF LEAD.
(In metric tons.)

Year.	Metal.						Ore.					
	1906	1907	1908	1909	1910	1911	1906	1907	1908	1909	1910	1911
Imports.	71,040	75,200	77,218	76,930	81,541	100,515	89,979	137,861	133,597	111,017	112,151	143,598
Exports.	27,039	38,259	29,967	31,674	31,025	32,063	1,916	1,296	1,189	2,556	2,383	3,746

Great Britain, United Kingdom.—The output of lead ore in 1910 was 28,534 long tons while the output for 1911 was only 23,900 tons. The lead mines are scattered. Derbyshire heads the list of producers. Mill Close mine, Darley Dale, is worked on a larger scale, and employs more labor than any other lead mine in England. All the machinery is uptodate. The Weardale Lead, Rookhope, Durham, is among the largest producers, and owes renewed prosperity to lateral oreshoots. The Greenside mine, Ullswater, Westmoreland, maintains its output, and is continually adopting improvements underground and at the surface, to meet the changing character of the ore. The Thornthwaite mine, Keswick, is making some needed changes to increase the output and the method of impounding sand and slime is really good and on a large scale. The Flintshire mines showed a decrease. The outlook is dark for the Shropshire mines, and not optimistic for Cumberland or Westmoreland.

The most important of all the lead-mine developments was the completion of the Milwr adit in Flintshire, which is over two miles in length

and drains the water for a depth of 900 ft., although the idle mines on its course are only 600 ft. deep. This should give an impetus to lead mining. One of the most painful events in the history of English lead mining was the closing of the Great Laxey, Isle of Man. Its overshot water wheel was a great attraction to the thousands of yearly tourists.

Italy.—Lead ores from Sardinia and Tunis are smelted at the Pertusola and the Monteponi works; 11,500 metric tons of lead and 12,000 kilogrm. of silver came from the first in 1910 and 2995 tons of lead and 2237 kilogrm. of silver from the latter. Dwight-Lloyd sinterers have been introduced.

Malaysia.—Consul General G. Cornell Tarler, Bangkok, Siam, reports the discovery of a rich lead mine in the vicinity of Kanburi and a company has been formed with a considerable amount of capital to exploit it.

Peru.—About 30 per cent. of the lead output is produced in the Yauli district of which about 15 per cent. is lead-silver ore. No attempt is made to recover the 5 per cent. lead in the Cerro de Pasco ores as a by-product. As a result of other lead prospecting, another plant is to be tried. The only lead smelter in operation is at Vesubia.

Russia.—There were 1200 tons of lead produced in Russia in 1910 compared with 800 tons in 1909 and 530 tons in 1908. It is obtained as a by-product at the Alagir works, from Sadon mine ores. In 1905 the production was about 1000 tons.

It is reported from Jena that rich deposits of lead ore have been discovered in the district of Opatow, department of Radom, Russian Poland.

Scandinavia.—The Raneus Lead and Silver mines will have an electric smelter for experiments on the ore containing 18 per cent. zinc.

Turkey.—It is estimated that the production of lead in 1911 in Turkey was 13,000 metric tons.

In Sivas, the lead ores are associated with antimony, two of the mines being worked by a British company. Near Smyrna, zinc occurs with the lead, while at Karalar, the ore is silver-bearing. The output is from 7000 to 8000 tons per year. The smelters are at Laurium. The Bulghar Maden Silver-Lead mine belongs to the peasants, they having worked it for 75 years at a price for ores fixed by the Government. At Balia-Maden, the deposits are in augite andesite contacts with carboniferous limestone; 7000 tons of galena (70 per cent. lead, 40 oz. silver), 3000 tons blende, (40 per cent. zinc) and 3000 tons raw lead and some calamine are annually produced. At Balia Karalden, a French company produced 113,000 tons of ore in 1908; in 1909 a second electrically-driven concentrator was erected.

Recent Improvements in Lead Smelting

By H. O. Hofman

Introductory—Pyhsical Properties—Alloys

New Publications.—H. F. Collins, "The Metallurgy of Lead," Griffin Co., London, Lippincott Co., Philadelphia, 2d Ed. 1910. The first edition of this alround treatise on lead and on silver in its relation to lead appeared in 1899 (Min. Ind., 1900, IX, 439). The ground covered by the new edition is the same as that of the first, the general arrangement of subjects has been changed in few instances, the original 368 pages and 157 illustrations are increased to 538 and 314 respectively. The matter for this enlargement has been largely drawn from the technical literature in which figure prominently the annual reviews of the Mineral Industry. The new edition will be welcomed by everybody interested in the metal, as a large amount of information has been made available in a convenient form. The policy has been apparently not to cut out old material, as what is antiquated in one part of the world may be just what is required in another.

Properties of Lead.—In a paper read before the Faraday Society, May 23, 1911, H. Greenwood gives the boiling points at atmospheric pressure of the following 11 metals in degrees C.: Mg 1120, Bi 1420, Sb 1440, Pb 1525, Al 1800, Mn 1900, Ag 1955, Cr 2200, Sn 2275, Cu 2310, Fe 2450.

E. Heyn ("Versuche über das Verhalten von Kupfer, Zink und Blei gegenüber Zement, Beton, und den damit in Berührung stehenden Flüssigkeiten," Ernst and Korn, Berlin, 1911, 59 pages) has published an extended study upon the behavior of copper, zinc and lead when enclosed in cement and cement concrete, and when placed thus enclosed in contact with liquids such as distilled water, faucet water, solutions of KOH, K_2CO_3 and $KHCO_3$, and of gypsum, sea water, cement water. An unexpected result is that distilled water free from CO_2 in contact with air also free from CO_2 quickly corrodes lead, and that the corrosion may sink to zero if the water contains much CO_2, as the lead becomes covered with a protective coating. For details the original must be consulted.

Lead-tin.—Mazotto (*Internationale Zeitschrift für Metallographie*, 1911, I, 289) has investigated the transformations that take place in lead-tin alloys in the solid state, about which there has been much discussion in recent years.

Lead-tin-antimony.—The constitution of this series of ternary alloys has been studied by R. Loebe (*Metallurgie*, 1911, VIII, 7, 33). Lead and tin as well as lead and antimony form eutectic mixtures; tin and antimony form a solid solution. As lead forms neither a chemical

compound nor a solid solution with either of the other two metals, the crystals separating upon cooling are the same as those of the binary alloys of tin and of antimony. There is no ternary eutectic point; the eutectic structure is composed of lead and of a solid solution of tin in antimony or of antimony in tin. The most fusible alloy is the lead-tin eutectic. Owing to the great differences in specific gravity of the lead crystals and the solid solutions of tin-antimony, the alloys show considerable segregation. The industrial alloys are composed of lead and solid solutions of tin and antimony.

The same subject has been treated by W. Campbell and F. C. Elder (*School Mines Quarterly*, 1911, XXXII, 244) whose results agree in part with those of Loebe; the authors yield doubtful points to Loebe whose methods show greater refinement.

Lead-cadmium-bismuth.—The investigation into this alloy-series by W. E. Barlow (*Zeitschrift anorg. Chemie*, 1911, LXX, 178; *Jl. Am. Chemical Soc.*, 1910, XXXII, 1390) shows that it is composed of three freezing-point surfaces terminating in a eutectic point at 91.5° C., the composition of the ternary eutectic being Pb 40.2, Cd 8.15 and Bi 51.65 per cent.

Lead-silver-tin.—N. Parravano (*Internationale Zeitschrift für Metallographie* 1911, I, 89) studied the constitution of the ternary alloys of lead-silver-tin.—Lead and silver form a eutectic; lead and tin a solid solution of lead in tin, and a eutectic; silver and tin a solid solution of silver in tin, the unstable compound Ag_3 Sn, and a eutectic Ag_3 Sn-Sn. The ternary alloys upon cooling form four different surfaces bounded by five curves. The constituents after complete solidification are, solid solutions of Ag–Sn and of Pb–Sn, and crystals of Ag_3Sn and of Sn. The conventional triangular diagram shows the curves intersecting at a transition point as well as at the ternary eutectic point; the former is near the binary eutectic Pb–Ag, and the latter near the binary eutectic Pb-Sn.

Diffusion of Alloys.—LeGris (*Revue Métallurgie*, 1911, VIII, 613) gives a number of excellent illustrations of diffusion alloys prepared, by pouring one fluid metal upon another in such a manner as to avoid mixing, by allowing diffusion to take place for some time, by cooling the alloy, by preparing sections and then reproducing the structures. Of the lead alloys three are shown Pb–Sn, Pb–Bi, Pb–Sb, Pb–Sn–Sb.

Lead Alloys in Die-casting.—According to E. F. Lake (*Iron Age*, 1911, LXXXVIII, 532), die-castings are made in steel molds into which the molten alloys are forced under pressure. In the alloys either zinc or lead is the prevailing metal. The following table gives the compositions of the lead alloys.

LEAD DIE-CASTINGS.

Alloy.	Pb.	Sn.	Sb.	Cu.
Electrotype	93	3	4	
Russian railroad car	90	2	8	
Ordinary bearing	87	6	7	
Jacobi metal	85	5	10	
Linotype	83	5	12	
Magnolia	80	5	15	
Magnolia	80	10	10	
Railroad bearing	75	10	15	
Monotype	74	8	18	
Standard white metal	71	5	24	
Graphite metal	68	15	17	
Graphite metal	65	10	25	
Railroad bearing	60	20	20	
Standard type	58	15	26	
Hoyle's metal	42	46	12	
Italian railroad car	38	37	25	

Magnolia Antifriction Metal.—The following is an analysis of magnolia antifriction metal: Pb 78.27, Sb 17.81, Sn 3.88, Cu 0.04. In preparing the alloy with pure metals, the mixture commonly used is Pb 80, Sb 15 and Sn 5 lb.; with 20-per-cent. antimonial lead, the mixture is, antimonial lead 100, pure lead 26.5, tin 6.5 lb.

Refining of Antimonial Lead.—L. E. Wemple (*Brass World*, 1911, VII, 395; *Eng. and Min. J.*, 1911, XXII, 1288) has patented a process (U. S. Patent, No. 1,006,323, October 17, 1911) for freeing antimonial lead from arsenic. See page 41.

Solder.—Soft solder (Editor, *Brass World*, 1911, VII, 193) is composed of lead and tin. There exist three grades: ordinary half-and-half, Sn 1 : Pb 1; best grade, Sn 2 : Pb 1; and wiping solder, Sn 1 : Pb 2. In manufacturing, either pure metal or scrap is used.

Lead Burning.—The Howard system of lead-burning (*Met. Chem. Eng.*, 1911, IX, 612) uses oxygen saturated with gasoline. It is claimed that better work is obtained than with the hydrogen-oxygen mixture commonly employed for this purpose, and that it is cheaper, faster and more convenient. The system is being introduced by the Linde Air Products Co., Buffalo, N. Y.

Lead Oxides.—Fay, Seeker, Lane, Fergusion (*Brooklyn Polytechnic Engineer*, 1910, X, 72; *J. Soc. Chem. Ind.*, 1910, XXIX, 877; *Eng. and Min. J.*, 1911, XCI, 406) have subjected various metallic oxides at elevated temperatures to the reducing effects of CO, H, NH_3 and CH_4 for periods of 6 hours and determined the temperatures at which the oxides began to give off oxygen. The most important oxides are given in the accompanying table.

Some of these data are very much at variance with those found by other investigators. The most striking is ZnO.

REDUCTION OF METALLIC OXIDES.

Oxide.	Degrees Cent.		
	CO.	H.	CH_4.
Pb_2O			202
PbO_2	110	150	45
Pb_3O_4	150	170	158
PbO	160	190	210
CuO	75	125	280
Cu_2O			230
CoO	140		
ZnO	170		152–159
As_2O_3	60		

Lead Silicates.—Hilpert and Nacken (*Berichte deutsch. chem. Ges.*, 1910, XLIII, 2565, *Metallurgie*, 1911, VIII, 157) have investigated the constitution of part of the series $PbO-PbSiO_3$. Their freezing-point curve is practically the same as that of Cooper-Shaw-Loomis (MIN. IND., 1910, XIX, 438). They ascertained the existence of the eutectic $3PbO.SiO_2$(?) $-Pb_2SiO_4$ freezing at 717° C., and the chemical compound Pb_2SiO_4 freezing at 740° C.; they have indicated in their diagram the compound $3PbO.2SiO_2$, corresponding to the mineral barysilite, with the eutectic $Pb_2SiO_4-3PbO.2SiO_2$ on one side, and the eutectic $3PbO.2SiO_2-PbSiO_3$ on the other, both freezing at 670° C. The melting-point of PbO is given as 870° C.

Decomposition of Carbonates.—Friedrich (*Stahl und Eisen*, 1911, XXXI, 1909) has determined the decomposition temperatures of various carbonates by heating in an electric crucible furnace in the atmosphere of CO_2 set free by heating. The procedure was the same as the one used in his decompositions of metallic sulphates (MIN. IND., 1910, XIX, 443) which gave temperatures from 100 to 200° C higher than if the decompositions are carried on in an air-current. The temperatures he found are the following: cerussite, $PbCO_3$, 315° C; smithsonite, $ZnCO_3$, 395°; siderite, $FeCO_3$, 380–415; hydro-cerussite, probably $2PbCO_3$, $Pb(COH)_2$, 430°; rhodocrosite, $MnCO_3$, 525°; magnesite, $MgCO_3$, 570°; dolomite, $CaMgC_2O_6$, 895°; aragonite, $CaCO_3$, 895°; calcite, $CaCO_3$, 895°; strontianite, $SrCO_3$, over 1100° C.

LEAD ORES

Leadville, Colo.—A statement of historic interest regarding the discovery of carbonate lead ore in the locality which later became Leadville, is the one of G. H. Loker (*Min. World*, 1911, XXXIV, 502) that the late August R. Meyer was the first man to identify the "strange mineral" found in the region then known as California Mining District, Lake

County. He concludes his communication by "When I first visited it (*i.e.*, the later Leadville) it consisted only of August R. Meyer's cabin."

Vanadium in Lead Ore.—The dumps of some old mines in the Georgetown district, Grant County, N. M., which produced considerable amounts of silver some 25 years ago, have recently been worked (Larsh, *Eng. and Min. J.*, 1911, XCI, 1248) by leasers who obtain from rough concentrating operations with jigs a concentrate assaying Va 7.5 per cent., Pb 40 per cent., Ag 317.6 oz. per ton. Considering the high value that vanadium has reached since its use in the manufacture of special steel, the occurrence is of much interest.

Purchasing Lead Ore.—Rzehulka(*Ztschft. angew. Chemie*, 1910, p. 483) discusses the general principles that govern the purchase of lead ores. He quotes the formula of Krutz, $V=\frac{P.\ T}{100}+\frac{p.\ t}{100}-X$ in which V is the gross value per 100 kgrm., P the market price of lead per 100 kgrm., T the dry assay, p the market price of silver per kgrm., t the silver assay, X the smelting charge. This formula offers nothing new and is readily translated into any standard of valuation.

Smelter Settlements and Contracts.—Barbour (*Eng. and Min. J.*, 1911, XCII, 314, 369) advocates the study of this important question by graphical representations of the different phases of a contract. The curves drawn will show clearly any irregularities by deviations from their usual parallel character. Such inequalities may be justified by the character or the amount of the ore, or they may show conditions by which the seller is treated unjustly. The first paper takes up copper, the second lead ores. For details the reader is referred to the originals.

Distribution of Silver.—E. M. Heriot (*Min. Jl.*, 1911, XCIII, 4011) gives some data regarding the varying tenor in silver of galena concentrates obtained in a mill treating a mixed lead-zinc ore. In the jig-product the coarsest galena (15 mm.) gave the highest, the finest (2 mm.) the lowest values in silver, and the intermediate grades correspondingly decreasing values. With the water-sorted materials of the spitzkasten the reverse was the case. The inferences are, that some of the silver was present in the metallic state or as argentite, and that these minerals were collected more readily with the finer and lighter than with the coarser and heavier products.

Smelting Practice

Smelting at Midvale, Utah.—L. A. Palmer (*Min. World*, 1911, XXXIV, 1023, 1075) gives a general description of the smeltery of the U. S. Smelting, Refining & Mining Co. (Min. Ind., 1910, XIX, 440). The

raw material for it is the blast-furnace flue dust from the bag-house which assays Pb 43.6 per cent., As_2O_3 21 per cent. It is treated in two Brunton furnaces. The roasted flue dust assays Pb 53.3 per cent. As_2O_3 11.3 per cent. which accounts for an expulsion of 56 per cent. of the As_2O_3. The gases from the fireplaces and the vaporized As_2O_3, leaving the furnace at 480° C., pass through a 24-in. steel elbow into a V-shaped structure at the entrance of a brick condensing chamber, 200 ft. in length, and divided in two by a central longitudinal wall. Each division has two connecting chambers, the first is 50 ft. long by 10 ft. wide by 10 ft. high, and the second 150 ft. long by 10 ft. wide by 8 ft. high. The latter has 26 vertical partition walls, 7 ft. apart and extending to within 3 ft. of the side walls, thus forming 25 zigzags through which the gases have to travel. Opposite each partition is a door, 3 by 5 ft., in the outer wall for removing the settled dust; the five doors nearer the furnace are of steel, the remainder of wood. The temperature of the first chamber is 300° C., that of the last zigzag about 30°. The condensing area of each division is 2268 sq. ft.; one division is in use while the other is being cleaned. The furnace treats 16 tons flue dust in 24 hours and produces 2 tons crude As_2O_3, 97–99 per cent. pure.

Smelting at Tooele, Utah.—A lead-smelting department is being added (*Met. Chem. Eng.*, 1911, IX, 623; *Eng. and Min. J.*, 1911, XCII, 1093) to this copper plant. Ore is to be blast-roasted in Dwight-Lloyd straight-line machines; there are to be three blast-furnaces, two of which are being erected at present. At first the daily capacity is to be 500 tons ore which is to come from Utah and Idaho. The plant is to go into blast early in 1912. The refinery is to be built near Chicago.

Smelting in the Harz Mountains.—A review of the present smelting and refining practice in this old metallurgical center (MIN. IND., 1907, XVI, 664) is given by O. H. Hahn (*Eng. and Min. J.*, 1911, XCI, 1106, 1163). At the Clausthal works, the old precipitation process has given way to blast-roasting in Huntington-Heberlein pots and smelting in 5-tuyère circular blast-furnaces 16 ft. 4 in. high. At Lautenthal the former smelting practice still seems to prevail, but the Pattinson battery of kettles, in which zinc desilverization used to be carried on, has been replaced by a modern plant built and run on American principles. The works of Altenau treating mostly copper ores, but also some lead ores, will be dismantled April 1, 1912; the same is the case with the silver-lead works of St. Andreasberg; the same fate awaits the Clausthal smeltery in the near future. The works of Oker, treating in addition to copper some lead ores, are in a flourishing condition, and promise to continue to do this as long as the Rammelsberg ore-deposit lasts. The same is the case with the smaller smelteries "Julius" and "Sophie,"

treating mainly sulphide lead-copper ores rich in blende. Here heap-roasting, followed by the recovery through leaching, of the zinc sulphate formed still prevails.

The Pape-Witter-Babe process of recovering the zinc in the old slags in the form of oxide (MIN. IND., 1910, XIX, 453, 692) is in successful operation. The smelting apparatus is a low blast-furnace patrly closed at the bottom with water-cooled grates through the spaces of which the slag formed trickles into iron cars, while the fumes are drawn off into dust chambers and there forced through bag-houses. The slag charged contains about 15 per cent. ZnO, the waste slag retains about 3 per cent.; a furnace treats about 10 tons of charge in 24 hours.

Smelting at Przibram, Bohemia.—The practice of lead smelting at these celebrated works has undergone considerable changes (O. H. Hahn, *Eng. and Min. J.*, 1911, XCII, 1039). The galena concentrate, which used to be slag-roasted in 12 long-hearth hand reverberatory roasting furnaces is now blast-roasted in 13 converters, 12 of which are in constant operation. The 8-ton converter-charge is made up of 6.4 tons raw concentrate and 1.6 tons blue billy. The blast-roasted ore, mixed with matte and refinery products, is smelted in eight circular blast-furnaces with 10 or 8 tuyères, 1 3/16 in. diameter; they are 30 ft. high from furnace to feed-floor, and 6 ft. 6 in. or 4 ft. 7 in. diameter at tuyères. The furnaces are run on a slag with 32 per cent. SiO_2 (Pb 1.5 per cent., Ag 1 oz. per ton) and a blast-pressure of 1 in. mercury; the capacity is 60, resp. 40 tons charge in 24 hours. The Stearn-Pattinson process for desilverizing the lead bullion has been replaced by that of Parkes. The lead bullion is not softened before desilverizing, but the antimony and zinc are removed afterward in a reverberatory furnace of 24 tons capacity, having a movable roof, sides lined with magnesite brick, and an air cooled fire-bridge incased in an iron box. The refined lead is drawn off into molds placed in a semi-circle, remelted in an iron kettle, and then cast into market bars.

Smelting in Casapalca, Peru.—The Casapalca smeltery (L. W. Strauss, *Min. Mag.*, 1911, V, 59) situated on the Central R. R. of Peru, 95 miles from Callao, was established in 1888; it has undergone numerous changes and is at present a custom plant treating lead-copper ores. The practice is to roast or blast-roast, to smelt in the blast-furnace and to ship the lead bullion and matte.

The ordinary roasting is carried on in two Brown single-hearth mechanical furnaces, 10 ft. wide, one 180 ft., the other 240 ft. long. The smaller furnace has six rabble-cars, the larger eight; a car has nine plows and travels at the rate of 55 ft. per minute. An extra plow is used daily to break up the hearth-accretions. Petroleum under an air-pressure

of 12 oz. serves as fuel. The raw concentrate of 4 mm. size has the following composition: SiO_2 25.0, Fe 15.6, CaO 1.5, Zn 10.0, Pb 11.0, Al_2O_3 4.4, S 24.0, Cu 5.0 per cent., Ag 45 oz. per ton; the pulverulent roasted product: SiO_2 27.0, Fe 18.0, CaO 3.5, Zn 9.2, Pb 10.3, Al_2O_3 2.1, S 7.6, Cu 6.4 per cent., Ag 50 oz. per ton. Roasting causes a reduction in weight of 10 per cent., and an elimination of about 70 per cent. of the S. The smaller furnace puts through in 24 hours 54.5 tons ore with 3.2 tons petroleum, the larger, 68 tons with 3.3 tons. The roasted ore is mixed with flue dust and burnt lime and briquetted in a Chisholm-Boyd-White 6-mold brick machine which makes per day 3000 brick, 3×4 1/2×8 1/2 in., weighing 8 3/4 lb. Blast-roasting has been recently introduced. The ore with SiO_2 25.0, Fe 15.6, Al_2O_3 3.8, CaO 9.0, Zn 11.0, S 24.4, Pb 6.0, Cu 3.5 per cent., Ag 44 oz. per ton, is crushed to pass a 3/4-in. screen, mixed with SiO_2 and CaO, moistened with 5 per cent. H_2O, and blast-roasted in a pot resembling that of Savelsberg. This is 8 ft. 8 in. diameter and 3 ft. 6 in. deep from grate to rim, the trunnions are 24 in. below the latter. The casting is 2 in. thick at the rim and tapers to 1 1/8 in. at the bottom; it weighs 11,000 lb. The grate, 1 1/8 in. thick, is slightly arched on an 11-ft. radius; it is made up of three pieces, one of which is central and stands 14 in. above the bottom of the pot. The air-holes, tapering 1 to 3/8 in. diameter, are 2 1/4 in. apart. The blast-inlet pipe is 6 in. diameter. The conical hood has eight openings, it ends in a cylinder which reaches into a stack 30 in. diameter and is raised and lowered by a hand-wheel. The pot, which rests on a carriage, is tilted by means of a worm-gear. The charge of 9 tons, introduced gradually, is worked in 12 hours with a blast-pressure reaching 9 or 10 oz. The blast-roasted material contains SiO_2 28.6, Fe 15.8, Al_2O_3 3.9, CaO 10.6, Zn 10.8, Pb 5.4, Cu 4.1, S 6.1 per cent., Ag 48 oz. per ton.

The blast-furnace, of American type, is 42×160 in. at the tuyères, of which there are 12 on a side, 6 in. diameter and 12 in. apart. The working height is 10 ft., and the top of the charge does not reach the feed floor. There are three water-jackets on a side and two on an end; the side-jackets have a 6-in. bosh in 6 1/2 ft., the end-jackets are straight, the water-space in both being 3 1/2 in. The crucible is 12 in. deep; the slag-discharges at the ends are 16 in. below the tuyères, and the matte tap at the side 12 in. below the slag-taps; the slag flows continuously into a settling pot, the overflow of which is granulated. A 2-ft. bustle-pipe supplies 260 cu. ft. effective (or 430 cu. ft. free) air per square foot hearth area at a pressure of from 12 to 15 oz. The charges (raw and blast-roasted ore, briquettes, matte, slag, lime) weigh from 4500 to 6200 lb.; they are hand-fed. The components of the charge have the following compositions:

COMPONENTS OF BLAST-FURNACE CHARGE.

	Raw Ore.		Reverberatory Roasted Ore.	Blast-roasted Ore.	Matte.	Slag.
SiO_2	17.2	47.0	27.0	28.6	.....	40.8
Fe	24.0	4.9	18.0	15.8	14	20.9
Al_2O_3	1.4	2.4	2.1	3.9	.....	5.5
CaO	tr.	6.4	3.5	10.6	.....	15.8
Zn	3.4	9.6	9.2	10.8	6	10.3
Pb	.....	7.2	10.3	5.4	6	.....
S	31.4	8.5	7.6	6.1	22	.....
Cu	13.7	1.6	6.4	4.1	50	0.3
Mn	.....	1.5	.....	.....	.....	.....
Ag. oz. p. t.	11.3	52	50	48	400	1.1

The coke consumed varies from 9 to 11 per cent., it comes from Westphalia and contains from 5 to 10 per cent. ash. The furnace puts through about 90 tons charge in 24 hours, and makes 7 per cent. flue dust, collected in a chamber 200 ft. long, and 16 tons matte.

Smelting at Broken Hill, N. S. W.—This subject, treated by Poole (*Mines and Minerals*, 1911, XXXV, 315) has recently been discussed in these reviews (MIN. IND., 1909, XVIII, 484).

Smelting at Webb City, Mo.—The plant of the Webb City Smelting and Manufacturing Co. (L. L. Wittich, *Mines and Minerals*, 1911, XXXI, 709; see also MIN. IND., 1910, XIX, 442) treats per week 250 tons galena concentrate with 80 per cent. Pb in six ore-hearths, filters the fume and smelts in a blast-furnace sintered fume and gray slag; the flue dust from the blast-furnace is treated as is the ore-hearth fume. The pig lead produced is remelted in a kettle holding 9900 lb., poled and molded into pigs weighing 90 lb. The six ore-hearths stand in a row in a room 28×80 ft. A hearth-box is 60 in. long by 18 in. wide and 7 in. deep. The blast is supplied by a No. 9 Buffalo blower, driven by a 20-h. p. direct-connected motor, to a main from which six branches lead to the several furnaces. A 100-in. Buffalo exhaust fan, driven by a 25-h. p. motor, draws off the gases into a main, forces them through 11 goose-neck cooling-pipes, arranged in two rows, into a distributing-pipe whence five branches lead into as many compartments of the bag-house. The goose-neck pipes stand upon brick chambers provided with cleaning-doors. The path the gases travel has a length of 1150 ft. The blast-furnace, 16 ft. high, stands in a building 28 by 40 ft., its gases are drawn off through a branch into the main fume-pipe. The bag-house, a frame structure 47 by 72 ft., has a concrete floor, a steel horizontal partition, and four vertical brick walls to furnish five bays. Each compartment has 144 woolen bags 18 in. diameter and 25 ft. long. The work is divided into four 6-hour shifts, each of which smelts 7000 lb. ore with 400 lb. lime. The loss in lead is estimated at 0.7 per cent.; of the

lead recovered, 65 per cent. comes from the ore-hearths, 34 per cent. from the blast-furnace.

The Ore-hearth.—The preceding paragraph gives the details of smelting at Webb City, Mo.

J. J. Brown (*Transactions* American Institute Min. Eng., June, 1911) describes the changes he introduced at works of the Granby Mining & Smelting Co., Granby, Mo. The smeltery has no blast-furnace department; it aims to produce a gray slag as low in lead as possible; the fume collected in bags is sintered in the usual way and then smelted with the ore. Until about 5 years ago the furnaces used were water-back ore-hearths 2 ft. wide, the fuel was charcoal, and a 14,000 lb. charge was treated in 14 hours; this period was divided into two shifts of four men working in pairs and relieving one another every 15 to 20 minutes. The aim was to convert into pig lead 70 per cent. of the galena, 50 per cent. of the white fume, and 40 per cent. of the blue fume. Of the lead charged, 85 per cent. was recovered as metal, 2 per cent. went into the slag. At present an air-back ore-hearth 5 ft. wide is in operation and bituminous coal is used as fuel; the furnaces work more quickly and give a higher extraction than the smaller hearths, although more fume is produced. The extraction required of the men has been changed to 65 per cent. of the lead in the galena charged, 60 per cent. of the white fume, and 50 per cent. of the blue fume. The cost of treatment per ton of material in the large furnaces is $2.4332, that in the small furnaces was $3.2992. The author believes that ore-hearth work can be made mechanical, and if mechanical the width of the furnace can be considerably increased; he has patented (U. S. Patent No. 888,582, May 26, 1908) a furnace of this character.

Zinc-Lead Pigment.—J. I. Blair (*Mines and Minerals*, 1910–11, XXXI, 698) publishes the drawings and gives a description of a plant of 72 tons capacity for producing zinc-lead pigment from mixed zinc-lead sulphides. The essential parts are, a series of 12 double Wetherill grates, 6 ft. 10 in. by 4 ft. 1 1/2 in.; an equalizing tower, 14 ft. diameter and 19 ft. high; a cooling tube, 5 ft. diameter and 200 ft. long; a 7-ft. suction fan; and a bag-house, 48 by 100 ft., with 640 woolen bags 30 ft. long and 20 in. diameter; next to the last is placed a packing shed, 40 by 100 ft. The fan is placed close to the bag-house, as experience at Collinsville, Ill., has shown that the mortar of the brick flue, situated between the fan and bag-house, became quickly corroded and caused the structure to sag and to fall.

Roasting Apparatus.—Guillet (*Revue de Métallurgie*, 1911, VIII, 561) discusses in a general way the roasting of metallic sulphides. As far as the treatment of lead ores is concerned, interest attaches to the plan and

vertical section of the pot-roasting plant at Laurium, Greece; and a plan and elevation of the von Schippenbach down-draft furnace (see below).

Efficiency of Roasting.—In a paper (*School of Mines Quarterly*, 1911–12, XXXIII., *Eng. Min. J.*, 1911, XCII, 1267) with this title A. S. Dwight reviews historically some of the leading apparatuses and furnaces, and compares them critically. Thus heaps and stalls, hand reverberatory furnaces and mechanical reverberatory furnaces with stationary and revolving hearths represent the ordinary modes of roasting, while up- and down-draft roasting apparatuses represent the great innovations that have been made in recent years in the roast-sintering of sulphide material. The roasting capacity of an apparatus depends upon the rapidity with which an ore can be freed from its sulphur in a given period of time. The best basis of comparison for the different roasting apparatuses is the pounds of ore treated per square foot of hearth area. The results of this comparison are given in the accompanying table. Some

EFFICIENCY OF ROASTING APPARATUS.

Apparatus.	Pounds ore treated in 24 hr. per sq. ft. hearth area.	Character of product for blast-furnace smelting.
I. Roast heaps and stalls	5–20	Good.
II. Reverberatory roasting furnaces:		
1. Hand-furnaces	24–35	Fair.
2. Mechanical furnaces:		
average conditions	33–75	Too fine.
special conditions	150	Too fine.
3. Revolving cylinders	128	Too fine.
III. Blast-roasting pots, range	500–900	Excellent.
Blast-roasting pots, average	600	Excellent.
IV. Blast-roasting, thin layers (Dwight-Lloyd system):		
1. Intermittent down-draft pans	1000–2000	Excellent.
2. Continuous sintering-machines	2200–3000	Excellent.

of the data in the table cover a considerable range, but this is due to the character of the ore treated and the degree of desulphurization desired.

Blast-roasting, the Behavior of Calcium and Barium Sulphate.—The researches of Hofman and Mostowitsch (MIN. IND., 1908, XVII, 591), and of Mostowitsch (MIN. IND., 1909, XVIII, 494) have shown that the decomposition of $CaSO_4$ by heat alone begins at 1180° C., that of $BaSO_4$ begins at 1400° C., also that in the presence of SiO_2 decomposition temperatures are lowered. In blast-roasting according to the Bradford-Carmichael process, using gypsum as flux, temperatures are not reached at which $CaSO_4$ is completely decomposed, nevertheless in the blast-roasted product of this process the $CaSO_4$ is decomposed. There remains the possible effect of the galena in the charge which may act according to $6CaSO_4 + 6SiO_2 + 2PbS = 6(CaO.SiO_2) + 8SO_2 + 2PbO - 293{,}000$ cal, or

$6CaSO_4+6SiO_2+3PbS=6(CaO.SiO_2)+9SO_2+3Pb-342{,}900$ cal., while the reaction $6CaSO_4+6SiO_2=6(CaO.SiO_2)+6SO_2+6O-495{,}000$ is much more endothermic than the two preceding. This question has been investigated by W. Schütz (*Metallurgie*, 1911, VIII, 228), who finds that the decomposition of $CaSO_4$ and $BaSO_4$ by SiO_2 in the presence of PbS begins below 800° C., and is practically completed with $CaSO_4$ at 1000–1100° C., and with $BaSO_4$ at 1100–1200° C. In order to accomplish the decomposition there must be supplied an extraneous source of heat, as the oxidation of the excess PbS in the charge is not sufficient for this purpose, hence FeS or FeS_2 has to be added to the charge. Further, part of the SiO_2 can be replaced by Fe_2O_3 which, as shown first by Hofman and Mostowitsch, and later by Kohlmeyer (MIN. IND., 1910, XIX, 438) forms fused ferrite at 1000–1100° C.; the replacement can be such as to make the blast-roasted ore self-fluxing in the subsequent smelting. Barium sulphate cannot well be substituted for calcium sulphate, because its decomposition, as shown by the experiments, requires a temperature at which PbS is volatilized to a considerable extent.

The von Schippenbach Down-Draft Roasting Furnace (Guillet, *Revue de Métallurgie*, 1911, VIII, 599; *Private Notes*, 1912).—This furnace, in operation at the Binsfeldhammer works near Stolberg, Rhenish-Prussia, recalls in its general features the Dwight-Lloyd horizontal table machine (Hofman, *Trans.* Am. Instit. Min. Eng., 1910, XLI, 761) which resembles a horizontal rotating picking-table in which the ring-shaped table is replaced by herring-bone grates. The leading differences are (1), that the withdrawal of roaster-gas is so regulated that rich gas with from 4 to 5 per cent. vol. SO_2 is continuously separated from poor gas with 0.2 per cent. vol. SO_2, and thus made available for utilization in the manufacture of sulphuric acid; (2) that the thickness of roasting-bed is 10 in. instead of the typical 4 in. of Dwight-Lloyd; (3) that the grate has a different construction; (4) that the ore is first rough-roasted as in the Huntington-Heberlein process. The furnace is 16.4 ft. diameter, and makes 1 revolution in 90 minutes. The ore treated is a mixture of 90 per cent. Australian and 10 per cent. German galena concentrate with Pb 60 to 70 per cent., S 13 to 14 per cent., Fe 3 per cent., SiO_2 3 to 4 per cent., Zn 5 to 6 per cent. The charge for rough-roasting is made up of 100 ore, 22 limestone and 5 silica; the rough-roasted charge retains from 9 to 12 per cent. S, of which 2 to 2.5 are sulphate ($CaSO_4$) and from 6 to 7 sulphide-sulphur; the blast-roasted charge contains total S 2.5 to 4.5 per cent., of which sulphate S is 2 to 3 per cent., and sulphide S 0.5 to 1.5 per cent.

The Dwight-Lloyd Straight-Line Roasting Machine.—At Trail, B. C. (Jacobs, *Met. Chem. Eng.*, 1911, IX, 405) two 100-ton machines, requir-

ing three men on a shift, have been put into operation to work in competition with the Huntington-Heberlein plant (MIN. IND., 1910, XIX, 450) comprising seven mechanical reverberatory roasting furnaces and 24 converting pots.

Blast-roasting has so far been confined to the treatment of sulphide materials; recently its sphere has been extended to the treatment of speiss (MIN. IND., 1910, XIX, 450). A new field of usefulness has been developed by Gayley (Trans. Am. Instit. Min. Eng., June, 1911; *Eng. Min. J.*, 1911, XCII, 50, 115) by the treatment of fine iron-bearing oxide material. Flue dust from the iron blast-furnace containing Fe 46, SiO_2 9, C 14–17 per cent., has been treated in a Dwight-Lloyd straight-line machine without any additions, while a magnetite concentrate with Fe 57 per cent. and SiO_2 9–10 per cent. required an addition of about 10 per cent. coal to furnish a hard porous clinker. At Birdsboro, Pa. (Gayley, *Iron Age*, 1912, LXXXIX, 73), the sintering process is in operation at the works of the E. & G. Brooke Iron Co. It has been found that 8 per cent. C in the charge is about the right amount of heating agent to form a strong cellular clinker. With sulphide ore the depth of bed is usualy 4 in.; with oxide it ranges from 5 to 7 in. The daily capacity of the machine with pallets 42 in. wide is 100 tons. The cost of installation of a 100-ton machine is $12,000. The paper gives examples of ore and flue dust mixtures treated at this plant.

Down-Draft Sintering Experiments.—A. W. Hahn (*Eng. Min. J.*, 1911, XCL, 1208) describes a small apparatus for making experiments in down-draft sintering which in its essential parts consists of a vertical sheet-iron cylinder, 14 in. diameter and 10.5 in. high, closed at the bottom by a 4-mesh screen and at the top air-tight by a cast-iron plate with a 1-in. blast-inlet pipe. On the screen is placed a 1/2-in. layer of limestone, crushed to 3/8 in., on this the charge, and on the charge burning charcoal. The blast is evenly distributed by suspending below the outlet of the pipe a screen 4 in. square. The apparatus is placed on a support, *e.g.*, an empty slag-pot. The materials with which experiments were carried out are given in the subjoined table:

MATERIALS FOR SINTERING EXPERIMENTS.

Material.	SiO_2 %	Fe %	Pb %	Cu %	As %	S %	Zn %	C %	Ag, oz. per ton.	Au, oz. per ton.
Pyrite	8.2	39.3	1.2	1.2	2.7	43.2	3.7	....	6	0.2
Sandy ore	30.1	22.6	12.6	trace.	2.6	4.0	1.7	....	40	1.6
Flue dust	28.6	14.3	24.0	.....	2.7	4.8	....	9.0	30	1.0
Matte	.....	45.0	14.0	0.4	5.8	23.0	....	....	65	0.3
Speiss	.....	64.7	3.0	.....	18.2	8.7	....	....	27	1.7

There were tried 28 mixtures; the results of the best four are given below:

RESULTS OF SINTERING EXPERIMENTS.

Sintered, per cent.	Mixture, parts.			Time of Blowing, min.	Depth of Bed, in.	Analysis or Product.						
	Pyrite.	Ore.	Flue dust.			SiO_2 %	FeO %	As %	S %	Pb %	Ag, oz. per ton.	Au, oz. per ton.
100	4	3	3	30	3	18.2	42.4	0.8	7.5	15.9	45	1.5
100	3	3	2[1]	42	3	17.6	38.4	....	6.9	15.6	52	1.3
95	2.5	3	1.5[2]	49	3	15.6	43.1	....	6.0	17.6	55	1.2
100	3	3	2[3]	60	3	10.0	46.8	....	4.7	13.2	35	1.5

[1] Two parts matte added. [2] Three parts matte added. [3] Two parts speiss added.

Temperatures of Some Metallurgical Reactions.—Friedrich (*Stahl u. Eisen.*, 1911, XXXI., 2040) studied the temperatures at which some metallic compounds react with metals and other agents. Thus, Fe_2O_3+C act upon one another at 900° C.; $PbSO_4$ undergoes a transformation at 850° C., and is decomposed by SiO_2 at 1030° C.; the reaction $CuO+Fe$ takes place at 1150° C., and that of $PbO+Fe$ at about 925° C. Other reliable observers give much lower temperatures for some of these reactions; this will lead the reader to think that the method followed in the experiments is one-sided.

New Blast-furnace.—J. W. Nesmith patented (U. S. Patent No. 995,365 and 995,366, June 13, 1911) a combination of shaft and reverberatory furnace for treating simultaneously coarse and fine ore. The shaft-furnace discharges its contents into the reverberatory furnace and acts at the same time as chimney for the reverberatory furnace; the latter is oil-fired, and fine ore is fed into it by means of blast.

Cooling Towers.—E. Jawger (*Min. and Sci. Press*, 1911, LXIV, 480–495) discusses the subject of cooling towers for the Intermountain States from the point of view of an electrical engineer. He shows that for the effective cooling of water there are necessary; speed at the surface of contact of air and water, power to produce this speed of contact, time for exposure, and fine state of division of water. Of the many types of water coolers in common use, two only are found in western smelting centers, the open type with horizontal slats and the chimney type with either horizontal or inclined slats. The former is probably cheaper in construction and power equipment. Two examples are described and illustrated in the original paper.

Fuel Consumption.—An extended thermal study, based on the coke consumption of lead and copper blast-furnaces, is published by G. Ralli (*Rev. Universelle des Mines*, 1911, XXIV, 213, XXV, 1) to which the reader is referred.

Blowing-in.—L. B. Harrison (*Eng. Min. J.*, 1911, XCI, 806) discusses the important operation of blowing in a lead-blast furnace, important

because the success of a campaign depends largely upon a correct start. He reviews briefly the usual method of warming and heating the crucible, of filling it with bars which are melted down, and then gives the blowing-in charges, etc. The disadvantages of the method are; expense of labor and fuel, waste of time, volatilization of lead, formation of crust due to the wood placed on the liquid lead becoming immersed in it and charred instead of being burnt. The so-called platform method receives some consideration. In it a platform of boards, 1.5×3 in., is built over the crucible leaving open spaces 1.5 in. wide; on it are placed the kindling,

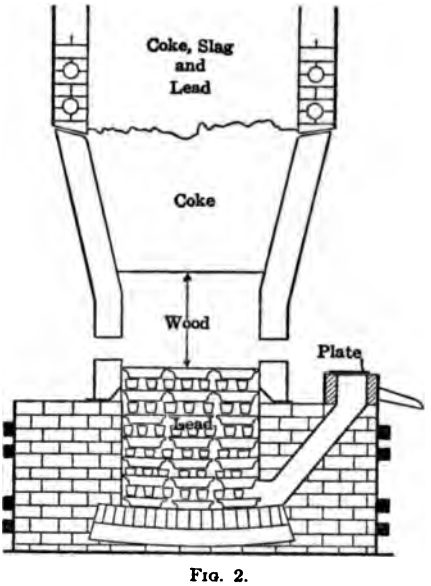

FIG. 2.

then a bed of coke from 3 to 5 ft. thick to be followed by lead bullion, coke and blowing-in charges. When the furnace has been filled, the fire is started at the tuyères, and the blast turned on. The supposition is that the lead will have been melted and will have filled the crucible before the wood has been burned up and the blowing-in-slag commenced to flow. The method is condemned by the author. The last method described is the so-called Mapimi method. In it pig lead is piled in the dried crucible, as shown in Fig. 2 split wood with oil waste is placed in front of the tuyères, and more kindling charged from the top; this is followed by a bed of coke, and blowing-in charges with the lead necessary to fill the crucible with molten lead; further, some matte is added to the last blowing-in charges, that the matte melting after the slag may cut or wash out impurities above the bath of lead. The fire is started

through the tuyères, and a gentle blast turned on to be increased later on. Two hours after firing, slag will be seen at the tuyères, and the crucible will have been filled completely with clean lead which becomes red-hot in about 8 hours. The plate closing the lead-well at the start may be removed in about 1.5 hours after starting the blast. A furnace 46×162 in. at tuyère-level requires 550 bars at 50 kgrm. in pig form to fill the crucible, and 100 additional bars in the blowing-in charges; 17 barrows coke at 87.5 kgrm. furnish the coke-bed. The blowing-in charges at Mapimi are as follows: 18 barrows coke; 8 barrows coke, 4 buggies slag at 830 kgrm., 100 bars lead; 8 coke, 4 slag, four times; 8 coke, 8 slag; 1 barrow matte; 6 barrows coke, 4 buggies slag, 2 ore-charges, three times; regular ore-charge with 15 per cent. fuel.

Manganous Oxide-Silica Series.—Doerinkel (*Metallurgie*, 1910, VIII, 201) has investigated the manganous silicates ranging from 20 to 60 molecular per cent. (17.8 to 56.1 weight per cent.) SiO_2. His equilibrium diagram shows the subsilicate Mn_2SiO_4 with 33.3 molecular per cent. (29.8 weight per cent.) SiO_2 (tephroite) solidifying at 1323° C., the bisilicate $MnSiO_3$ with 50 molecular per cent. (45.9 weight per cent.) SiO_2 (rhodonite) freezing at1230° C., and an eutectic between the two with 45 molecular per cent. (41 weight per cent.) SiO_2 freezing at 1190° C.

Lead Content of Blast-furnace Charges.—L. S. Austin (*Min. and Scientific Press*, 1911, CII, 177) states that at the lead-smelting works of Trail, B. C., and East Helena, Mont., the blast-furnace charges contain 35 per cent. lead and over, a figure which in former years would have been thought to be next to impossible with the production of a slag low enough in lead to be discarded.

Metallurgical Products.—Von Komorowski (*Metallurgie*, 1911, VIII, 741) has examined the freezing-points and constitutions of a number of lead products obtained in smelting and desilverizing argentiferous lead ores. The results, which have no general application, may serve as guides in investigations of similar materials.

Silver in Matte.—The equilibrium diagram of Ag_2S–FeS drawn by Schoen (*Metallurgie*, 1911, VIII, 737) offers special interest as regards the presence of Ag_2S in matte. It shows that Ag_2S and FeS form a eutectic mixture with the eutectic point lying at 600° C., the composition being 11 per cent. FeS and 89 per cent. Ag_2S. The two sulphides are slightly soluble in one another so that within the limits of mattes produced in smelting, Ag_2S and FeS form solid solutions.

Concentration of Matte.—A series of laboratory experiments has been carried out by R. Hesse (*Metallurgie*, 1911, VIII, 321, 365) to find a quick method for producing metallic copper from concentrated lead-

copper matte. The raw material used was a matte produced at the Altenau, Harz Mts., smelting works with ore from Otavi, Southwest Africa, which contained Ag 0.04, Cu 40.81, Pb 23.65, Fe 10.45, Mn 0.20, Zn 1.88, S 17.99, As 0.20, Sb 0.10, SiO_2 2.57, CaO 1.17, Al_2O_3 0.84, O (difference) 0.10 per cent. The first plan tried, the application of a roast-reacting procedure, was a failure. Roasting the matte, adding SiO_2 and smelting in an electric furnace gave black copper (Cu 84.52, Pb 12.91, Fe 0.01, S 0.72, insoluble 0.28 per cent), copper matte (Cu 74.93, Pb 4.74, Fe 0.57, S 18.39 per cent.), and slag (SiO_2 39.17, PbO 28.81, FeO 25.22, CuO 3.47, MnO 1.45, MgO 0.47, Al_2O_3 1.46 per cent.). The method advocated, crushing the matte in a ball mill to 2 mm., roasting in a mechanical reverberatory furnace down to 4 per cent. S, and smelting in an electric furnace with an addition of SiO_2, resembles very much the one commonly employed in this country excepting that the reverberatory furnace is in use for smelting; and this has the advantage of producing a copper running low in lead. In recent years (MIN. IND., 1910, XIX, 454) converting such a matte with the recovery of fumes has begun to replace all older modes of treatment.

Converting Matte.—The following table of analyses of converter fume recovered at the smelting of Agua Calientes in treating lead-bearing copper matte in upright converters (diameter 8 ft., height from tuyères to beginning of hood 7 ft. 4 in., blast pressure 12 to 16, sometimes 18 lb. per square inch) has been published by C. C. Semple (*Eng. Min. Jl.*, 1911, XCI, 508.)

The converter hoods ended in an oval steel flue which ran at right angles to the direction of the hoods and was provided with hoppers for drawing off settled dust. The flue led the gases to an exhaust fan placed in a brick chamber whence they were forced successively through four

ANALYSES OF CONVERTER FUME.

Source of Material.	Weight kgr.	Ag. Kgr. p. met. ton.	Au. Kgr. p. met. ton.	Pb %	Cu %	As %	SiO_2 %	Fe %	Zn %	S %
At throat of converter ...	.....	4.328	0.0176	16.8	34.0	.....	.....	.....	.....	.....
Converter flue dust.....	6,348	3.564	0.016	25.2	30.7	.....	0.4	0.5	7.3	12.1
Oval pipe hopper.....	698	2.962	0.0112	30.0	24.9	.....	0.6	trace	8.0	11.1
Fan-house section 1.....	1,209	1.993	0.006	37.0	16.2	.....	0.6	0.3	9.3	10.7
Fan-house, section 2.....	2,358	2.690	0.011	29.5	20.7	.....	0.7	0.3	8.8	11.4
Fan-house, section 3.....	13,346	0.552	0.001	52.5	0.75	.....	0.4	.....	14.4	8.4
Fan-house, section 4.....	5,585	0.435	trace	57.0	0.50	.....	0.4	trace	11.5	8.3
Curtain chamber.....	12,871	0.495	trace	54.8	0.90	.....	0.5	trace	11.1	8.7
Condenser.....	32,784	0.455	trace	53.5	0.15	3.6	0.4	.....	10.4	8.9
Bag-house 1.....	16,782	0.0405	trace	58.0	0.10	trace	0.3	.....	9.8	8.7
Bag-house header	1,198	0.505	trace	58.5	0.10	trace	0.2	.....	9.4	8.8

brick sections, a curtained chamber and four steel cooling flues, into the bag-houses connected by a passage to the stack. The data show that

the fumes are rich in lead, and that the loss in precious metal is proportional to that of copper, and independent of that of lead. This is contrary to the opinion formerly held, but in accordance with other recent investigations upon the subject.

Speiss.—The paper on blast-roasting speiss by Guillemain abstracted in last year's review (MIN. IND., 1910, XIX, 450) has been translated from the German by O. H. Hahn (*Eng. Min. Jl.*, 1911, XCI, 858).

Smelting Losses.—In a dissertation, L. Schertel (Studien über einige Verlustquellen des Blei-und Kupfer-Hochofen Processes, Freiberg, 1911) discusses some sources of loss in the treatment of lead and copper ores in the blast-furnace. The sources considered are solubility of matte in slag, the separation of matte from slag, and the volatility of lead in matte. The conclusions arrived at are: (1) that PbS and Cu_2S are insoluble in the slags tested, but that FeS is probably soluble to some extent; parenthetically it may be noted that W. Wanjukoff (*Metallurgie*, 1912, IX, 1) arrives at the opposite conclusion as regards Cu_2S; (2) that the copper content can be reduced only to from 0.10 to 0.15 by settling, but that by stirring, suspended matte particles can be forced to form pellets which separate, whereby the copper content can be reduced to 0.04 per cent.; (3) that lead in matte is volatile at 1200° C., and that the volatilization increases rapidly with the temperature; (4) that slag is insoluble in matte. While matte analyses usually show the presence of some slag, this can be separated by raising the temperature of matte to say 1500° C.

New Process.—H. Hof and B. Rinck patented (German patent No. 227,389) a process for the extraction of PbO and $PbSO_4$ from lead drosses. Lead oxide is readily soluble in a hot concentrated solution of $MgCl_2$ forming the salt $2MgCl_2PbCl_2$, while the H_2MgO_2 formed combines with the excess $MgCl_2$ to an oxychloride. The dissolved $2MgCl_2.PbCl_2$ readily separates upon cooling the solvent. The $PbSO_4$ is similarly dissolved in boiling $MgCl_2$ forming the same double chloride as the $PbCl_2$, the resulting $MgSO_4$ falls out in part with the double chloride upon cooling, and is later removed by treatment with warm water.

Condensation.—C. Guillemain has published a small book entitled, "Theorie und Praxtis der Staubverdichtung und der Reinigung und Entstaubung von Gasen," Knapp, Halle a/S., 1911, pp. 54, 8°, $0.70. This book which is to treat of the theory and practice of condensation, the purification of gases, and the separation of suspended dust, has a few general considerations of the subject as a whole, which are followed by a compilation of the German patents granted. It may be of use to anybody wishing to take out a patent in Germany; the metallurgist will put it aside for possible reference.

In the dry condensation of flue dust (Rzehulka, *Zeitschrift für angew. Chemie*, 1911, XXIV, 1803) the two leading steps taken are the cooling of the gas and the retardation of its velocity. As the mutual attraction of particles decreases with the square of distance, cooling, which diminishes the volume of gas, will bring the particles more closely together, and increase the mutual attraction, with the result that they form flakes which settle readily; further, the decreased volume of gas will have a smaller velocity, and this again favors settling. No decided improvements have been made in the devices for settling dust, excepting perhaps that Freudenberg plates have been staggered instead of being hung in line one back of the other. The Roesing wires at Great Falls, Mont., appear to be giving much satisfaction.

A review of flue gas treatment (Editor, *Eng. Min. Jl.*, 1911, XCII, 1116) discusses briefly the different steps that have been taken for the recovery of dust and fume produced in lead and copper smelteries. Long flues are characterized by the dismantled plant of the Omaha and Grant Works, Denver, Colo., which with over half a mile of flues delivered lead fume into the air through its 300-ft. stack. The Roesing wire-system serves as an efficient example of what can be accomplished with flues for settling fume, in this case of vapors of As_2O_3. In the settlement of dust, the experiments of the Copper Queen smeltery appear to have shown that, if the velocity of the gas current drops to 2.5 ft. per second, an enlarged chamber is all that is necessary to recover practically the whole of it. A water-spray, *e.g.*, of the Koerting nozzle type, is efficient, if the gases are sufficiently hot to vaporize the water which, cooled further on, becomes a mist and thus wets every particle of fume. A recent discovery is that beside sulphuric acid, selenium compounds have a destructive effect upon filter bags, while tellurium appears to be only slightly active. Centrifugal scrubbers have not so far made much headway in sulphur-carrying furnace gases.

Sprague Process.—C. B. Sprague has taken out additional patents (U. S. Patents No. 984,498, Feb. 11, 1911; No. 992,391, May 16, 1911) for his well-known process (MIN. IND., 1908, XVII, 604; 1909, XVIII, 495; 1910, XIX, 458) for neutralizing the sulphuric acid in fumes by means of ZnO and CaO. The process is in operation at the Midvale and Mammoth (*Eng. Min. Jl.*, 1911, XCI, 615) smelteries of the U. S. Smelting, Refining & Mining Co.

F. W. Bocking (*Eng. Min. Jl.*, 1911, XCI, 895) calls attention to the similarity of the effect of ZnO in the Bartlett process at Canyon City, Colo. (Hofman, "Lead," 1899, pp. 138–148) in preventing the corrosion of the bags in the bag-house.

Halberge Filtering Plant.—Müller (*Stahl u. Eisen*, 1911, XXXI, 229)

describes the working of the experimental filtering plant for 30,000 cu. ft. gas per minute at the iron blast-furnace works of Halberge, near Saarbrücken, Germany, the self-cleaning device of which has general interest. The gas enters the apparatus with a temperature ranging from 80 to 120° C. and charged with 6 gr. dust per cubic meter, it leaves it with 60° C. and 0.01–0.30 gr. per cubic meter. The apparatus is shown in Fig. 3. The gas arriving through A enters the water-

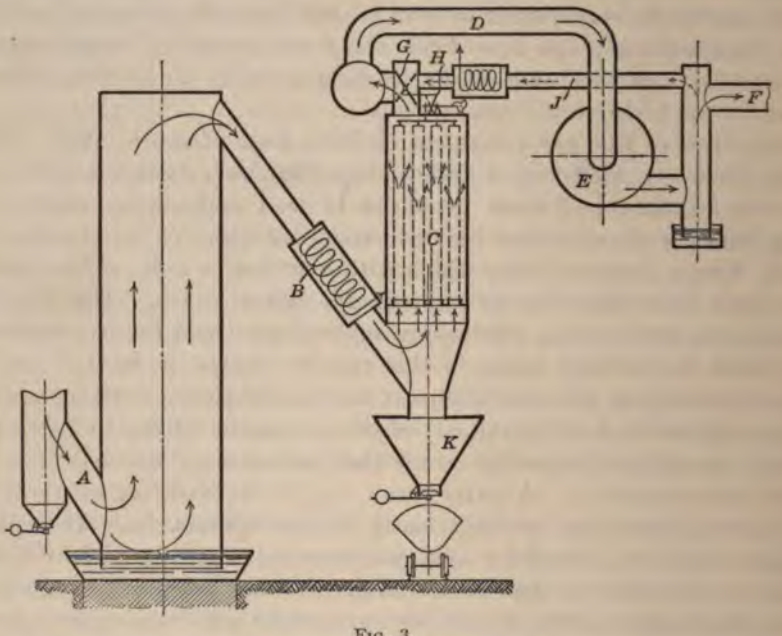

FIG. 3.

sealed settler in which it is cooled to the dewpoint in order that the dust particles may become wetted. Upon leaving, it enters the chamber B, warmed by waste steam or waste gas, in order to have its temperature raised to 10 or 20° C. above the dewpoint before it passes into the bag-room C, containing from 8 to 12 bags, 10 ft. long and 8 in. diameter. The filtered gas passes the turn-stile damper G into pipe D and through fan E, which delivers it into the outlet pipe F. After traveling in this direction for about 4 minutes, the damper G is turned mechanically to occupy the second position, shown by the dotted line, when the back pressure from the outlet pipe F forces cleaned gas through pipe J with its heated chamber, and thence through the outer surface of the bags; the dust gathered on the inside is loosened, the bags are shaken by the mechanism H, and the dust drops into the hopper K to

be discharged at intervals. This cleaning takes from 15 to 20 seconds. By means of the device a small filtering surface can handle a large volume of dust-laden gas, and recover the dust automatically.

Cottrell Electric Condensing Process.—A radical departure from the usual methods of condensation is the Electric Precipitation Process of F. G. Cottrell (*Jl. Ind. Eng. Chem.*, 1911, III, 542; *Eng. Min. Jl.*, 1911, XCII, 763; *Min. Scientific Press*, 1911, CIII, 255, 286; *Min. Science*, 1911, LXIII, 337, LXIV, 350, 364; *Eng. News*, 1911, LXVI, 495) in operation at the Selby Lead Works near San Francisco, and the Balaklala copper smeltery, Coram, Cal. Experiments by Lodge and Walker have shown that an alternating current will cause the agglomeration of particles suspended in a fairly quiescent atmosphere. In a moving gas current agglomerating and settling are too slow to be effective. Cottrell therefore uses a high-potential direct current. If one pole of such a current is connected with a needle-point, and the other with a flat plate, the air space between the two becomes highly charged with electricity of the same sign as the needle-point, and any insulated body brought into this space receives a similar charge. If this body is free to move, as is the case with suspended flue dust, it will be attracted to the plate of the opposite charge; it will therefore move toward the plate, and do this at a speed which is proportional to its charge and to the potential gradient between the point and the plate. Observation has shown that in a perfectly transparent gas there is a strong wind from the point to the plate.

In order to obtain a high-potential direct current, an alternating current from a lighting or power circuit of from 20,000 to 30,000 volts is commuted by means of a special rotary contact-maker driven by a synchronous motor. For the needle-point of the laboratory experiment there was substituted at first cotton-covered wire, the loose fiber of which furnished innumerable fine sharp points for the discharge of the high-potential current since it was sufficiently conductive from its natural hygroscopic water. The cotton, readily attacked by hot acid gas, was replaced by asbestos or mica, the fibers or the edges of the scales furnishing the desired discharge points.

At the Selby works the first unit was introduced to condense in a gas current of 5000 cu. ft. per minute, the fumes resulting from the sulphuric acid parting plant. In a lead flue, 4 by 4 ft. there were suspended 4 in. apart, rows of vertical grounded lead plates, 4 ft. long and 4 in. wide, alternating with lead-covered iron rods carrying mica discharge material. A current of 460 volts and 60 cycles was transformed up to 17,000 volts and then passed through a rectifier to the electrodes. The power consumption was found to be only 2 kw.; the result obtained, com-

plete elimination and collection of the sulphuric acid vapor in the gases as H_2SO_4 of 40° Bé. The plant has been in operation for 3 years; the monthly outlay is $20; the acid recovered in 3 years has paid five times for the cost of the installation.

The second unit at the same works was intended to handle per minute the 50,000 cu. ft. gas coming from the roasting and blast-furnace departments. It was thought at first that the suspended material could be precipitated in a reasonably dry form which would permit brick and iron construction. Experience showed that this was not the case, and that lead construction was necessary. A flue, 6 by 6 ft. and 32 ft. long, containing 38 rows of 16 lead plates, each 6 ft. long and 4 in. wide, alternating with discharge electrodes was sufficient to collect the H_2SO_4, As, Sb, Pb in the gases as long as these had been cooled to below 150° C. The collected material was a grayish mud easily washed from the plates and run into settlers. For this purpose, the gases were turned into a by-pass every 4 or 6 hours. In a new plant there would be 2 or 3 units so that one could be cut out at intervals and cleaned. The power consumption was from 10 to 15 kw. The permanent installation has not yet been made.

Dissipation of Smoke.—An interesting development in the disposal of smelter smoke is the Wislicenus Smoke Stack (Austin, *Min. Meth.* 1911, II, 269; Hahn, *Eng. Min. Jl.*, 1911, XCII, 596). The sulphurous smoke issuing from the stacks of lead and copper smelteries remains undispersed for a long distance, the dust falls out gradually, but the gases come down not sufficiently diluted to make them harmless for vegetation. In order to dilute the gas, it is necessary that air be introduced at an early stage in such a manner as to divide the smoke column into strips surrounded by air. This is done by the Wislicenus stack, shown in Fig. 4, which consists briefly in placing a lattice-like extension on top of the ordinary chimney. Air will pass through the lattices from whatever direction the wind may be blowing, it will come in contact with the ascending gases, divide them into bands and pass out with them at the opposite side. Thus, instead of a single column of smoke rising from the stack, strips, each surrounded by air, will pass off at the sides, the column having been broken up at the point of issue. The smoke of the single bands will readily diffuse into the enclosing air, and the smoke be dissipated so that, when it finally comes down, the sulphurous gas will be diluted sufficiently to be harmless.

Effect of Sulphurous Acid.—The effect of sulphurous gas upon vegetation continues to occupy investigators. Thus, Sorauer (Sammlung von Abhandlungen über Abgase und Rauchschädeen, Heft. 7) finds that the chemical analysis for SO_3 and microscopical examination of vegeta-

tion have to supplement one another in determining accurately the effects of sulphurous gas. In considering the harm done, a distinction has to be made between the clearly visible large volumes of gas acting periodically, and the slow but steady influence of gases containing only little SO_2. The question whether injury to a plant results from gas or from infected soil can be settled only by actual experiment. This

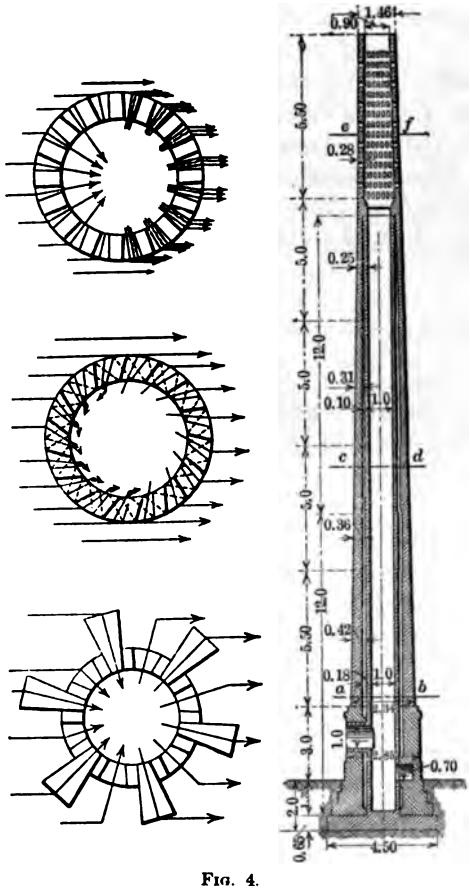

FIG. 4.

consists in placing in suspected ground a box of 1 cu. yd. capacity filled with soil from a sulphur-free section, and doing the same with sulphur-free soil in suspected ground, and planting in both string or pole beans, (Phaseolus vulgaris). If the plants in the first show signs of decay, the sulphur in the air is the cause, in the second the soil is poisoned.

Recovery of Sulphur from Smoke.—The reduction of SO_2 to S with

recovery of the latter has been repeatedly suggested. S. W. Young (*Min. and Scientific Press*, 1911, CIII, 375, 386) has taken up the matter and developed what he calls the Thiogen Process. This consists of the following steps (1) Action of dry SO_2 upon warmed CaS as expressed by $2CaS+3SO_2=2CaSO_2+3S$; (2) recovery of the S from the intermediary product by sublimation, and treatment of $CaSO_3$ with CxHy at an elevated temperature, when $2CaSO_3+2CH_2=2CaS+2CO_2+2H_2O$ takes place with the regeneration of the original reagent. It appears that the presence of pyrite is favorable to the process, as it is decomposed according to $3FeS_2+2SO_2=Fe_3O_4+8S$, although very slowly. Instead of the imaginary CH_2 given in the formula, there would be used hydrocarbons such as C_2H_4, CH_4, petroleum vapor etc. In this connection it may be noted that in 1894 Kosmann (*Glückauf.*, May 2, 1894) experimented with the action of H_2SO_3 upon a solution of CaS in water, when $CaSO_3$, S, and H_2O were formed, but the process was not successful.

Desilverization

Cupellation.—Examples of cupellations of lead bullion carried on at the works of the Kendall Gold Mining Co., Kendall, Mont., for the recovery of precious metal in zinc precipitate collected in lead by smelting with litharge, are described by R. F. Coolidge (*Eng. Min. Jl.*, 1911, XCII, 756). The test was elliptical, 24 in. wide by 30 in. long by 4 in. deep; the filling a mixture of 70 per cent. portland cement and 30 per cent. limestone; the cavity 3.25 in. deep at the back and 3.5 in. at the front; the rim at the back and two sides 3 in. wide, at the front 4 in. A cupel carriage with four screws supported the test; the blast was supplied by a No. 4 (24 in.) blower making 2400 r. p. m., a 2-in. pipe flattened to 1×2 in. at the nozzle served as blast-pipe; gasoline at 40-lb. pressure served as fuel; there were four 2.5-in. Carey burners. Details of five cupellations are given in the accompanying table.

CUPELLATION TABLE.

Lead cupelled, lb	736	597	398	391	496
Began charging lead, a. m	6.30	8.00	9.15	8.45	8.30
Lead bath cleared, a. m	7.30	8.45	10.00	9.45	9.45
Finished charging, p. m	6.45	7.00	3.00	2.00	4.30
Finished cupelling, p. m	11.00	10.45	7.00	5.45	6.00
Litharge recovered, lb	660	575	300	304	309
Litharge, Au value per ton, $	320	62	170	225	200
Cupel Bottom, soaked, lb	135	100	120	85	120
Cupel Bottom, Au value per ton, $	500	250	500	400	680
Doré Silver, weight, oz. troy	227.15	173.91	273.05	456.93	352.08
Doré Silver, fineness, Au	725	690	765	821	723
Doré Silver, fineness, Ag	62	71	67	75	56

Pattinson Process.—G. P. Hulst has taken out an additional patent (U. S. Patent No. 1001525, Aug. 22. 1911) for his hydraulic press to free the crystals in the Stearn-Pattinson process from adhering liquid lead (MIN. IND., 1910, XIX, 459). By this procedure the usual 12 crystallizations necessary for desilverizations have been reduced to six.

Parkes Process.—In the desilverization of lead bullion by means of zinc, it was at first assumed that the crust taken from the kettle was a zinc-silver-lead alloy contaminated with lead bullion which could not be completely separated by liquation on account of a partial oxidation of some lead and zinc. The modification of the usual mode of operating by Edelmann and Roessler (Hofman "Lead," 1899, p. 450) in which 0.5 per cent. Al was added to the zinc to prevent oxidation showed that one had to deal with a Zn–Ag and not a Zn—Ag—Pb alloy. Since then the studies of Petrenko (*Zt. anorg. Chemie*, 1906, XLVIII, 347) have proved the existence of four chemical compounds of zinc and silver, viz.: Ag_2Zn_5, with 60 per cent. Zn, Ag_2Zn_3 with 47.60 per cent, Zn, AgZn with 37.7 per cent., Zn, and Ag_3Zn_2 with 28.18 per cent. Zn. The latest research is that of Kremann and Hofmeier (*Monatshefte für Chemie*, 1911, XXXII, 337) which deals with the ternary alloys of zinc, silver and lead, and bears directly upon the Parkes process. The author investigated the following five ternary systems: Pb—Zn—Ag_2Zn_5; Pb—Ag_2Zn_5—Ag_2Zn_3; Pb—Ag_2Zn_3—AgZn; Pb—AgZn—Ag_3Z_2; Pb—Ag_3Zn_2—Ag. Only the system of lowest silver-content, Pb—Zn—Ag_2Zn_5, is of importance in the Parkes process. In the binary system Pb—Zn, the two metals are known to be slightly soluble in one another in the solid state, a eutectic point lies between 100 and 98.8 per cent Pb at 310° C. The system Zn—Ag_2Zn_5 forms a series of solid solutions, reaching from the melting point of Ag_2Zn_5, 638° C., to that of Zn, 419° C. For the third system Pb—Ag_2Zn_5, no data are available. The investigation shows that at 320° C, there is a eutectic point with 98.5 per cent. Pb and that there is no mutual solubility in the solid state.

The desilverization of lead bullion by zinc therefore is accomplished by the formation of the compound Ag_2Zn_5 which forms solid solutions with Zn, but not with Pb; no ternary alloy of Zn—Ag—Pb is formed in the process, the lead present is a mechanical admixture.

Electrolytic Refining.—Betts Process.—D. M. Liddell (MIN. IND., 1910, XIX, 459) gives a general review of the subject, the main point of which is that the work of the three plants operating the Betts process differs greatly as regards details, while all, of course, follow the same general outline.

Mathers-Siemens Process.—F. C. Mathers (*Trans.* Am. Electrochem. Soc., 1911, XVII, 261; *Min. World*, 1911, XXXIV, 309) publishes the

results he obtained in the electrolytic refining of lead bullion with a solution of lead perchlorate 2, Pb $(ClO_4)_2$. $3H_2O$, as electrolyte. The salt is prepared by the action of $HClO_4$ upon litharge; it is readily soluble in H_2O(1:1), forms no basic salts, is very stable, not being decomposed by boiling acids or alkalies, but readily so by fusion with Na_2CO_3 or $NaNO_3$. Solutions of $HClO_4$ have an electric conductivity similar to that of HCl. Additions of small quantities of glue, tannic acid, licorice juice, and especially of peptone, greatly assist in the formation of smooth cathode deposits. In three laboratory experiments with a solution of 800 cc. containing 6.1 per cent. $HClO_4$, 7.18 per cent. Pb and 0.05 per cent. peptone the following readings were obtained:

ELECTROLYSIS OF LEAD.

Amp. per sq. dcm.		Volt.
Anode.	Cathode.	
1.0	0.21	0.08
2.2	0.45	0.15
2.6	0.55	0.18
3.8	0.80	0.25
4.2	0.90	0.26
5.0	1.60	0.36
6.2	2.00	0.34

A similar process has been patented by Siemens & Halske (*Eng. Min. Jl.*, 1911, XCI, 902; German Patent, 223668, K. 40 c. g. 12, Aug. 24, 1909, June 27, 1910), the experimental work having been carried out by J. Nussbaum. Mathers (*Eng. Min. Jl.*, 1911, CXI, 1091) claims priority of invention (U. S. Patent, No. 931944, 1911).

Anode-Mud.—E. F. Kern (*Met. Chem. Eng.*, 1911, IX, 417) reviews briefly the different methods that have been proposed or that are in use for the treatment of the anode mud produced in the electrolytic refining of copper and lead. He gives the following data as covering the range of composition of anode mud from the electrolytic refining of lead bullion by the Betts process: Pb 6 to 18, Cu 5 to 25, Sb 5 to 70, As 2 to 50, Bi 0 to 20, Sn 0 to 2, Se 0 to 3, Te 0 to 4, Ag 10 to 40, Au 0 to 1 per cent. Two typical analyses from Trail, B. C., show: Ag 32, Au 0 to 3, Cu 9, Sb 27, As 7, Pb 14, H_2O, etc., 11 per cent.; and Ag 19, Au 0 to 1, Cu 11, Sb 31, As 9, Pb 16, H_2O etc. 14. The methods of treatment discussed are: Direct cupellation, fusion in basic-lined reveberatory furnace, treatment with sulphuric acid and air or niter, and the Cabell-Whitehead process which uses Ag_2SO_4 to dissolve the Cu from the mud. The reader is referred to the original which represents the lecture-notes for the students of Columbia School of Mines.

LITHIA.

No lithium mineral was produced in the United States in 1911. The minerals mined for lithium are spodunmene, lithiophilite, and amblygonite. The only deposits that have been worked commercially for these minerals are found in Costa County, South Dakota, in the Black Hills district. They also occur in connection with pegmatite. Another mineral, lepidolite, occurs at Pala Alta, California. The industry has been on the decline for a number of years and would, no doubt, be stimulated by new uses found for lithium compounds. The companies in a position to furnish lithium compounds in the United States are the Western Reduction Company of Keystone, S. Dak., the Standard Essence Company, Maywood, N. J., and the Pahasa Mining Company, 16 miles east of Hill City, S. Dak. All these companies own similar deposits.

A new mineral containing lithium was found by Waldemar T. Schaller, about 4 miles northwest of Canyon City, Colorado. On account of the sodium present in this mineral, it is easily distinguished from the amblygonite which gives the red flame test. The amount of soda in this mineral is about 11 per cent.; the lithia is 3.21 per cent.

In the Coolgardie district, Australia, amblygonite (phosphate of alumina and lithia) was discovered in some of the tin veins. An analysis was: alumina, 36.88 ; potash, .43; sodium, 3.29; lithia, 6.68; fluorine 8.11; and phosphoric acid, 47.58 per cent.[1]

In France, which is the principal source of amblygonite, it is produced at Montrobas. The mineral also occurs in the granite of Chursdorf, Saxony; at Arendal, Norway; and at Hebron, Maine. Lithia is used for the manufacture of lithia water and medicinal purposes. A small quantity also goes into the manufacture of fire works. The consumption in the United States is very small. Lithium has ceased to be prescribed as it was formerly, by the medical profession in cases of uric acid disease. This came about by the experiments of Dr. Pfeiffer at Wiesbaden who found deleterious effects on his patients, whereas previously in England, healthy subjects had been benefited as far as the elimination of uric acid was concerned.

[1] *Aust. Min. Standard*, April 20, 1910.

MAGNESITE

In the United States the only deposits of magnesite worked commercially in 1911 were in California. A detailed description of these deposits is found in *Bull.* 355 of the United States Geological Survey. The number of mines and quarries operated in California was less than six. The production of these same mines in 1910 was reported as 8000 short tons of crude ore and it is probable that the production in 1911 does not exceed 6000 tons although it is stated that the markets were good. Most of this material is not sold in the crude state but an approximate value at the mines is given as $8 per ton. The material is sold ground and calcined and generally a ready sale was found in 1911 at $35 per ton. It is also stated that some of the mines in California were temporarily shut down in 1911. The price during the year 1910 was lowered owing to competition and a price war, but in 1911 a low-grade market was interfered with only by the lack of transportation. Most of the calcined material is consumed by manufacturers of paper from wood pulp in California and Oregon. A small quantity was used for artificial stone tiling or building material. The front of one building in San Francisco is made of magnesite blocks molded into ornamental stone. One difficulty with the California magnesite thus far produced is that it does not have the proper binding qualities for manufacturing of refractory bricks.

Experiments have been made for the manufacture of magnesite brick but they have proved unsuccessful. The magnesite, therefore, does not compare with that imported into the United States. It is hoped to compete with eastern points after the opening of the Panama Canal, on account of cheaper freight rates, in all cases where magnesite is not necessary for refractory uses. Another use for California magnesite is in manufacturing carbonic acid. The residue from this process is then sold to the other consumers. The producers that operated on any scale during 1911 were the Tulare Mining Company, San Francisco, Cal., the California Magnesite Company, Los Angeles, Cal., Western Magnesite Development Company, San Francisco, Cal., and George Madeira, Healdsburg, Sonora County, Cal. The magnesite deposits on Red Mountain, owned by the Western Magnesite Company, will again be worked in 1912 as the lease given to the Fuller Red Mountain Magnesite Company, San Francisco, Cal., has expired.

Imports.—The chief source of foreign magnesite is from Greece and Hungary. It is imported into the United States, both crude and calcined, in large quantities. In 1910 the quantity of crude imported magnesite was 52,002,557 lb. valued at $162,069, and of calcined material

was 297,652,901 lb. valued at $1,380,731. These are the import values without freight. There was more calcined magnesite imported into the United States in 1910 than in any other year. The amounts of this material imported during the calendar years were as follows: in 1909, 84,814,624 lb. (for the last half of the year); 1910, 286,904,705 lb; 1911, 244,840,369 lb. The import values given were as follows: $9.04 per short ton in 1909; in 1910, $9.46 per ton and in 1911, $9.12 per ton.

During 1910, a shipment of 1500 tons of crude ore was brought to San Diego, Cal., from Santa Marguerita Island, Lower California. The shipment was made by the Compania Mexicana de Magnesita, of Los Angeles, but no manufacturing has been commenced by this Company. In 1911 a small shipment was received from the Island of Margarita, belonging to Venezuela, and American interests took option on other deposits there during the year. A deposit in Quebec was under option to Americans during the summer, but no exploitation was undertaken.

The following tables show the magnesite and magnesia, being all the magnesium compounds imported into the United States, as given by the United States Geological Survey.

IMPORTS OF MAGNESITE INTO THE UNITED STATES. (a)
(Pounds.)

	1908		1909		1910	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Magnesia:						
Calcined, medical	46,823	$7,451	52,247	$8,697	61,471	$9,519
Carbonate, medical	62,514	3,475	49,115	3,328	46,926	2,799
Sulphate	4,990,875	15,543	6,612,956	28,180	6,748,388	23,565
Magnesite:						
Calcined, not purified	129,462,109	655,245	208,947,602	939,014	297,652,901	1,380,731
Crude	39,526,865	81,518	19,635,479	46,005	52,002,557	162,069

(a) From *U. S. Geol. Surv.*

STATISTICS OF MAGNESITE IN THE UNITED STATES.
(Tons of 2000 lb.)

Year.	Production. (a)		Imports.		Consumption.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.
1898	1,263	$19,075	16,039	$134,130	17,302	$153,205
1899	1,280	18,480	20,807	(e) 174,779	22,087	193,259
1900	2,252	19,333	28,821	(e) 216,158	31,073	235,491
1901	4,726	43,057	33,461	(e) 250,958	38,187	294,015
1902	2,830	20,655	49,786	373,928	52,616	394,583
1903	1,361	20,515	54,776	461,399	56,137	481,914
1904	2,850	9,298	38,704	286,828	41,554	296,126
1905	3,933	16,221	74,374	638,619	78,307	654,840
1906	4,032	40,320	90,396	863,492	94,428	903,812
1907	6,405	57,720	99,008	875,359	105,413	933,079
1908	8,967	52,342	84,494	736,763	93,461	789,105
1909	7,942	62,588	114,292	985,019	122,234	1,047,607
1910	8,000	59,780	170,327	1,542,800	178,327	1,602,580
1911	(e) 6,000	48,000	232,209	1,114,520	238,209	1,162,520

(a) Reported by the State Mining Bureau of California. (e) Estimated.

THE PRINCIPAL SUPPLIES OF MAGNESITE.
(In metric tons.)

Year.	Austria-Hungary. (a)		Greece. (d)		India. (d)	United States. (d)
	Crude	Calcined	Crude	Calcined		
1898	.....	(b)	14,829	.....	(e)	1,146
1899	.....	(b)	17,184	.....	(e)	1,161
1900	.....	(b)	17,277	.....	(e)	2,043
1901	.....	40,236	20,348	.....	(e)	4,286
1902	.....	53,467	23,020	.....	3,597	2,567
1903	.....	69,058	28,415	.....	838	1,234
1904	.....	53,781	9,133	.....	1,193	2,585
1905	.....	92,359	37,063	.....	2,645	3,568
1906	.....	87,765	40,584	.....	1,861	7,093
1907	452	113,695	55,816	.....	188	(f) 6,858
1908	212	87,049	63,079	.....	7,655	(f) 5,994
1909	1,027	125,666	56,797	16,609	199	(f) 8,588
1910	341	182,911	18,073	19,982	1,405	(f)11,291

(a) Exports of calcined magnesite. (b) Previous to 1901 magnesite was included with other minerals not elsewhere specified. (d) Crude magnesite. (e) Not reported. (f) U. S. Geol. Surv.

Sepiolite or meerschaum is a silicate of magnesia produced at Eski-Chehir and other villages in Turkey; also Egypt, Moravia, Spain and Portugal. Most of it goes to Vienna. The town of Ruhla alone, in Austria, exports annually pipes made from this material valued at $1,500,000. The production of Turkey in 1911 is estimated at 250 metric tons[1] and is decreasing. In digging for meerschaum a reddish brown earth is met at a depth varying from a few to 20 yd. beneath the surface, but such depths as 40 yd. and more are not unknown. The meerschaum is found in this layer of earth in nodules, the size of which rarely exceeds 16 cu. in.

ANALYSES OF MAGNESITE.

	SiO_2 Per cent.	Al_2O_3 and Fe_2O_3 Per cent.	CaO Per cent.	CO_2 Per cent.	MgO Per cent.	$MgCO_3$ Per cent.
Greece	1 to 2	1.75 to 3.20	1.10 to 3.20	.....	.....	92 to 98
Elba	8.5	Trace.	3.50	.....	41	.....
Transvaal	2.3	0.8	.....	49.8	45.3	.....
New South Wales	0.42	0.54	*Nil.*	.....	.....	99.01
New Caledonia	0.8	0.8	3.3	51.5	42.4	.....
Silesia	5.6	0.85	0.12	.....	.....	93.0
Hungary	0.75	3.50	1.2	50	45	.....
Lower California	Trace.	0.21	0.43	.....	.....	99.36
Atlin, B. C.	45.7	5.10	.....	27	22	.....
Margarita, Venezuela	0.2	0.15	0.56	51.6	46.93	98.04
Magnesium Carbonate	.....	.....	.....	52.4	47.6	.....

There are two varieties of magnesite. The amorphous or dense variety has its origin from olivin acted upon by carbonic acid waters, forming veins, with the associated serpentine residue. The crystalline variety seems to be associated with iron carbonate deposits in the vicinity and is supposed to have originated from the action of solutions of lime-

[1] *Min. and Sci. Press.*, Jan. 6, 1912, p. 59.

stone containing magnesia. The residues, in this case, are products from the solution such as talc, clay and quartz. The dense variety occurs in Greece and also in the oldest producing locality of Kraubath, Obersteier. It occurs also in Scandanavia, Transvaal, India, California and the northeast coast of South America. This material is used for moulded and burnt ornaments, and also for the manufacture of carbonic acid. In order to use it as a refractory, the finely ground material is given sintering qualities by the addition of finely powdered iron oxide.

The crystalline variety occurs in the Austrian Alps, Upper Hungary and in the South Urals at Ufa. It is generally in the form of superficial deposits or sometimes as funnels. The analysis of the raw material is magnesium carbonate, 80 to 95 per cent., calcium carbonate 5 to 1 per cent., iron carbon, 8 to 2 per cent., alumina 2 to 1 per cent., insoluble 5 to 1 per cent. For the low limit of magnesium carbonate, it is generally impossible to remove impurities such as talc, etc., to bring the magnesium carbonate up to 83 per cent., which is the requirement for sintering magnesite. The requirements for the best quality of sintered magnesite are magnesia 83 per cent. at least, iron 2 1/10 per cent. at least, lime 5 per cent., alumina 2 per cent. and silica 5 per cent. at most. A deposit containing fine inclusions of calcite and dolomite might render it valueless as it would cause sintering at too low a temperature.

The methods of working magnesite are generally crude as the prices are not such as to induce working on a large scale. It is usually worked in terraces beginning at the highest point. After crushing, the rock is hand picked and it usually takes from 2 to 3 tons of crude material for 1 ton of the finished product. In sorting, quartz is often left in the product as it will generally break off on burning. A difficulty in the sintering of magnesite in the furnaces generally used in Europe is the crumbling of fine particles which tend to choke the draft. As the material sinters, the pieces shrink and adhere together but the mass is easily broken up with bars.

The following details are largely taken from Austrian practice.[1]

In 1887, the Lezius furnace was adopted in Austria for magnesite burning. It is a shaft kiln 5 to 6 meters high and is 1 to 1.2 meters in diameter. For large and more refractory or purer pieces, the height is made as much as 12 to 15 meters and the diameter 2 to 2 1/2 meters. The products of combustion enter the column at one-third the height from the bottom. A preheated air blast is used under the fire-boxes, which are separate to one side of the shaft, and one or two in number. The lower third of the shaft is the cooler and in order to discharge, is

[1] J. Hörhager, Preparation of Magnesite, *Stahl und Eisen*, June 15, 1911.
J. Hörhager, The Origin of Alpine Magnesite Deposits, *Oestr. Zeitschr. f. Berg-u. Huttenwesen*, April 22, 1911.

provided with grate bars and air tight drop doors. The lining of the kiln is made of silica; it adheres only slightly to the sintering magnesite and much less so than if the lining were magnesite. At the time of drawing the kiln which is four times in each 24 hours, the cool air current causes the st cking pieces of ore to break off. The drawing takes 1 1/2 hours each time. The capacity of such a kiln is 30 to 50 cubic meters, and produces 10 to 15 tons of product in 24 hours. The coal fuel required is 30 to 40 per cent. of the product. The temperature for burning pure magnesite is practically 1700° C. For the iron bearing magnesite, which runs as high as 3 to 4 per cent. ferric oxide, the temperature is 1400° C. The color of this burnt material is dark brown or black, the iron contents being converted into magnetite, more or less, depending upon the oxidation of the flame and the continuity of the process.

In cleaning the sintered product, the calcite and dolomite are found in the fines and separated by screening, cobbing and hand picking. It has not been found successful to burn coke fuel mixed with the magnesite as it is too high in ash. Retort furnaces have been tried for sintering magnesite but have been found unsatisfactory as the process is not continuous. The space required produces only one-half of the volume as product, thus requiring 16 retorts to equal one shaft furnace. The fuel combustion, however, is found economical. In trying the use of gas firing, it was found that the product was not uniform; in this method the product, when sintered was drawn from the shaft into two fore hearths. The process answers for the amorphous or non-sintering magnesite. An air pressure of at least 1/2 lb. per square inch is necessary to prevent choking in the first stage on account of calcite and dolomite present. Reverberatory furnaces and rotary kilns have also been tried but give difficulty in that the material sticks to the interior of the furnace.

In the manufacture of brick, the hot material is wet with water and then allowed to weather for 4 weeks. It is then sorted and ground to 1 millimeter size in a ball or Chilean mill. Several sizes are sometimes mixed so the fines fill the spaces between the coarse material. The pressure for molding these bricks is from 80 to 300 atmospheres. The damp mass of material is kept for several weeks to allow uniform distribution of the water. This also probably causes a chemical combination of the water with the magnesia. The bricks are then dried at 20° C., which will show up cracks if lime, etc., are present. Good magnesite does not require any binder.

The bricks are burned in muffle furnaces of the type invented by G. Mendheim in München, for silica brick. The time of burning is 24 hours and the fuel used 30 per cent. The muffle linings in this case are

magnesite. An ordinary brick does not shrink more than 1 or 2 mm. in burning. The finished bricks are dark brown and if not sufficiently uniform are reground in the mills. Magnesite changes to a lighter color at 2000° C. and at 2500° reduction of magnesia begins. The present consumption of magnesite for the world is in the neighborhood of 50,000 tons.

Austria.—In Austria, at Radenthein, a new factory was completed for the manufacture of calcined magnesite. The capacity is 100 tons per day and is expected to be increased after some minor alterations. The total production of this factory is exported to the United States. This is the property of the Austro-American Magnesite Company, a related company to the American Refractories Company of Chicago, Ill.

India.—The principal deposits of magnesite in India are in the Chalk Hills, near Salem, Madras. They have been described by R. B. Foote,[1] C. S. Middlemis[2] and H. H. Dains.[3] The area is about 2000 acres and the depth extends to 60 ft. The veins occur in dunite. At Salem the caustic was first obtained by calcination in ordinary lime kilns but this gave an inferior product. A refined gas fired kiln now erected runs well. The continental consumers of magnesite in 1908 stipulated at Leipzig that impurities be less than 4 per cent., a requirement easily fulfilled by the Indian magnesite. Calcination experiments in India for electrically burning and partly fusing the ore gave a crystalline product having a gravity of 3.58. The material is an excellent wash and lining for electric furnaces. Bricks are usually made in India by grinding the calcined product in edgerunner mills, mixing with water and molding. Another process is to grind the crude material, mix it with water, mold and then burn the resulting bricks. On the continent tar is used with the carbonaceous residue acting as a binder. Analysis of the Indian burnt magnesite is as follows: magnesia 93.94, lime 1.53, iron and aluminum oxide, 1.91, silica 2.60 per cent. A cargo sample of Indian magnesite gave the following analysis: magnesium carbonate, 96.34; lime, .78; silica, 1.17; iron and aluminum oxide, .14 per cent; magnesia oxide, .06; combined water, 1.30 per cent. The production of magnesite from these deposits in 1910 was 5182 tons valued at £1382. In 1909, the output was 737 tons valued at £196, in 1908, the production was 7543 long tons.

Greece.—The magnesite that was dead burnt in 1910 amounted to 2278 tons; 2090 tons sold for 173,164 francs. The stock on hand at the beginning of the year was 508 tons. Calcined magnesite was produced to the extent of 17,016 tons, the magnesia contents being 90 per cent.

[1] *Mem.*, Geol. Surv. Ind., IV, pp. 312–317.
[2] *Records*, Geol. Surv. Ind., XXIX, p. 31.
[3] *Journ.*, Soc. Chem. Ind., XXVIII, p. 503.

The value of 17,892 tons sold was 1,237,928 francs. The stock on hand of calcined magnesite at the beginning of the year was 14,059 tons. The crude magnesite produced in Greece in the year 1910 was 48,913 tons. The actual amount sold during the year, however, was only 18,073 tons valued at 354,406 francs. The two principal producers in Greece are the Société Anglo Greek de Magnesite operating at Limni and Chalcis, and the Société Hellenique de Mines, operating at Limni, Mantoudi and Pyli. The magnesite is reported to contain 90 per cent. magnesia. There were also sold during the year 295 tons of magnesite bricks valued at 75,000 francs.

Norway.—The deposits in Norway are in the vicinity of Snarum and the veins run 13 ft. thick. Various analyses of magnesite for these localities are given in the table.

Laying A Magnesia Furnace Bottom

The method of laying furnace bottom in the 15-ton electric steel furnace at South Chicago was described at the meeting of the Chicago section of the American Electro-Chemical Society.

The furnace shell is of plate steel, 1 in. in thickness, riveted together. The outside horizontal cross-section plan is approximately that of a complete circle of 13 1/2 ft. in diameter, with two flattened portions situated at the front and back respectively. On the bottom of the furnace, within the 1-in. plate, and next to it, one row of magnesite brick laid the 4 1/2-in. way is placed across the flat portion. The side walls of the furnace are vertical and consist of two rows of magnesite brick laid the 9-in. way, giving a thickness of 18 in. of magnesite brick. These solid magnesite brick walls extend up to the furnace roof. The bottom proper of the furnace consists of dead burned Spaeter magnesite to a depth of 12 in. at its thinnest point, which is, of course, at the extreme center. From this thinnest point the bottom slopes gradually upward so as to form a portion of a sphere 7 ft. 2 in. in radius.

This bottom was fixed in the following manner: Dead burned and carefully ground Spaeter magnesite was mixed with basic open-hearth slag in the proportion of four of magnesite to one of open-hearth slag. To this mixture enough tar was added to make the mass sufficiently plastic to enable it to be tamped into the furnace in the usual manner. The entire depth of the bottom was tamped in this way. Next the furnace was filled with wood, dried out for about 48 hours, and then filled with coke and the electrodes lowered and the current turned on. In this way the bottom was fluxed into place.

The practice of using basic linings in copper converters in the United

States is of recent importance. The use of magnesite gives much better results than the formerly used linings.

As near as can be ascertained, the present production of refractory magnesite in the form of bricks, in the United States, amounted to a little over 100,000 tons in 1911. A new product made of magnesite is a crucible that withstands the high temperature in the manufacture of iron alloys free from carbon. Magnesium oxide made by the electric heating process appears to answer these requirements. There is a possibility that this product will come into commercial use.

MANGANESE

The manganese industry of the United States is very small, having declined from a considerable production 20 years ago to hardly any high-grade product in 1911. Foreign ores continue to be imported to a large extent each year, but no new finds of high per cent. material are being made. The United States Geological Survey reports the production in this country as 2258 tons in 1910, and 2457 tons in 1911, mostly from Virginia, though Arkansas is stated to have some production. The average prices of this ore were $12.74 and $10.14 in 1909 and 1910, respectively. The total production to 1911 is given as 406,600 long tons.

The imports into the United States were 212,755 long tons in 1909; 242,348 tons in 1910 and 176,852 tons in 1911. The import values averaged about $6.62, $9.68, $6.72 in these years respectively. The ore is duty-free.

The production of manganese-iron ore is increasing. This ore is too low in manganese to be classed as a manganese ore and is therefore classed as iron ore. It is produced largely in the Lake Superior district together with smaller amounts from the Appalachian region. The manganiferous-iron ores are used in the blast-furnaces for production of a high-manganese pig iron which has its special uses.

In the accompanying tables of production and imports of manganese ores, it should be borne in mind that the figures representing total domestic production include an excess tonnage of low-manganese materials, while the imports consist almost entirely of ores containing at least 50 per cent. manganese.

The ores are all oxides containing such minerals as pyrolusite, bauxite, etc. The mines that were formerly good producers are not as productive as formerly and new discoveries are desirable. Specially where the oxygen content of the ore is valuable for the manufacture of chlorine, the use of low-grade material with quartz and other inert substances was necessary and the price of high-grade material increased, so that substitutes were sought. The regeneration of oxide, from the manganese chloride produced in this industry was then accomplished by processes such as those of Dunlap, P. W. Hofmann, and W. Weldon.

PRODUCTION OF MANGANESE ORES IN THE UNITED STATES. (*a*)
(Tons of 2240 lb.)

	Manganese Ores.				Manganiferous Iron Ores.				Man. Zinc Ores.	Total Production.	
	California.	Georgia.	Virginia.	Other States.	Arkansas.	Colorado.	Lake Superior.	Va. & N. C.	New Jersey.	Long Tons.	Value.
8.	393	2,477	3,307	1,250	2,775	17,792	112,318	....	47,470	187,782	$16,627
9.	263	1,623	3,626	105	855	29,161	53,702	....	53,921	143,256	306,476
0.	131	3,447	7,881	312	*Nil.*	43,393	75,360	*Nil.*	87,110	217,546	1,172,447
1.	610	4,074	4,275	3,036	*Nil.*	62,385	512,084	20	52,311	638,795	1,644,117
2.	846	3,500	3,041	90	*Nil.*	13,275	884,939	3,000	65,246	973,937	2,145,783
3.	16	500	1,801	508	*Nil.*	14,856	566,835	2,802	73,264	660,582	1,670,349
4.	60	*Nil.*	3,054	32	600	17,074	365,572	*Nil.*	68,189	454,581	789,132
5.	1	150	3,947	(*e*) 20	3,321	45,837	720,090	*Nil.*	90,289	863,663	1,681,472
6.	1	.....	6,028	892	8,900	32,400	1,000,008	*Nil.*	93,461	1,141,681	*e*)3,403,993
7.	100	*Nil.*	(*d*)4,604	900	4,133	(*d*) 99,711	(*d*) 314,316	*Nil.*	93,413	517,177	1,510,156
8.	321	*Nil.*	(*e*)6,144	200	(*d*) 4,066	(*d*) 51,524	(*d*) 467,140	274	110,225	633,650	1,280,989
9.	3	*Nil.*	1,334	*Nil.*	(*d*) 3,325	(*d*) 65,024	(*d*) 775,035	305	141,264	986,481	2,209,461
0.	(*f*)	*Nil.*	2,059	199	(*d*) 5,030	(*d*) 55,770	(*d*) 558,634	301	137,173	759,166	1,938,429
1.	(*f*)	*Nil.*	*g*)2,457	*Nil.*	(*d*)2,177	(*d*)41,753	479,920	507	109,296	634,110	1,386,464

(*a*) Statistics of 1900–1906 are by the U. S. Geological Survey. (*b*) Includes 1300 tons of manganous iron ore from Vermont. (*c*) As reported by Virginia Geological Survey. (*d*) As reported by S. Geological Survey. (*e*) Estimated. (*f*) Figures not yet available. (*g*) Includes Tennessee and ifornia.

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*)
(In metric tons.)

Austria-Hungary.	Belgium.	Bosnia (*b*)	Brazil (*d*)	Canada.	Chile. (*d*)	Colombia.	Cuba.	France	Germany.	Greece.	India.
10,043	28,372	5,344	16,054	14	23,528	8,382	.....	37,212	46,427	11,868	74,862
14,219	16,440	5,320	26,417	45	20,851	11,176	.....	31,935	43,354	14,097	61,469
10,484	.....	5,270	65,000	279	40,931	10,160	.....	39,897	61,329	17,600	88,520
14,550	10,820	7,939	108,244	34	25,715	8,748	21,973	28,992	59,204	8,050	129,865
12,077	8,510	6,346	100,414	447	18,480	95	25,588	22,304	56,691	14,166	122,831
12,883	14,440	5,760	157,295	175	12,990	*Nil.*	40,048	12,536	49,812	14,960	160,311
11,489	6,100	4,537	161,926	135	17,110	(*c*)	21,070	1,583	47,994	9,340	174,563
15,460	485	1,114	208,260	123	2,324	(*c*)	33,152	11,254	52,886	8,549	152,601
23,732	*Nil.*	4,129	224,377	22	1,323	(*c*)	(*d*) 8,096	6,751	51,463	8,171	250,788
20,577	120	7,651	121,331	84	35	(*c*)	(*d*) 13,997	11,189	52,485	(*d*) 9,200	579,231
24,954	2,100	7,000	236,778	1	(*c*)	(*c*)	30,486	18,200	74,683	10,000	916,770
27,257	7,130	6,000	166,122	*Nil.*	1	(*c*)	1,492	15,865	67,692	10,750	685,135
29,966	6,270	5,000	240,774	*Nil.*	(*c*)	(*c*)	2,976	9,378	77,177	5,374	652,958
28,964	*Nil.*	4,000	253,953	*Nil.*	(*c*)	(*c*)	(*c*)	7,925	80,559	41	813,722

ır.	Italy.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	Spain.	Sweden.	United Kingdom	United States. (*e*)
7..	1,634	15,448	182	1,652	403	263,115	100,566	2,749	609	161,138
8..	3,002	11,497	220	907	68	329,276	102,228	2,358	235	190,787
9..	4,356	11,336	137	2,049	747	659,302	104,974	2,622	422	145,548
0..	6,014	15,831	166	1,971	77	802,236	112,897	2,651	1,384	221,714
1..	2,181	16,270	208	904	221	522,395	60,325	2,271	1,673	649,016
2..	2,477	10,844	*Nil.*	(*c*)	4,674	536,519	46,069	2,850	1,299	989,519
3..	1,930	5,616	71	30	1,341	414,334	26,194	2,244	831	671,151
4..	2,836	4,324	199	(*c*)	843	430,090	18,732	2,297	8,880	461,854
5..	5,384	14,017	55	(*c*)	1,541	508,635	26,020	1,992	14,582	877,482
6..	3,060	54,339	16	22	1,131	1,015,686	62,822	2,680	23,126	1,141,681
7..	3,654	20,589	26	1,374	1,134	995,282	41,504	4,334	16,356	517,177
8..	2,750	11,130	*Nil.*	(*c*)	1,403	362,303	16,945	4,616	6,409	633,650
9..	4,700	18,708	6	(*c*)	613	574,938	7,827	5,212	2,812	986,477
0..	4,200	11,120	5	(*c*)	805	(*d*)668,050	8,607	5,762	5,554	784,464

a) From official statistics. (*b*) Includes Herzegovina. (*c*) Statistics not available. (*d*) Export rns. (*e*) Includes output of manganiferous iron ore. (*f*) Six months.

CONSUMPTION OF MANGANESE ORE IN THE UNITED STATES.
(Tons of 2240 lb.)

Year.	Imports.		Consumption.		Production of Manganese-Silver Ores. (b)	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1898	114,885	$ 831,967	302,667	$1,248,594	99,651	$295,412
1899	188,349	1,584,528	331,605	1,891,004	79,855	266,343
1900	256,252	2,042,361	473,798	3,214,808	188,509	897,068
1901	165,722	1,486,573	804,568	3,130,690	228,187	865,959
1902	235,576	1,931,282	1,209,513	4,077,065	174,132	908,098
1903	146,056	1,278,108	806,638	2,948,457	179,205	649,727
1904	108,519	901,592	563,100	1,690,724	105,278	348,132
1905	257,033	1,952,407	1,120,696	3,633,879	127,170	445,095
1906	221,260	1,696,043	1,362,941	5,100,036	163,760	573,160
1907	209,021	1,793,143	1,531,882	5,653,408	103,844	259,473
1908	178,203	1,350,223	825,353	2,644,523	51,554	123,407
1909	212,765	1,405,329	1,199,246	3,920,592	65,024	209,495
1910	242,348	1,711,131	880,248	3,573,112	55,770	176,448
1911	176,852	1,186,791	769,209	2,464,000	41,753	109,340

(b) Mined in Colorado and used as flux in silver-lead smelting; not included in the statistics of consumption. The statistics of manganiferous silver ore for 1908–10 are as reported by the U. S. Geological Survey. (c) Figures not available.

UNITED STATES PRODUCTION AND IMPORTS OF IRON-MANGANESE ALLOYS.
(Tons of 2240 lb.)

	1908		1909		1910		1911	
	Production	Imports	Production	Imports	Production	Imports	Production	Imports
Ferromanganese	40,642	44,624	82,209	88,934	71,376	114,227	74,482	80,263
Spiegeleisen	111,376	4,579	142,831	16,921	153,055	25,383	110,235	20,971
Totals	152,018	49,203	225,040	105,855	224,431	139,610	184,717	101,234

PRODUCTION OF MANGANESE ORE IN INDIA. (a)
(In tons of 2240 lb.)

State.	1905	1906	1907	1908	1909	1910
Baluchistan	.....	.....	15	.....	.....	.....
Bengal	.....	1,000	2,933	20,000	55,060	41,958
Bombay	640	7,520	22,821	23,232	17,657	30,921
Central India	30,251	50,073	35,743	13,315	10,324	12,664
Central Provinces	151,547	351,880	565,017	431,055	381,285	552,239
Madras	64,989	114,710	162,455	118,089	138,454	120,607
Mysore	.....	46,312	113,307	68,624	39,895	42,518
Totals	247,427	571,495	902,291	674,315	642,675	800,907

(a) From *Records* of the Geological Survey of India.

The principal manganese minerals are psilomelane, 49 to 62 per cent. Mn; and pyrolusite, containing up to 63 per cent. Mn. The carbonate, rhodochrosite, equivalent to 61.72 per cent. MnO_2, and the silicate, rhodonite, equivalent to 54.15 per cent. MnO_2, are possible sources of manganese. It should be noted, however, that in an ore intended for

blast-furnace use, such of the manganese as is chemically combined with silica is not only useless, but is actually a detriment on account of the increased amount of manganiferous slag produced. Inasmuch as manganese ores are not consumed for the purpose of extracting metallic manganese, but as a means of adding manganese to pig iron, by making ferromanganese, any elements that would be objectionable in an iron ore are also detrimental in a manganese ore. Manganese is a usual constituent of iron ores, and it is estimated that 2 per cent. is an average content; part of this goes into the slag and part into the iron.

Most of the high-grade manganese ore is obtained in Russia, India and Brazil, the buyers' requirements have steadily advanced, until now ores averaging from 50 to 60 per cent. manganese are commonly required.

During the last 10 years Russia has produced about 40 per cent. of the total amount of manganese ore mined in the world. Its annual production is about 550,000 long tons, of which about one-fifth comes from Nikopol, in southern Russia, and about four-fifths from the Caucasus. About one-fourth of the total production is consumed in Russia, and of this consumption about 75 per cent. comes from Nikopol, so that nearly the entire output of the Caucasus is exported. The principal competitors of Russia in the world's market are India and Brazil; the largest consumer of manganese ore is the United States, followed by Germany and England. In the American market, Russian ore has now disappeared, and in the other markets also the consumption of manganese ores mined elsewhere than in Russia is growing. Consumption in England and Germany has increased exceedingly in recent years, with the result that exports from Russia to these two countries have nearly doubled; whereas up to 1907 about 46 per cent. of the English imports and 72 per cent. of the German imports came from Russia, in that year Russia provided only 24 per cent. of the manganese ore consumed in England, and only 50 per cent. of that consumed in Germany.

United States.—"The Manganese Deposits of the United States" is made the subject of a 298-page bulletin, No. 427 of the United States Geological Survey, written by Edmund Cecil Harder and published in 1910. This bulletin describes practically every known deposit of manganese in the United States, giving typical analyses of ores from different districts. There is also a chapter on the uses of manganese ore and notes of historical interest. Chapter III is devoted to foreign manganese deposits in which is included descriptions of the principal manganese deposits of the world. Chapter IV is devoted to the chemistry of manganese and a study of its various minerals and alloys with other metals. A bibliography which covers the manganese literature, is added.

In the American market, the basic requirements for manganese ore are that it shall not contain more than 0.20 per cent. phosphorus, nor more than 8 per cent. silica, for each 0.02 per cent. phosphorus in excess of 0.20 per cent., 2 cents per unit of manganese is deducted per ton, and for each 1 per cent. of silica in excess of 8 per cent., 15 cents per ton is deducted. The base price fluctuates according to the manganese contents, as follows: 49 per cent. and over, 26 cents per unit; 46 to 49 per cent., 25 cents per unit; 43 to 46 per cent., 24 cents per unit; 40 to 43 per cent., 23 cents per unit. Ore containing less than 40 per cent. manganese, or more than 12 per cent. silica or 0.225 per cent. phosphorus is subject to acceptance or refusal at buyer's option. Settlements are based on analysis of sample dried at 212° F. The percentage of moisture in the sample as taken is deducted from the weight.

Virginia.—This state has more manganese mines than any other in the United States. During the year 1910 over forty small producers furnished the tonnage. The Mountain or Valley province on the west slope of the Blue Ridge furnished the principal production. In addition there are two other regions, namely the Piedmont, chiefly in Campbell and Nelson Counties, and the Coastal Plain area. In the first two areas the ore distribution generally conforms to the clay beds. In the Piedmont region the residual clays are derived from the crystalline schists and while some of the stringers follow the structure of these, the ore is also formed in nodules and pockets. In the mountain region are many iron mines, and the ores are manganiferous. Similarly the manganese mines contain much iron. The manganese ores are contained in residual clay derived from shales, which overlie Cambrian quartzite. In some places the clays are derived from Cambro-ordovician limestone. The Crimora mine in Augusta County has been a producer for over 30 years, and has produced more manganese ore than all other mines in the United States combined.

Tennessee.—Wilbur A. Nelson[1] describes the occurrence of manganese ore. In Fort Payne chert is a formation which had been neglected in the search for manganese in Tennessee heretofore. The old deposits in Tennessee are associated with either the Beekmantown formation of Ordovician age or the limestone and associated shales and quartzite of Cambrian age. The best deposits are owned by Mr. J. H. Artz of Cleveland, Tenn., who has done considerable prospecting. The deposit is on the eastern flank of White Oak Ridge about 8 miles west of Cleveland. As no regular mining has been done, it can only be stated that a large amount of chert-bearing ore is scattered over the surface and that ore

[1] Resources of Tennessee, December, 1911.

found assays manganese 10 to 60 per cent., limonite 50 to 5 per cent., and the silica less than 10 per cent.

Manganese in Foreign Countries

The figures showing the movement of manganese ore increase in interest from year to year, and the rapidly growing consumption of this article in the steel and iron trades of the world is striking. During the 8 months from January to August, 1910, the manganese arrivals in European trade centers amounted to 1,230,000 tons, against 924,000 tons in 1909 and 763,000 tons in 1908. The receipts were nearly 1,830,000 tons for 1910. The total arrivals for the year 1898 were 539,000 tons. In 1905 they had risen to 963,000. The requirements of the trade almost doubled in the seven years and have again almost doubled since 1905.

Africa.—Consul-General Julius G. Lay, of Cape Town, states that the first shipment, of any importance, of manganese ore from Cape Colony consisted of 5000 tons, billed to Antwerp. It is stated by the mine owners that this is part of an order for 150,000 tons, to be executed within the next two years. Already 13,000 tons more of manganese and about 7000 tons of iron ore are ready for shipment. The ore contains phosphorus, and is unmarketable in the United States.

Australia.—The Mount Miller mine, Queensland, has developed considerable ore reserves. In 1910, 786 tons of 71.2 per cent. manganese ore were produced, the value being £3,106.

Brazil.—Since 1895 much manganese ore has been exported from Brazil, where it is mined in Minas Geraes, Bahia, Sao Paulo and Matto Grosso. This ore generally carries 50 per cent. or more of metallic manganese and brings the highest prices in London and New York.

Large deposits occur in the Miguel Burnier, Ouro Preto, and one or two other distrcits in the State of Minas Geraes. Analyses of ore from Miguel Burnier district show manganese dioxide about 80 per cent. and monoxide 5 per cent. with from 0.02 to 0.03 per cent. phosphorus. The manganese value is 55 per cent. Ore from the Ouro Preto district contains about 53 per cent. manganese and 0.03 per cent. phosphorus.

The ore reserves of the Queluz, Miguel Burnier and the Gandarella have been stated as 192,000,000 in 1908. In Matto Grosso several mines are said to contain 210,000,000 reserves in addition.

Early in 1911 only four mines were active producers, the mines being situated on a zone served by the Brazil Central Railway. The distance from Rio de Janeiro is from 438 to 501 kilometers. The farthest mine is the Wigg, which has a reputation on account of the purity of the ore, this being nearly free from silica and phosphorus. The average

output is transported 463 kilometers (288 miles); up to June, 1910, the freight rate was 6 milreis per metric ton to Rio for distances under 500 kilometers and 12 reis per ton kilometer for mineral sent a longer distance. This rate was altered to 5 milreis per ton for the short hauls and 10 reis per ton kilometer for long hauls. In the case of the Usina Wigg being 1 kilometer over the short limit, the rate is 5 mil and 10 reis. These rates correspond to .5194 cents per ton mile. In India the rate is lower, namely, .428 cents per ton mile, but in the Caucasus it is higher. The Brazilian Government to develop the iron industry gives special rates for iron ores and other minerals, .3854 cents per ton mile (1 milreis is .33 cents), but this does not apply to manganese.

The exports of ore are estimated for 1911 at 211,200 tons.

Canada.—But one company was engaged in mining this mineral, namely, the Nova Scotia Manganese Company at New Ross. This company has been actively engaged in mine development, and in the erection of surface buildings and placing equipment. Only for test purposes has any ore been shipped so far.

High-grade pyrolusite, such as the product of this mine, is used by glassmakers, paintmakers, steelmakers, electrical equipment makers, and manufacturing chemists. The poor grades or run-of-mine product being taken by the steelmakers, and the highest grade, or what is known as air-float product, being taken by the manufacturing chemists.

Chile.—In 1907, 51,682 tons of manganese ores were exported but at present prices there is no profit in this industry. The large deposits 55 miles east of Apolillado Bay, (Serena) have a future, being near the new Railway.

China.—The manganese deposits of China are receiving some attention as shown by the Hsing-Kuo Chou mine which produced 1500 tons of manganese ore during 1909. This mine is under the same management as the Hanyang Ironworks. In 1908, 3600 tons of manganese ore of good quality were produced at Ping-hsiang, in the province of Kiangsi. The same company is also in a position to mine large quantities of limonite carrying 20 per cent. manganese in other localities. Some of these are along the Yangtze in the northwestern part of the province.

Germany.—The consumption of manganese ores is about 250,000 tons per year, of which only about 50,000 tons are produced in Germany. The most convenient source of supply for the extra 200,000 tons is the Caucasus district, but on account of the political uneasiness in Russia, it is becoming an interesting problem to arrange for future supplies obtainable with regularity. In the Bonn district in Prussia 80,325 metric tons of ore were produced, valued at 957,053 marks, in 1910. The production of the Empire in 1911 was 87,297 metric tons.

Greece.—Forty-one tons of ore were sold to foreign countries in 1910, but the stock on Dec. 31, 1909, was 10,678 metric tons. The average grade was 32 per cent. from Milos and 35 per cent. (about 1/10 of the whole) from Andros. In 1909 the production was 1287 metric tons value of 58,944 Dr. for the sale of 5374 metric tons. In 1910 only 41 metric tons were sold, at an average price of $11.75. In 1910, 35,594 metric tons of ore were produced; stocks on hand were 45,000 metric tons.

India.—The manganese industry of India was described in detail in Vol. XXXIX of the *Records* of the Geological Survey of India. In Vol. XL of this publication it is stated that the production of manganese ore from the four principal provinces shows a decrease for 1909, but the production in 1910 increased to 800,907 long tons valued at £849,155, an average of £1 1s compared with 18s 10d per ton in 1909. The exports of manganese ore amounted to 467,454 tons in 1909 and 586,577 tons in 1910. These figures do not include the exports from Mormugao which port deals with the Saundur and Mysore manganese; these exports will amount in addition to about 100,000 tons. In the Central Provinces, the average number of persons employed daily in manganese mining is 10,947; in Madras, 5451 and in Bombay, 651. The following average prices, a good indication of the world's price fluctuation, are taken from the *Mining Journal:* First-grade ore in 1909, 9.2d and in 1910 9,5d; second-grade ore 8.3d and 9.3d; third-grade ore 8.8d in 1910 per unit per ton.

In 1905–1907[1] the rise in price was due to the demand by the steel trade which uses .9 of all the manganese ore produced, just when the political disturbances in Russia caused the reduction of the most important source of supply. Prospecting in India and elsewhere was then stimulated and led to the discovery and working of new deposits of the ore. In 1907 the Indian production reached a maximum of 916,770 metric tons. Up to 1900 the production was limited to the Vizagapatam district, in Madras, but since 1902 the Central province mines have furnished the bulk of the ore. In 1907 the prices rose for first grade from 8 3/4d in 1904, to 16 and 18d per unit. Owing, however, to the financial crisis in 1907, the steel production being lessened in the fall of that year, the large stocks of ore depressed the market price. The next year only those mines that had contracts at high prices were worked. The total production in 1911 is estimated at 656,512 tons.[2] The manganese deposits of India can be divided into three general classes but the ore occurrences are similar in character.

In the Vizagapatam district the ore is strewn in soft decomposed rock, cropping on level ground. The deposits at Sandur, Mysore and Goa form the crests of small hills or are segregations in the subsoil or

[1] Hubert A. Carter, *Min. and Sci. Press*, Dec., 30 1911.
[2] British Consul, Batoum.

laterites. Other Central Province deposits are regular beds interstratified in metamorphic rocks, usually schist and quartzite with steep dips. Many tons of ore exist obtainable by the present open-cut method, but a general lack of underground development prevents an estimate being made. Though the quantity is large, those best fitted to speak on the subject consider it necessary to make use of much second-class ore, using concentration methods to bring up the grade.

The method of working in India is generally by contract with coolie labor. Blasting is done in the rock or mere digging to recover the ore boulders in the soil. Breaking the ore by hand and sorting is a common practice. The ore is headed or carried by tramway or chute from the mine. The total mining cost varies from 1 rupee in the Central Province to 6 in some of the Vizagapatam or Mysore mines. The royalty is 2 1/2 per cent. on the selling value at the mine, or a fixed royalty in the native states of 4 to 6 annas is charged. After sampling and measuring the stacks of ore made by the headers, the ore is hauled to the railroad at a cost of 2 to 4 annas per ton-mile or by tramway which costs 6 pice. The railroad rate to either the ports Bombay, Vizagapatam, Marmagoa, or Calcutta costs as follows: for Vizagapatam deposits R 1–4–0; for Sandur and Mysore ores R 5–12.0; for Central Provinces (Bombay) R 8 and (Calcutta) R 10 Port charges are R 1–4–0 and R 1–8.0 in addition. Freight to English ports is usually R 12 and to the United States R 13–8–0. Central Provinces ore costs about R 27 delivered c. i. f. at English and Continental ports and Vizagapatam ore costs R 28–8–0.

In contracts for buying and selling the ore the silica and phosphorus allowable are arranged for; generally 8 or 10 per cent. and .1 to .12 per cent. respectively being upper limits above which ore may be rejected. Some ore containing phosphorus up to .25 per cent., is sold at a lower figure, and used for smelting sulphury iron ores. Fines are also penalized when above 15 per cent.

Japan.—The production of manganese ore in 1907 was 20,586 metric tons; in 1908, 11,130 tons, while in 1909 the output was only 6607 tons. The decrease of production in 1908 was due chiefly to a general depression in the mining business.

Japanese ores are sold at Hamburg on the following schedule, in marks per ton on the basis of contents of manganese dioxide: 87 per cent. and above, 115 marks per ton; 85–90 per cent. (minimum 85), 95 marks per ton; 80 per cent., 85 marks per ton; 70–75 per cent., 75 marks per ton; 65–70 per cent., 50 marks per ton. These Japanese ores average from 44–56 per cent. manganese, phosphorus is about 0.05 per cent. and silica 7 to 10 per cent. They go by the name of Japanese brownstone, and bring a high price on account of their purity.

Mexico.—Lower California possesses some manganese deposits. These are on the point of land that encloses the Gulf of Concepcion, and are 14 miles across the bay from Mulegé on the Gulf of California. The deposits are known locally as the Guadalupe and the Gabilan. The peninsula consists of igneous rocks and the manganese occurs in what appear to be veins; much black float abounds. The material of the Guadalupe in place has been assayed, samples giving 17.54 Mn, 1.63 Fe, 3.61 CaO and 28.22 per cent. silica. The deposits of the Gabilan are in the form of cementing in irregular cracks through the pyroxenite rocks, the thickness varying from knifeblade to 11 in. Analysis showed 59.9 per cent. Mn 25 Fe, 1.5 silica, no lime. The brecciated area extends one mile and for 500 from the shore line; the shore line was strewn with many boulders of ore. The amount of pure ore in any volume of the rock is estimated at 2 1/2 to 4 per cent. but as it has not been worked this is only an estimate, which ought to be fairly close.[1]

Russia.—The manganese ore occurs in iron beds in upper Eocene formation in limestone on both banks of the Quirella, near Tchiaturi. Pyrolusite, polianite with some braunite, hausmannite and psilomelane are the chief minerals. A bed of 6 to 15 ft. in thickness, having at the bottom a rich layer of 6 in. of pyrolusite, is the general average, which runs 55 per cent. Mn and to 7 per cent. silica. The phosphorus contents is .15 per cent.[2] High transport makes possible considerable competition ores from India and Brazil.

The Causasus is one of the greatest manganese producers, following India, which now ranks first. Export of the ore from Batoum in 1911 were, at prices (7 3/4d. per unit ton) which allow of little profit after dues, taxes, etc., have been paid. The Indian exporter gets 9d per unit ton, and makes a handsome profit out of an ore, which is admitted, is much superior to the Russian sort. There seems to be the usual juggling going on between the big firms and the small men. There is a council of manganese dealers in which the small men have a decided majority. This Council has obtained permission from the Government to levy transport dues on all ore sent by rail from Tchiaturi to the extent of 1/2d. per 36 lb., which in effect works out a step to enable the small men to compete with the large capitalists. As other expenditure is being proposed on behalf of the small man, such as the provision of rail transport within reach of all, water to the mines, and better housing accommodation, all of which facilities the large companies already possess, there is evidence of a pretty fight. The local authorities meanwhile are urging municipalisation as the only means now open to them to prevent the industry from falling entirely into the hands of foreign firms. Added to these diffi-

[1] H. V. Wallace, *M. and Eng. World*, July 15, 1911.
[2] *Mining Journal*, Oct. 15, 1910.

culties, the industry suffers from lack of control, the habitual disregard of boundaries, and the stealing of ore from closed mines. Many of the unworked lands are now in the hands of German firms, and it would seem that the small producer is being placed at the mercy of the Government and the capitalist. The Government contemplates the manufacture of ferromanganese, either by the electric furnace or by ordinary blast furnace, the coal being obtained from the coalfields of Kvarkhel.

According to a report by the council of manganese miners, the quantity of manganese ore mined in the Tchiaturi district of the Caucasus in 1910 was 33,805,900 pd. (about 545,250 tons), or 7.39 per cent. less than in 1909. The producing power of the fields, however, increased by 32 per cent. Of the manganese ore exported from Poti and Batoum in 1910, 137,850 tons were shipped to the United Kingdom, and 256,650 tons to the Netherlands. In 1911[1] of the exports 232,000 tons went to the latter country. In the same year 16,000 went directly to the United States. The results of the industry were most disappointing to the producers; from the outset prices of ore declined, and they remained low throughout the year. It is generally believed that the maintenance of the high railway tariff was responsible for the distressed situation at the mines during 1910. It is estimated[2] that shipments in 1910 were 637,718 tons and 22,592 tons, and in 1911 574,335 and 13,888 tons, respectively from the Caucasus and from Donetz.

Russian manganese ore is bought in Germany, on the basis of 50 per cent. manganese, at 1.10 marks per unit, with 1 mark added or deducted for each 1 per cent. above or below the basis; the sample is dried at 100° C. The maximum limit for phosphorus is 0.15 per cent. and for silica 9 per cent. The price for Russian ore in all European ports ranges from 0.65 to 1.30 marks per unit of manganese, with deductions of 35 to 40 pfennig per ton for each per cent. of silica over 9 per cent.; deductions vary for excess of phosphorus above the allowable limit.

Until recently Russia occupied the leading position in the manganese markets of the world.[3] From 1907 to 1908 the prices of Russian manganese ore from Tchiaturi fell from 6 to 7 d. per unit to 1 1/2 d. per unit and remained at the lower price. In 1908 the position of Russia as a manganese producer changed suddenly, the British Indian production amounting to 657,000 tons, while that of Russia remained 362,000 tons. While the amount of ore consumed in metallurgical factories remained the same, there was a falling off in trade as shown by exports, in 1907, 657,000 tons, in 1908, 50,000 and in 1909, 75,000 tons. Toward the

[1] British Consul, Batoum.
[2] British Consul, Batoum.
[3] E. DeHautpick, *Mining Journal* (Lond.), March 11, 1911.

end of 1910, the freight rates became so high that shipping ceased. The position also of the two producing districts, Tchiaturi in the Caucasus and Nikopol on the Dnieper river, was changed, the former having produced 80 per cent. before 1910 and the Dnieper 20 per cent. Nikopol produced 105,000 tons in the first six months of 1911. The amount of ore sent by land to Germany was also much increased. A special consul was called by the Government to save the industry. The freight rate from Poti to Rotterdam had risen to 14 s. per ton thus making the charge on Caucasian manganese ore .000113 cent per pound-mile. Comparing this with India, the corresponding freight rate was .000214 cent and with Brazil it was .000197 cent per pound-mile. The Indian ore is carried 491 1/2 miles at a total cost of .105 cent per pound. As a matter of fact, the price paid is not sufficient to work the mines. The explanation of some of the low markets is that most of the manganese ore is used by large steel interests and the scale of operation of the manganese industry on the part of the poor natives either in Brazil, Russia, or India, is such that they cannot rise to protest against the loss of their livelihood. Aside from the disorganization of trade and exports, a feature of the industry is the irregular exploitation of the mines. The ore extends over 80 or 100 miles and occurs in beds, in some cases hundreds of feet thick. The working is based only on the extraction of the rich parts of the mines by such an imperfect system that a large part of the ore is rendered useless for further handling. Only the large concerns have control over the work, but the great bulk of the small contractors work in a primitive way without any sorting or cleaning of ore. Hauling to the railroads is done in two-wheel carts along very bad roads. Exploration is done only periodically. The very best part of the station of Tchiaturi, the loading area, is covered over with loads of ore. This has long been covered with grass, although at first was considered a reserve. In the shipping port of Poti where there is a large space for storing ore, the steamers have to wait the arrival of cars at the time of loading. As the trade does not depend upon the seasons, the demands have been fairly regular throughout the year. A movement to regulate the exports will help to recuperate the industry. For this purpose a regular steamship service should be effected by the union of manganese industrialists. The low price might also be regulated by manufacturing ferromanganese, and shipping the same. Coal for this purpose exists near Poti in the Tkvartchely coal mines.

Spain.—The manganese mines in Huelva, Spain, are only small producers of the ore. In 1908 they produced 22,069 metric tons, in 1909 it is estimated they produced 6000, and in 1910, 7000 tons. A full de-

scription of these deposits is found in *Zeitschr. für Praktische Geologie*, 1911.

Turkey.—It is estimated that 18,000 metric tons of manganese ore were produced in Turkey in 1911. For prices of Turkish manganese ores the basis is 45 per cent. manganese, with not more than 0.03 per cent. phosphorus, nor more than 10 per cent. silica, and the price is about 95 pfennig per unit in Germany, or somewhat higher for ores of better grade; deductions for silica and phosphorus are subjects of negotiation.

It has been thought that the operation of mines in the province of Aidin under European auspices would compete with the ores of the Caucasus. As the ore is pockety and in limestone, it is doubtful that the development will prove sufficient.

Uruguay.—In an article on the resources of Uruguay, in *Bergwirtschaftliche Mitteilungen*, September, 1910, C. Guillemain describes the occurrence of workable deposits of manganese ore at Zapucay, near Corales, in the department of Rivera. The principal showings are in two ridges, Cerro Iman and Cerro Papageio. The original source of the ores is a quartz rock bedded among metamorphic slates, and charged with braunite and manganiferous hornblende. A few zones in the original rock consist of almost pure iron-manganese minerals, but the main body of that rock is silicious. Secondary deposits, however, are found nearby, containing ore of better quality. In order to develop them, it would be necessary to build a railroad of considerable length, or a canal, utilizing the rivers Negro, Uruguay and Takuarembo as means of transport. Deposits have been observed in other parts of the country, but no workable ones have yet been developed.

Uses of Manganese Ore

Manganese ore is used principally in the manufacture of steel, where it is applied either in the form of speigeleisen or ferromanganese, or indirectly in the manufacture of pig iron. It is also used for the manufacture of chlorine and a number of alloys. A number of pigments are made, such as umber, for example, at the mines Cypra, Turkey; Thuringia, Sicily, etc. Dutch umber is obtained by burning the brown umber. Green manganous oxide pigments and also violet ones are produced. A preparation of some of these as driers in oils is also made use of in the paint industry. Permanganates are also used extensively in the chemical industries.

Alloys.—The most important alloys are spiegel and ferromanganese. In 1856 Mushet found that a good quality of steel is produced by the addition of spiegel after the complete removal of carbon in the converter.

Bessemer, Henderson and others tried to make ferromanganese, but failed as far as a commercial product goes. The spiegel used by Mushet contained 10 to 12 per cent. manganese and 5 per cent. carbon. In 1836, spiegeleisen was first made in the blast furnace and contained 22 per cent. manganese. About 1870, three blast furnaces in Europe made ferromanganese, namely, Shyshytta in Sweden, 20 per cent.; Reschitza in Banat, 69.6 per cent. and the works of the Krainische Industriegesselschaft in Sava and Jauerburg which made speigeleisen 8 to 14 per cent. and ferromanganese 33 per cent. manganese. This material was also made at other works in France and England. Today there is no difficulty in making 90 per cent. metal.

Ferromanganese, which is added as a deoxidizing and dephosphorizing agent to steel baths to the extent probably of 1/2 per cent., can be added in the solid form or liquid. The cooling of the bath and mechanical losses, as well as slowness in acting, make this method objectionable. When the ferromanganese is first melted, it is estimated that with ordinary methods, there is a loss by oxidation. Experiments made at the works of Keller-Leleux, in Livet, France, with a Keller electric furnace, German patents 194,897, 218,054, and 219,575, prove that no oxidation takes place in premelting the ferromanganese in this furnace. The molten material can therefore be obtained in the best form for efficient work and it is estimated that 35 per cent. of the ferromanganese is saved over ordinary methods. The furnace in use continuously at Saarbrucken is rigged to smelt steel part of the time, as the amount of ferromanganese required is small. Also small pieces of ferromanganese could not be added in converter practice, but are saved by melting the metal. The average price of ferromanganese in the last 10 years has been $51.10 per ton in Europe, and is not likely to rise.

The carbon in ferromanganese is all combined and is not present in the form of graphite. An analysis of material recently imported is manganese, 93.9; silicon, 1.6; iron, 2.3; and carbon, 1 per cent.

The presence of over 1 per cent. of manganese makes steel brittle. The brittleness increases with the addition of more manganese until, when 4 to 5 1/2 per cent. is present, the steel can be crushed under a hammer. With further additions of manganese, up to 7 per cent., the brittleness does not increase. Upon addition of as much as 7 per cent. manganese an entirely new set of properties is acquired by the steel; these new properties are marked at 10 per cent. and reach a maximum at 12 or 15 per cent. manganese. Steel containing 12 to 15 per cent. manganese possesses great hardness and ductility.

Manganese steel is made by Hadfield's Steel Foundry Co., Ltd., Sheffield, England; the Edgar Allen American Manganese Steel Co.,

Chicago, Ill.; and the Taylor Iron and Steel Co., Highbridge, N. J. It is specially suitable for casting wearing parts of grinding, and elevating and crushing machinery. It is difficult to cut or machine and requires grinding. It casts well without blowholes and forges well at a moderate temperature. It cannot be magnetized. Manganese steel may be rolled hot into sheets as thin as No. 18 gage, .049 in., without special difficulty; it has been rolled to 28 gage, .014 in., with frequent annealing. It has been rolled cold into much thinner sheets, but it hardens quickly in cold rolling and requires frequent annealing. It has been drawn into wire .0105 in. thick No. 30 American wire gage.

The imports of ferromanganese into the United States for the year 1911 were 80,263 tons valued at $3,015,062. Spiegeleisen amounting to 20,971 tons, valued at $405,444 was also imported, the duty amounting to $2.50 per ton.

Alloys of manganese, such as German silver in which part of the nickel is replaced by manganese, have been used for forks and spoons. Phosphorbronzes in which manganese replaces part of the phosphorus, are made by adding a copper alloy containing a few per cent. of manganese. A small quantity of this alloy suffices to remove the oxygen from the metal and the excess manganese does not injure the quality of the resulting bronzes. Such bronzes are called manganese bronzes. By fusion of manganese chloride with aluminum, Mn_3Al is produced. This is called manganese aluminum and it is hard enough to scratch glass. A softer alloy is made with magnesium.

MICA

The value of mica produced in the United States in 1911 was $355,804. In 1910 the corresponding value was $337,097 and in 1909, $280,929. A maximum total production of mica occurred in 1907 when it reached a value of $392,111. Sheet mica which constitutes the greater part of the value, was produced to the extent of 1,887,201 lb. valued at $310,254, as compared with 2,476,190 lb. in 1910 valued at $283,832. The scrap mica produced amounted to 3,512 tons compared with 4065 tons in 1910. The value of this material in 1911 was $45,550 and in 1910, $53,265. These figures show a falling off in the production as some of the largest consumers and miners of mica did not do any mining in 1911. The important producers were the same states as heretofore, North Carolina and South Dakota being the principal ones. The mines of the Westinghouse company which were taken over in 1910, continued to produce about the same amount per month as before. It is estimated that the production of the mine, which is near Custer, South Dakota, amounts to something like 4000 or 5000 lb. per week, in addition to as much more in value in other kinds. This company was therefore the chief producer in 1911.

STATISTICS OF MICA IN THE UNITED STATES.

(In pounds and tons of 2000 lb.)

Year.	Production. (a)			Imports.			
	Sheet. (b)	Scrap.		Unmanufactured.		Cut or Trimmed.	
	Pounds.	Tons.	Value.	Pounds.	Value.	Pounds.	Value.
1898	110,918	3,529	$39,837	877,930	$115,930	78,567	$34,152
1899	97,586	6,917	50,596	1,709,839	233,446	67,293	42,538
1900	127,241	5,417	42,889	1,892,000	290,872	64,391	28,688
1901	360,600	2,171	19,719	1,598,722	299,065	78,843	35,989
1902	373,266	1,400	35,006	2,149,557	419,362	102,299	46,970
1903	619,600	1,659	25,040	1,355,375	288,783	67,680	29,186
1904	668,358	1,096	10,854	1,085,343	241,051	61,986	22,663
1905	851,800	856	15,255	1,506,382	352,475	88,188	51,281
1906	1,423,100	1,489	22,742	2,984,719	983,981	82,019	58,627
1907	1,060,182	3,025	42,800	2,227,460	838,098	112,230	77,161
1908	972,964	2,417	33,904	497,332	224,455	51,040	41,601
1909	1,809,582	4,090	46,047	1,678,482	533,218	168,169	85,595
1910	2,476,190	4,065	53,265	(c)1,972,221	721,541		
1911	1,887,201	3,512	45,550	(c)1,467,688	469,089		

(a) Statistics for 1901 to 1911 inclusive are those of the U. S. Geological Survey. (b) The value of sheet mica being so widely variable, and so little indicative of commercial results, and all previous statistics being of doubtful accuracy, they have been omitted from this table. (c) Includes cut or trimmed.

Imports.—The imports of mica during 1911 amounted to 1,467,688 lb. valued at $469,089. In 1910 the total imports were 1,972,221 lb. valued at $721,541 and in 1909, 1,527,444 lb. valued at $493,978. The mica imported from Canada amounted to about 10 per cent. of the imports. A large percentage of the imports came from India.

In recent years the imports of sheet mica into the United States have run close to the quantity produced at the domestic mines.

The consumption of sheet mica in this country, particularly in the electrical industry for insulating dynamos, motors, switch-boards, induction apparatus employing high voltage, etc., and to a smaller extent in stoves, gas lamp chimneys, lamp shades, etc., amounts to about 3,500,000 lb. annually.

Ten years ago, the consumption of sheet mica in the United States was about 2,000,000 lb., and ten years previously, in 1891, the total approximated 175,000 lb. For over a quarter of a century, the domestic demand for sheet mica was satisfied largely by imports, chiefly from India, although in recent years the receipts from Canada have multiplied.

Imports fell below the domestic production of sheet mica for the first time in 1908. The variation shown in imports in 1911, as compared with previous years, may be explained by the general restriction in manufacturing industries in which mica is an important mineral of utilization.

A small quantity of foreign mica is reexported and on the quantity that remains for domestic consumption importers have to pay a duty of 5 cents per pound and 20 per cent. *ad valorem* for rough mica (uncut or untrimmed) and 10 cents per pound and 20 per cent. *ad valorem* for manufactured mica, that is mica ready for use in the various industries, such as mica plates, built up mica or cut or trimmed. Ground mica pays an import duty of 20 per cent.

It is noteworthy that the domestic consumption of scrap mica produced at the mines and factories has grown appreciably in recent years, and now amounts to something like 4000 or 5000 short tons.

This expansion in the utilization of scrap mica is due largely to the fact that aside from the profitable business of reconstructing small sheets into larger, more serviceable ones, used in electrical apparatus, pulverized mica finds ready sale among the manufactures of wall paper, paints, lubricants, and numerous other minor articles.

London Mica Markets.—In 1910 the year closed with limited stocks of block mica. Supplies were drawn almost entirely from the presidencies of Calcutta and Madras at which latter a number of sources of supply were closed. During 1910 there were also a few new sources

of supply from Africa and Japan. The quality of Japanese amber mica was hardly satisfactory.

According to Baker and Startin of London, the mica market during 1911 showed an advance in price of 25 per cent all around toward the close of the year. No material changes in values are expected for some time to come. India, as usual, furnished the bulk of the shipments but Canada, Africa and South Africa contributed. The European demand has been big throughout the year and trade with the United States is slowly improving. The stocks of unsold mica at the close of the year consisted chiefly of Bengal splittings. The position in the various grades is as follows: Steady advance in all qualities of Calcutta except in the stained grades; a very limited supply of Madras and marked increase in medium sizes. There was a fair demand for splittings but weak prices. Africa amber mica sold at steady rates. The following figures are compiled from the returns of the London warehouses.

	Arrivals cases	Deliveries cases	Stocks Dec. 31.
Cases (averaging 1 cwt. each), 1911	23,236	20,972	7,502[1]
Cases as compared with 1910	18,995	22,841	5,238
Cases as compared with 1909	19,929	21,878	10,174

[1] Including purchases uncleared.

Canada.—Canada produces about one-fourth of the world's supply of mica. The value of the crude mica produced during 1911 was $119,-863 compared with $190,385 in 1910. The table gives the exports, showing the large percentage that went to the United States. Of the total production Ontario furnished 322 tons, valued at $43,058 and Quebec product valued at $76,433. Corresponding values in 1910 were $85,294 and $51,901. In Quebec, better ruling prices have caused

STATISTICS OF MICA IN CANADA.

Year.	Production.	Total Exports.	Exports to U. S. (*a*)	Year.	Production.	Total Exports.	Exports to U. S. (*a*)
1902	$135,904	$391,812	$184,287	1907	$312,599	$422,172	$596,321
1903	177,857	196,020	196,470	1908	139,871	198,839	(*b*)140,166
1904	160,777	198,482	137,191	1909	147,782	256,834	(*b*)137,383
1905	178,235	179,049	121,560	1910	143,409	299,107	(*b*)287,048
1906	303,913	581,919	328,991	1911	119,863	309,546	(*b*)308,851

(*a*) Fiscal year ending June 30. (*b*) Fiscal year ending March 31.

renewal of mining activities and the shipments in 1911 were 50 per cent. higher than in 1910. Mica is mainly mined in the valley of Gatineau and the Liévre.

Since 1905 when a monograph was issued on Mica in Quebec and Ontario, mica mines have been located north of the districts covered at that time, while from recent investigations by Hugh S. de Schmid,

it would appear that the mica-bearing region extends further in that direction than the prospected area.[1] The mica near Notre Dame De Laus, is of a dark amber color and of only medium splitting quality. In the Gatineau region the mineral has been mined as far north as range 2, lot 28, Egan township, where several thousand dollars worth of fair amber mica have been taken out. The most westerly mica deposits so far worked in the area under consideration are situated in Litchfield and Huddersfield townships.

The mica in Argenteuil County, Wentworth Township, as in the Saguenay region, Berthier County, still farther to the east, is chiefly of the muscovite variety.

The chief centers of mining activity in Ontario are Sydenham and Micaville, in the townships of Loughborough and North Burgess, respectively.

In the former district the General Electric Company's mine—the old Smith & Lacey—still remains the chief producer and employs an average force of 35 men. The output and reserves of this mine play an important part in fixing the market price of the mineral, and the General Electric Company, which carries on its mining operations under the name of the Loughborough Mining Company, and owns several other mines, practically controls the market.

With the exception of some half dozen, all the mines are phlogopite or amber mica. The color of the mica ranges from almost black to almost colorless, the darkest and the lightest varieties being as a rule the poorest splitters.

The suitability of mica for electrical purposes depends essentially on its degree of cleavage and its non-conducting properties. Consumers vary in their opinions, some preferring relatively dark mica of medium splitting quality, while others refuse to take any but the light-colored variety, on account of its apparent freedom from iron, though its cleavage may be much inferior to that of the dark colored. Brittleness is also an important consideration, though this fault is not so detrimental as formerly, as with the advent of mica-board manufacturing the most brittle mica is sufficiently flexible in thin layers (1/2 to 1 mm.) to be used in the building up of mica plate.

Market conditions could be much improved by the establishment among mica dealers of a satisfactory system of sorting.

India.—There is again an apparent reduction in the output of mica. The total production of India is given in the accompanying table up to 1911. The largest production, more than half, comes from Bengal and about one-third comes from Madras. The majority of these mines are

[1] *Summary Report*, Mines Branch of the Canadian Department of Mines, 1910.

under the control of the Indian Mining Act of 1901. In a period of 15 years the average number of persons employed was 15,667—about the same number as was employed in the gold mining industry. India produces nearly two-thirds of the world's mica supply. In 1910, 32,699 cwt. were produced compared with 32,903 cwt. in 1909, but the value rose from £38,157 in 1909 to £54,427 in 1910. In spite of the great apparent decrease in production, the exports in 1910 amounted to 39,612 cwt. valued at £177,152. It is believed that the output returns were not complete. During 1910 the number of persons employed in this industry was 14,073.

PRODUCTION OF MICA IN INDIA FOR THE YEARS 1905 TO 1910. (a)
(1 cwt. = 112 lb.)

Province.	1906.	1907.	1908.	1909.	1910.
	Cwt.	Cwt.	Cwt.	Cwt.	Cwt.
Bengal	22,360	28,579	36,060	22,084	18,356
Madras	24,420	15,865	11,249	8,948	3,586
Rajputana	5,763	7,759	6,234	1,871	757
Total	52,543	52,203	53,543	32,903	22,699

(a) *Records*, Geol. Surv. India, Vol. XXXIX.

The mica mined in India[1] belongs entirely to the variety known as muscovite. The two principal areas of production are the Nellore district of the Madras presidency and a section about 12 miles broad and 60 miles long in Bengal. In 1898 the Indian mica miners discovered that the waste dumps were suitable for the manufacture of micanite. For this purpose thin films are cemented together and molded in sheets and in many cases can replace the natural sheets. In the mica industry, the clear sheets are split into thin films by children who have become experts in selecting the required thicknesses. The industry is more advanced in Nellore than it is in the mines of Bengal. The latter are worked on practically the same plan used 100 years ago by native workers. Mica is generally discovered in the exposed outcrops and followed up by underhand stoping. These cork-screw workings extend 300 ft. below the surface. Water is dipped out in buckets. The daily wages of the natives is from 2 to 10 annas, or 4 to 20 cents; nevertheless, it would appear that on a modern system of mining, the mines would pay better.

Ceylon.—Although it is known that most of the mica is of special value, the methods of mining it have been on a small and primitive scale thus far.

[1] W. H. Michael, *Mining and Engineering World*, Jan. 20, 1912

AVERAGE ANNUAL DISTRIBUTION OF INDIAN MICA EXPORTED DURING THE YEARS 1903 TO 1908 INCLUSIVE. (a)
(1 cwt. = 112 lb.)

Exported to	Average Quantity.		Average Value.		Value per cwt. £
	Cwt.	Per cent. of total.	£	Per cent. of total.	
United Kingdom	17,226	52.8	102,307	61.9	5.94
United States	4,781	14.7	29,497	17.8	6.17
Germany	7,391	22.7	21,337	12.9	2.89
Belgium	1,050	3.2	3,551	2.1	3.38
France	558	1.7	2,497	1.5	4.47
Other countries	1,599	4.9	6,214	3.8	3.89
Average Total	32,605	100.0	165,403	100.0	5.07

(a) *Records*, Geol. Surv. Ind., Vol. XXXIX.

WORLD'S PRODUCTION OF MICA. (a)
(Five-year periods.)

Period.	Canada.	India.		United States.	Total.
	Value.	Value.	Per cent.	Value.	Value.
1894–1898	£72,991	£316,016	68.08	£75,147	£464,154
1898–1903	160,552	426,851	60.14	122,382	709,785
1904–1908	285,351	867,553	61.78	251,299	1,404,203
Total	£518,894	£1,610,420	62.50	£448,828	£2,578,142

(a) *Records*, Geol. Surv. India, Vol. XXXIX.

United Kingdom.—The United Kingdom produced 23,927 tons of mica in 1910 as compared with 21,743 tons in 1909. It is obtained in the preparation of China clay as a by-product, mostly in Cornwall and also in Devon.

MINERAL WOOL

The production of mineral wool in the United States during 1911 was only 7514 short tons valued at $65,500. This is about 880 tons less than in 1910. About one-third of the output was slag wool and the remainder rock wool. The average price for both kinds at the factory or works was $5.72 as compared with $9.99 per ton for the previous year. Business conditions are reported as fair; profits are low and competition keen. No new producers are reported for the last 2 years.

PRODUCTION OF MINERAL WOOL IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Amount.	Value.	Per Ton.	Year.	Amount.	Value.	Per Ton.
1900	6,002	$60,320	$10.05	1906	5,375	$55,550	$10.33
1901	6,272	68,992	11.00	1907	9,008	81,769	9.08
1902	10,843	105,814	9.67	1908	9,197	77,228	8.40
1903	(a)			1909	11,626	101,621	8.74
1904	(a)			1910	8,408	84,012	9.99
1905	6,164	69,560	11.28	1911	7,514	65,500	8.72

(a) No statistics collected.

The following concerns are the principal manufacturers of mineral wool: United States Mineral Wool Company, 140 Cedar Street, New York; Pennsylvania Mineral Wool Company, Norristown, Penn.; Banner Rock Products Company, Alexandria, Ind.; Hoosier Rock Wool Company, Alexandria, Ind.; Union Fiber Company, Yorktown, Ind.; Columbia Mineral Wool Company, 112 Clarke street, Chicago, Ill.

As stated above, there are two kinds of mineral wool made from two very different substances. One is made from blast-furnace slag and the other from a fusible rock, such as siliceous limestone. Most of the producers use the latter material in manufacturing products, according to the properties required. At the factory of the Union Fiber Company, at Yorktown, Ind., rock wool is manufactured from limestone. This is broken into pieces varying from 3 in. to 12 in. in diameter, and placed in a furnace consisting of a steel water-jacketed cupola 5 ft. in diameter and 20 ft. high. A certain percentage of coke is charged and the whole is burned under a fierce draft, thus melting the charges. The molten glass is generally of a light yellow color and runs through the tap-hole of the furnace, directly in front of a powerful steam blast, impinging on it through very narrow apertures. The steam blast blows the molten stream very suddenly into fine particles which are the mineral rock

wool of commerce. Comparative analyses of this material and some iron slag mineral wool are given in the following table:

	Rock wool.	Iron slag mineral wool.
Silica	42.84	41.70
Alumina and iron oxide	1.40	19.46
Lime, CaO	51.74	28.90
Magnesia MgO	2.75	7.02
Sulphhric acid	None	1.68
Undetermined	1.27	1.24
Total percentage	100.00	100.00

It will be noted that in the rock wool there is no sulphuric acid. The rock wool is more nearly a satisfied silicate, whereas the slag wool is a complicated mixture in which sulphates and sulphides appear to be present. The rock wool is lighter and more elastic, having a longer fiber and showing a marked difference in the weight of a given bulk, thus packing less densely. This makes the rock wool a better non-conductor. When fused, the rock wool remains white or nearly so, while the slag wool generally becomes dark on account of the sulphur and iron present. The above analyses were made by Regis Chauvenet and Brother.

Mineral wool is used extensively for insulating purposes. It is packed into hollow spaces in walls of cold storage buildings and refrigerators of various types. It is used as a strainer in copper and zinc works. While the product enters, to a certain percentage, into the manufacture of insulating materials at the present day, there are to-day higher grades as, for example, in board form. The uses of mineral wool are therefore not expanding but it can be obtained on the markets in the crude form, although most of it enters into manufactures.

MOLYBDENUM

There is no large demand for molybdenum ores in the United States although offers are received from Europe at intervals. In recent years there were no molybdenum deposits in the United States worked on a commercial scale. Molybdenite occurs in the United States in Arizona, Maine, Oregon, Nevada and Washington. It also occurs in foreign countries especially in Canada, Germany, Japan, Mexico, New South Wales, New Zealand, Norway, Peru and Queensland. The lead molybdate or wulfenite, usually contains impurities making it unsuitable for the extraction of metallic molybdenum. The chief use of molybdenum is in steel making in the electric furnace. The product is put on the market as iron-molybdenum alloy but on account of the competition of tungsten and also on account of small and uncertain ore supplies, it does not find extensive use. The principal purchasers in the United States are the Electrometallurgical Company of America, New York; Primos Chemical Company, Primos, Penna.; Atkins, Kroll & Co., San Francisco, Cal.

Most of the consumers ship the ore for treatment to Europe. The requirements are various, ranging from 50 per cent. molybdenite to 90 per cent., the latter being most usual. Prices paid are $6.50 to $7.50 per unit of molybdenum sulphide per long ton. In Europe, the principal buyers, or users, are Friedrich Krupp of Essen, Germany, and Paul Girod of Savoie, France, but there are other large consumers.

Maine.—The Holman molybdenum mine at Catherine Hill, Maine, consists of a deposit in granite. Though low grade, this deposit warrants the expectation that it could furnish a source of supply especially with the improvements in concentration methods which have lately been made. Many samples of the ore have given values, representing large quantities of 1.5 per cent. molybdenite. The attempts at wet concentration have not been successful although in some cases, concentrates assaying 3 to 4 per cent. have been readily obtained, by former methods. The owner of this mine is C. Vey Holman, Bangor, Me. Some of the assays of this ore show 1.5 oz. of silver per ton. There are several zones in which this ore runs higher than the average but the eventual working on an economical scale will require treating the whole from which the product will be concentrated.

Canada.—A paper by T. L. Walker, being a report to the Canadian Department of Mines, treats the occurrence of molybdenum ores very fully.

Five types of deposits are mentioned; granite, or pegmatite, carries quantities of minerals as those at Romaine in Quebec and Glengarry in Cape Breton. Quartz veins which contain more or less feldspar also carry molybdenite as at Jordan Falls, Shelbourne County, N. S. and Montcerf, Ottawa County, Quebec. In a few instances granitic rock, intersected by series of joint planes, appear to have been traversed. Fluorspar, quartz and molybdenite have been scattered in the granite, some distance from the joints. This type of deposit is best represented by ore bodies at Cooper and Catherine Hill in Maine. Massive and very fine grained molybdenite occurs in another kind of deposit which also carries copper. These are represented by the Giant Mine, Rossland, B. C., the Marble Bay Mine, Texada Island and the Harvey Hill Mine, near Broughton, Quebec. Molybdenite also occurs associated with pyrrhotite and pyrite at the contact of granite with limestone. Such a deposit occurs at Lynedoch township in Renfrew County and is not infrequent in the Ottawa vicinity. The preparation of crude ore for market is a difficult problem in ore dressing.

As the mineral is soft and the gangue very hard, large losses result by sliming. The larger flakes resist the action of the rolls and may be separated by screening, but some scales containing molybdenum float away with wet treatment. Dry concentration has been employed, as at Cooper, Maine. Another process which appears very well suited for these ores is that of F. E. Elmore.[1] This is the same oil process in which the oil has a selective action for the metallic mineral particles especially in the presence of an acid.

Much can be done by hand picking as is always the case with difficult ores. (Prof. J. B. Porter.) It would seem as if a reasonable combination with dry screening process for the removal of coarser flakes, followed by a treatment of the tailings by the vacuum process, should give results better than those obtained by either of these methods alone.[2]

Much of the world's production of molybdenum ores comes from New South Wales and Queensland where molybdenum bearing quartz is hand picked, crushed through rolling and then jigged. The oversize is then recrushed, hand picked again and finally treated on Wilfley tables. Impurities in the production may be even small quantities of copper, arsenic or bismuth and these may render the ore less suitable for steel manufacture.

Uses of Molybdenum

Until recently, molybdenum was required only for the manufacture of chemical preparations, principally ammonium molybdate, which is

[1] *Engineering and Mining Journal*, May 11, 1907.
[2] A. B. Frenzel, *Mining Science*, Jan. 25, 1912.

much used in the determination of phosphoric acid. The iron and steel works of the United States alone are said to consume as a chemical reagent several tons of molybdenum annually. Molybdic acid is used as a blue pigment in the manufacture of porcelain, and for dyeing silks and woolens. In various compounds molybdenum is used for coloring leather and rubber. Ammonium molybdate is said to be used for fireproofing; also as a disinfectant for cloth used in railway passenger carriages.

The chief use to which most of the world's molybdenum is devoted is the preparation of special varieties of steel. The result of the addition of a small proportion of metallic molybdenum to steel is very similar to that obtained by the use of tungsten. Molybdenum is more potent than tungsten, and only about half as much of the molybdenum is necessary as would be required were tungsten used instead.

Molybdenum steel is used for rifle barrels, propeller shafts, guns, wire, and particularly for the manufacture of high speed tools. Molybdenum high speed steel contains from 8 to 10 per cent. molybdenum. When the other elements exist in the right proportion, a steel is obtained of great hardness, with the peculiar property of retaining its temper when heated to a high degree. Owing to this property, it is possible to take extremely heavy cuts at high speed, the tool often being heated through this hard use to a dull red heat without impairing its usefulness.

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MONAZITE SAND

By George Letchworth English

Monazite sand is the principal source of the rare earths, ceria and thoria. The mineral monazite is a cerium phosphate in which part of the cerium is replaced by thorium, lanthanum, neodymium and prascodymium. The most valuable constituent is the thoria, the percentage of which determines the value. The principal producer of monazite is Brazil. Up to 1909 there was a considerable domestic output in the United States entirely from the Carolinas, but the industry established under a protective tariff would not pay when competing with the foreign product with the tariff reduced from 6 to 4 cents per pound. There are also deposits in Idaho and elsewhere but no shipments are reported. In 1911, the production of monazite in the United States was practically nothing, all former producers being idle, except one company working for the purpose of keeping leases in force. The table gives the production of monazite for 18 years and imports of thorium products for 10 years. The demand for the foreign thorium products has always been great when the relative value of the imports is compared with the value of the domestic production. The greatest production of monazite in the United States took place in 1895 when over 1,500,000 lb. valued at over $137,000 was produced. The value of a somewhat smaller amount of product in 1905 was greater, being $163,908. In late years the production reported in the table is refined sand, thus making it appear that the value per pound has increased in the course of 20 years. The recent production of monazite sand has been valued as 12 cents per pound and it is estimated that the 1911 production was equal to .268 of the weight of crude sand. The extinguishing of this industry has seriously affected a large number of farmers that owned or leased the placers. In recent years, since 1909, monazite sand has been imported in addition to thorium nitrate and the import value is as low as 8 2/3 cents per pound. The rate of duty is now 4 cents per pound. The duty on thorium nitrate is 40 per cent. ad valorem, the import value per pound being $2 approximately.

The former largest producer, the Carolina Monazite Company, was idle in 1911. There was no activity in North Carolina excepting the sale of a few hundred pounds at Cleveland Mills and probably the local representative of the German American Monazite Company bought a few small lots of sand during 1911.

MONAZITE PRODUCTION IN THE UNITED STATES.

Year.	United States. (a)			North Carolina (b).		
	Pounds.	Value.	Per Pound.	Pounds.	Value.	Per Pound.
1894	546,855	36,193	0.066	364,570	24,129	0.066
1895	1,573,000	137,150	0.087	1,048,667	91,433	0.087
1896	30,000	1,500	0.050	20,000	1,000	0.050
1897	44,000	1,980	0.045	29,333	1,320	0.045
1898	250,776	13,542	0.054	167,184	9,028	0.054
1899	350,000	20,000	0.057	233,333	13,333	0.057
1900	908,000	48,805	0.054	605,333	32,537	0.054
1901	748,736	59,262	0.079	499,157	39,508	0.079
1902	802,000	64,160	0.080	534,667	42,773	0.080
1903	862,000	64,630	0.075	773,000	58,694	0.076
1904	745,999	85,038	0.114	685,999	79,438	0.116
1905	1,352,418	163,908	0.121	894,368	107,324	0.120
1906	846,175	152,312	0.180	697,275	125,510	0.180
1907	547,948	65,754	0.120	(c)456,863	54,824	0.120
1908	422,646	50,718	0.120	(c)310,196	37,224	0.120
1909	541,931	65,032	0.120	(c)391,068	46,928	0.120
1910	99,301	12,006	0.121	83,454	10,104	0.121
1911	3,561	427	0.120	400	48	0.120

(a) Statistics are those of the U. S. Geological Survey. (b) Statistics for N. C. from 1893 to 1902 are estimates; those from 1903 to 1906 are from "The Mineral Industry of North Carolina"; those for 1910 are furnished by Joseph Hyde Pratt, State Geologist. (c) Statistics for 1907, 1908, and 1909 were collected jointly by the U. S. Geological Survey and the N. C. Geological and Economic Survey.

IMPORTS OF THORIUM NITRATE AND MONAZITE. (a)

	Year.	Pounds.	Value.	Per Pound.
Thorium nitrate entered for consumption	1902	42,815	$131,350	$3.07
	1903	64,520	232,155	3.59
	1904	58,655	249,904	4.26
	1905	52,378	269,504	5.14
	1906	40,090	139,929	3.49
	1907	51,441	152,666	2.96
	1908	65,289	173,239	2.65
	1909	125,088	236,057	1.89
	1910	108,597	218,713	2.01
	1911	121,111	238,841	1.97
Monazite sand (and thorite) general imports	1909	661,880	57,545	0.0869
	1910	2,427,732	217,360	0.0895
	1911	1,781,290	157,229	0.0883
Monazite sand (and thorite) entered for consumption.	1909	110,088	9,543	0.0867
	1910	453,554	39,699	0.0875
	1911	705,149	60,542	0.086

(a) Statistics from 1902 to 1908 are those of the U. S. Geological Survey; those for 1909 and 1910 were furnished by the Bureau of Statistics of the Department of Commerce and Labor.

Supplies and Prices.—The Thorium Convention in 1909 tried to control the industry but to-day this syndicate exists only in form and is ineffective. The relatively high price at which certain German firms had to purchase monazite sand allowed English competition, until the former did not rely much on the thorium production for their profit but have developed other industries such as the osram lamp, etc. In 1894, the price of thorium nitrate was about $500 per kilogram, whereas to-day it is less than $5 per kilogram in Europe. Prices have steadily

declined and as mentioned by the London correspondent of the London Financial Times, are not unlikely to go considerably lower. The markets being glutted with monazite sands at the syndicate's high prices, the users, such as the Welsbach Incandescent Light Company and the German Incandescent Gas Company were forced to obtain independent supplies regardless of costs. In this they were successful and prices fell rapidly. The consumption of monazite sands for the manufacture of thorium compounds is from 2000 to 2500 tons per annum, but the supply has been for years, double the consumption. A new lease of the Brazilian fields is about to be effected between the Government and the producers. The Brazilian Government is not likely to allow any monopoly to obtain control of the supplies, as the resulting flooding of the markets and consequent decline in prices is feared.

Monazite furnished the raw material also for the metal cerium, in addition to thorium. Originally the mineral thorite (50 per cent. thoria) from Norway and Sweden was used for thorium, but the supply was too limited to allow the development of the gas mantel industry on the present scale. The methods for treating thorite had become well established, however, and proved quite simple for the manufacture of gas mantles, etc. Monazite sand was at first a waste from certain gold washings and soon became the raw material when its supply was found to be certain. At the same time the industry was revolutionized because the methods of extracting thoria, of which there was only 5 to 6 per cent. in the sand, were entirely different than with thorite.

Brazil.—The deposits, especially those of the southwest coast of the State of Bahia, are of great importance. The town of Prado was the starting point of the large export trade but later on sands were also shipped from other coast districts such as those in the State of Espirito Santo. The sands are concentrated in the beaches and these were the first deposits mined, but later the sands were also found in bars inland. These latter are along the rivers where the monazite sand commonly occurs in layers from 4 in. to 4 ft. thick and is covered by peat and barren soil. On an average the monazite sands contain 62.7 per cent. Ce, 1.5 to 3.5 per cent. Th, 1 to 3 per cent. Yt, 2.5 per cent. La, 2.5 per cent. Fe, and 3 per cent. Al. The thoria ranges from 1 per cent. to 20 per cent. and must be 3 per cent. or over to be of commercial value.

The concentrating treatment of monazite sands is described by F. Freise.[1] The difficulty is in the fineness of the washed sand to be treated. A large average sample taken from about 200 different deposits was examined, the result being that only about 11 per cent. consists of grains more than 1 mm. in diameter. The analysis is as follows:

[1] *Oestr. Zeit. f. Berg u. Hüttenwesen*, 27, 1911.

Larger than 2.5 mm.	1.7 per cent.
Larger than 1.5 mm.	3.4 per cent.
Larger than 1.0 mm.	6.5 per cent.
Larger than 0.7 mm.	11.8 per cent.
Larger than 0.4 mm.	25.2 per cent.
Larger than 0.2 mm.	16.4 per cent.
Less than 0.2 mm.	34.6 per cent.

The monazite sand is separated into the colored and the lighter materials like clay and silica. The colored minerals are then further separated. An investigation of the magnetic properties of all the usually accompanying minerals was made, showing that there is sufficient difference to obtain a practical separation of nearly all of them. On the other hand, the slight differences in gravity would prevent wet dressing. The pneumatic process also proved unsuccessful. A process invented by the author is based on the difference in the friction angles, the monazite grains being round, while the grains of the accompanying minerals are angular. According to this process, the material is placed on a suitably constructed inclined plane with cross channels. This process works on sands containing no iron minerals. The Witt friction process, which is based on the difference in friability of the different minerals, can be used only after previous magnetic separation. Two factors that may seriously interfere with successful magnetic concentration are the lack of uniformity in the deposits and also the fact that the usual magnetic proportions may be seriously interfered with by adhering minerals. It would be impossible to mine monazite from the solid rock, granite, syenite or gneisses, profitably. These occurrences, however, are very few and the percentage of monazite is rarely as high as 1/2 of 1 per cent.

The occurrence and treatment of monazite in Brazil is described by Mr. Moravia, Jr.[1] On the disintegration of certain rocks the monazite sand is deposited on places having little slope as found in the levels alongside any stream. On the seacoast the sands are concentrated by the combined action of rivers, rain, wind and sea. The deposits are pronounced near bars, the richest parts of them being nearest the bars. As the sea retreats, the bars cease to be marine and some rich ones are therefore found in estates some distance from the shore. A bed of clay may cover them and usually does to a thickness of 3 ft. and sometimes 6 ft. At Boâ-Vista, Itapermerim, State of Esperito Santo, the deposits occur in parallel bands, and one of these contains 3000 tons of monazite. Exploration is continued by borings or shafts down to a colored clay called tapatinga, which rests on sandstone. This tapatinga occurs over the

[1] *Annaes da Escola de Minas de Ouro Preto*, No. 11, 1909. Abstract from translation in the *London Mining Journal*.

Amazon valley and is normally just below the concentrated sands. In the municipality of Mar, Fazenda de Arribade, "Spanish Mines" there is an ancient gravel containing 1 per cent. monazite and 0.5 per cent. of xenotime in splendid crystals. Xenotime is the yttrium phosphate.

The alluvium of the interior, containing monazite, is formed of sand and gravel sometimes associated with clay. The so-called torrential sands (areia de encharada) occur in the older placers and therefore the later ones are barren of monazite.

The monazite increases in richness when reddish garnets preponderate. The deposits having the form of a basin or funnel and also a narrow outlet, are likely to be rich. Above the deposits are vegetable remains such as tree-trunks and cocoanuts in perfect preservation and below the deposits is a layer of argillaceous earth. Some of these deposits contain 300 tons of monazite. In these deposits the monazite is from 0 to 2 per cent.; there are spots, however, as in the Fazenda dos Campos Elysios, containing as high as 50 per cent.

In the interior river sands, as on the Parahyba, many iron minerals occur. The percentage of monazite is three-tenths. Washed in sluices, black sand remains which contains 3 to 10 per cent. monazite and which is concentrated electromagnetically to 80 to 96 per cent. monazite.

Some of the deposits in the interior are preferred to the coast ones on account of higher thorium contents. The crystals of the mineral are more perfect and less worn. The interior deposits also are abundant, and average 2 per cent. monazite; they are going to last long after the coast deposits are gone. The coast material is richer in monazite, however, containing 30 to 90 per cent. very commonly.

The coast deposits frequently have a cap of clay or quartz sand or only the latter. Some are worked only at low tide as they are covered at high water. In working these in pits, advancing up hill, the covering serves as a filling below. The cost on the sea coast never exceeds 30,000 reis per ton although wheelbarrows are used to transport the material to to the washery. As the sands of the level lands contain only a small amount of iron, this material can be concentrated by a cradle called a berço, to 40 or 50 per cent. monazite, and by means of a dry blower to 70 or 80 per cent., which grade is then exported. The pebbles in all cases are separated on inclined tables, the sand going to screens. These are rotary and submerged in water. The marine sand dried on iron plates is concentrated on hand screens of 40 to 60 mesh. On a 40 screen one man in 9 hours can produce 1500 kilos of enriched sand, this being two-thirds of the original. This is dried for the electromagnets. The dry blowing can be used instead of screening. Sluicing never brings the monazite contents

above 60 per cent. and this is too low for the commercial (4 per cent. thorium) product.

Shaking tables are less satisfactory because zircon, ilmenite, garnets, etc., prevent concentration beyond a certain point, unless heavy loss is incurred. The electromagnetic method generally follows the washing. In the State of Rio de Janeiro two types of magnetic machines are used; one at Sapucaia on the Parahyba, was invented by Charles Rau, and the other in Barra de Itbapoana, belongs to the Societé Minière et Industrielle, Franco Brésillienne. On the latter machine currents of 2 to 20 amperes are used and the product is 96 per cent. monazite.

Carolinas.—The deposits of monazite in the Carolinas which have proved of commercial value have been described in Vol. XIX of the MINERAL INDUSTRY, page 485. The placer deposits are derived from the near-by rocks, some of these having been entirely altered to soil and in these cases they may be best worked by hydraulicking or sluicing. The methods of concentration are described in last year's article in the MINERAL INDUSTRY and are reviewed as follows: The usual method for washing the sand is in a trough 12 ft. long, 18 in. wide and 1 ft. high. Two men operate such a trough by simply shoveling the gravel on the perforated plate, a stream of water, about 25 gallons per minute, being turned on the gravel. In this way, the smaller sized material falls through the plate into the trough where it is constantly agitated by the troughman's shovel and the coarser gravel remains on the plate and is shoveled away. The process is continued for about 2 hours, the water finally leaving the monazite sand at the head of the trough. This sand is then dried either in the sun or by artificial means. Sometimes the process at the mine is carried on in two stages, a rough washing and a final rewashing, making the sand ready for delivery to the magnetic separator. Probably 95 per cent. of the monazite produced in the Carolinas has been secured by this method. Other means such as hydraulicking, sluice washing, or concentrating tables, have been tried.

The association of heavy oxides of iron with the monazite in the Carolinas often constituting as much as one-half of the sand prevents wet dressing. Formerly the crude material was selected so as to make it possible to obtain 66 per cent. monazite sand directly. The latest practice was the use of magnetic concentrators which make it possible to utilize sands containing no more than 10 per cent. monazite.

The two types of separators used in the Carolinas are the Daggett and Wetherill. The Wetherill machine makes a nearly perfect separation of the monazite, the capacity averaging 12,000 lb. of crude sand in 10 hours. The Daggett magnetic separator has also been used successfully by one company.

Idaho.—Other localities in the United States that might prove of commercial value are the deposits of Idaho. These are described by F. C. Schrader.[1] The occurrence is in the southeastern part of Nez Perce County and near Centerville, Boise County.[2] A number of placer mines were annually operated for gold during a short season and monazite was found to be plentiful in the clean-ups. This was especially true at Pierce and also at Musselshell creek. Monazite was first reported in the western States from Boise County, by Lindgren in 1896, but a number of localities furnished sand containing monazite for the experiments conducted by the Survey in 1905, on black sands. These sands come from Washington, Oregon and Idaho. The Idaho deposits are believed to extend into Montana.

In the process at the plant of the Centerville Mining and Milling Company, near Centerville, Idaho,[3] the gravel was hydraulicked into sluice boxes passing by the milling plant. Grizzlies in the bottom of the sluice boxes permitted the fine material to drop through to the mill, where it was handled by Wilfley tables. The concentrate was passed through a drier, and then through a series of magnetic separators, of gradually increasing intensity of field. The first separator took out the magnetite (about 50 per cent. of the entire mass) which had no commercial value. The second took out ilmenite and garnet. The ilmenite has no commercial value, but may be of use some time for its titanium content, while the garnet is an abrasive. The next series of magnets took out the monazite sands, which at the Centerville plant, are said to contain about 5 per cent. thorium and 50 per cent. cerium and didymium. The best data obtainable tend to disprove the Company's claim. (*Mineral Resources*, 1909.) The cerium and didymium are of value for making smokeless photographic flashlight powder. The plant was destroyed by forest fires, in 1910, but it is expected to rebuild, and also to make gas mantles in order to get the entire profit from the thorium.

Foreign Countries

India.—In the Sankara mica mine, a notable occurrence of the mineral samarskite has recently been described.[4] It occurs in imbedded masses in the granite. An analysis shows cerium and yttrium with tantalum and niobium and also the presence of uranium probably combined with iron and calcium. The mineral is strongly radioactive. In the same mine is found a pale grey-brown mineral, a hydrated zirconium silicate,

[1] *Bull.* 430-D, *United States Geological Survey*, p. 184–191.
[2] D. B. Sterrett, Monazite and Zircon, Mineral Resources for 1909.
[3] Report of the Idaho inspector of mines for 1910.
[4] G. H. Tipper, F. G. S. Vol. XLI, part 3, *Records* Geological Survey of India.

generally near quartz bosses or the above samarskite, containing the following: ZrO_2, 62.3 per cent.; SiO_2, 26.1 per cent.; Fe_2O_3, 5.4 per cent. U_3O_8, 2.4 per cent.; water, 3.3 per cent. and traces of cerium and manganese.

An extensive deposit of naturally concentrated monazite sand has been discovered near Quilon, Travancore. This sand contains 4 per cent. of thoria, and consists of from 46 to 50 per cent. of monazite, with ilmenite and zircon as the other chief constituents, together with some rutile and traces of spinel, garnet, quartz and hornblende. A specimen of pure monazite separated from the sand contained 8.5 per cent. thoria. A specially prepared concentrate contained 8.87 per cent. thoria, and pure monazite separated from this contained 10.08 per cent. thoria. The commercial concentrate contains 5 to 6 per cent. thoria, as compared with less than 4 per cent. in the Brazilian sand, and the pure monazite from the Travancore deposit is richer than that from the Brazilian and Carolina deposits, resembling the monazite from Ceylon.

Italy.—Salts of thorium, cerium, and zirconium were imported into Italy in 1911 amounting to 3303 kg. valued at 165,150 lire.

Nigeria.—Monazite-bearing sands have been found in streams to the west of Naraguta, in Nigeria. In one case the tin concentrate contained nearly 30 per cent. monazite and zircon. The Mineral Survey in 1908 found a good deal of monazite among the granite hills at Kirigama and Kajuru. Analysis of some tin concentrate obtained at Jalingu showed 6 per cent. ceria and thoria compounds present.

Rare Earths

Under these are included, for practical purposes, cerium, thorium and zirconium. The latter does not properly belong to this group, but is included because its practical uses are similar. In addition, there are a good many rare metals which need not be described separately.

Such as are definitely determined are: scandium, yttrium, samarium, europium, gadolinium, terbium, dyprosium, erbium, thulium, holmium, lutecium and ytterbium. Of these latter yttrium is the commonest and most important. All these metals occur in minerals found in granites. The chief source of yttria is from gadolinite, found among other places at Ytterby, Sweden. Yttria was required at first for the Nernst lamp and it was obtained from gadolinite.

Although monazite sand is abundant, its mass only forms probably one hundred-millionth part of the earth. The rare minerals that furnished these earths, cerite, thorite, and monazite, are only a small volume of the crude sand. Gold similarly occurs disseminated; and

is often found with the naturally concentrated sands, in fact the original monazite minerals were thrown away in North Carolina in some early gold washing operations. The name "rare earths" is not justified as formerly, specially as at the time the incandescent mantle lamps were first put on the market. There are known to-day 100 minerals containing these elements.

The compounds in a pure state of these elements are difficult to prepare. With the exception of cerium, thorium and zirconium, the methods of obtaining distinct products are not accurate and no commercially feasible separation has resulted. The fractionating processes on some salts allow only very small percentages of the products to be obtained and these are generally mixtures though simpler than the original salts. The repetition of similar operations finally leaves only a very small amount of substance making it impossible to go further.[1]

Cerium and Thorium

The cerium oxide is used in about the purest state of any of these earths for the manufacture of gas mantles. It is also more quickly prepared and the cerium product forms about 50 per cent. of the waste resulting from the treatment of the crude material. The thorium product contains, as it is finally used, only 1 per cent. cerium oxide and 99 per cent. thoria.

The fine Brazil monazite is directly leached while the coarser Carolina sand is first ground and then leached with concentrated hot sulphuric acid. This converts the phosphates into sulphates which are then stirred and dissolved with water. At this stage the enrichment in thorium is begun. The excess sulphuric acid is neutralized giving a slimy precipitate of phosphate which, with the proper conditions applied, contains all the thoria and with final enrichment of 50 per cent. of the cerium earth. Everything depends on the first precipitation and it is interesting to know that the methods used to-day are able to extract 90 to 95 per cent. of the thorium from 5 per cent. monazite sand. Having arrived at a 50 per cent. thorium product, the further cleaning of the substance is the same as with the old processes for thorite, although in this case the treatment is applied to silicates instead of to phosphates as at present.

The phosphate-cakes from the filter presses and containing 50 per cent. enrichment, are converted by means of oxalic acid into oxalates. Formerly these were treated with ammonium chloride until the cerium oxide was reduced to 20 per cent. Then acetate was added by converting an added 98 per cent. carbonate with acetic acid, which heated gave a

[1] C. R. Bohm, *Die Chemische Industrie*, 1906.

white crystalline precipitate of pure thorium acetate; this was then converted into nitrate. This method would be too expensive at present prices of thorium nitrate. The slower but cheaper sulphate method is therefore used.[1] The large quantities of cerium carbonate contained in the wastes of this industry are not marketable except if the cerium could be utilized. Bohm has brought on the market the cerium carbon for arc lights which is more effective than with calcium salts that have been used for this purpose by Bremer.

This, however, was the sole use until the pyrophoric metal industry was established. The carbon industry did not use much cerium. Much search has been made to turn this waste product from the thorium industry into use. The spark emitting property of crude cerium metal combinations when struck with steel, caused Auer von Welsbach to apply it to practical use. The matter was however described before in 1875 by Hillebrandt Norton and others. In turning a cerium alloy in a lathe, Bohm found the heated sparks dangerous and had to apply oil for lubricating the cutting tool. The use of these alloys for lighters is well known. The manufacture of the cerium alloys has been described by Muthmann who electrolyses the anhydrous chlorides in a furnace and produces any alloys with say iron, magnesium, etc. The magnesium alloy made by Kunheim and Cie in Berlin, does not stand exposure as well as the hard Auermetall which is an alloy with 30 per cent. iron. The cause of this disintegration in the atmosphere has been attributed to carbon contents, due to carbon electrodes. This pyrophoric metal has been replaced largely, however, with the other combinations of iron manganese and uranium, which require special conditions for striking fire just as does flint and the ordinary steel. The consumption of cerium alloys is about 4000 kg. per year the world over. The metals have been applied in automatic lighters for gas jets.

Cerium is found in beds in gneiss at Bastnas, Sweden, in cerite, a hydrous silicate, long the only cerium source.

Allanite, a cerium epidote is of common occurrence in many localities in the United States.

Metallic Cerium.—From recent investigations made, which are the latest and most thorough on this subject,[1] it was found that the thermal reduction of the purest obtainable oxides by means of metallic magnesium, calcium, aluminum and black thermite, gave alloys or lower oxides. It was also found that carbon and silicon gave carbide and silicide. With the electrolytic method, trichloride or double fluorides are usually used; but it was found that impure chloride does not give good results for the reduction of metallic cerium. The pure chloride was finally prepared by

[1] Bohm, *Darstelling der Seltenen Erden*, Leipsig, 1905.
[1] Alcan Hirsch, Am. Electrochemical Society, Sept., 1911.

evaporating the nitrates very slowly with hydrochloric acid and the resulting mixture was dehydrated in an atmosphere of hydrochloric gas. The anhydrous chloride alone breaks up into oxide and hydrochloric acid and in order to overcome this difficulty a scheme was used of mixing it with sodium chloride before electrolysis. From the fused or anhydrous chlorides an alloy, metallic cerium 98 per cent. and iron 1.2 per cent., is prepared in a wrought-iron vessel cathode, the anode being graphite; currents of 200 amperes and 15 volts were found efficient. The current efficiency was 40 per cent. The physical properties of cerium are as follows:

Specific gravity, 6.92 at 25° C. It is non-crystalline and very hard but at the same time malleable and ductile. It has a high heat conductivity, is ferro-magnetic, has a melting-point of 635° C. and its specific heat is .0524. The heat of combustion is −60,900 cal. The ultimate strength is 12,900 lbs. per square inch and the test bar will snap like cast iron.

A matter of 600 grm. of the pure metal has thus been manufactured during the year 1911. At first, it fetched a price of $6 per gram but the present price is $1. Cerium is the strongest reducing agent known, the reaction in taking oxygen being endothermic and requiring therefore the addition of heat.

Cerium Pyrophoric Alloys.—Alloys or compounds, usually containing about 70 per cent. of cerium, were prepared by A. Hirsch,[1] with Ag, Au, Pt, Cu, Sn, Sb, As, C, Si, S, Se, Te, Pb, Ca, Na, Al, Zn, Cd, Cr, Fe, Mn, Ni, W, Hg, and Mg. The magnesium-cerium alloys form effective reducing agents. The magnesium-cerium alloys, carrying 75 to 85 per cent. of cerium, are highly pyrophoric, and those of high cerium content make fine flashlight powders. Cerium silicide ($CeSi_2$) is formed by reducing cerium oxide with carbon and silicon. The result, when added to cerium in such proportions as to give an alloy carrying 15 per cent. of Si, is highly pyrophoric. The silicide also is an excellent reducing agent.

The non-cerium alloys are interesting, as they were the first pyrophoric alloys of cerium known. The explanation of their peculiar properties is given by Doctor Hirsch as follows: Cerium is soft and malleable, and when filed, small particles are not broken off. If the alloy is hard and brittle, small pieces are easily detached from the mass, and the friction is sufficient to raise the temperature of these small particles to the point of incandescence for cerium (160° C.) As metallic compounds (such as Cu_3Sn) are, as a rule, hard and brittle, the addition of such compounds to cerium usually insures a pyrophoric alloy. The alloy should be high in cerium, so that ignition occurs at low temperature, and should contain

[1] *Met. and Chem. Eng.*, 1911, p. 540.

an excess of cerium over that required to form a definite compound (*e.g.*, $CeSi_2$), to act as a binder and to prevent the disintegration.

An alloy which seems anomalous, in view of the above considerations, is that of manganese, antimony and cerium (U. S. pat. No. 1,006,849). In this the addition of about 5 per cent. of metallic cerium is made to give the manganese-antimony alloy softness enough to be abraded easily by friction. In this case the manganese-antimony alloy is itself pyrophoric before the cerium is added.

Ceric Oxide.—The ordinary slow methods of obtaining cerium earths from the crude double sulphates with alkaline hydroxides or with hydrochloric acid, may be replaced[1] by heating them one hour with charcoal. This reduces the cerium earths to sulphides which are soluble in hydrochloric acid from which solution oxalic acid, as usual, precipitates them.

A new method of separation in analysis and preparing pure cerium material, was found by C. James and L. A. Pratt,[2] who use bromate in a faintly acid or neutral solution and boil. A method of separation of ceric oxide by permanganate is given by E. J. Roberts.[3]

Thorium.—The best binding agent found for mantles has been thoria applied in the form of thorium nitrate which when heated melts, then gives a foamy solid product on account of the evaporation of the gaseous products.

The property is more pronounced when 2 per cent. of sulphuric acid is added to the solution. It is only desirable with cotton mantles and not with those of silk, which latter disintegrate easily with thorium; in this case the acid has to be neutralized by ammonia, alkali, etc.

Thorium, the radioactivity of which is of great interest, gives off an emanation that penetrates many metals such as aluminium; actinium discovered by Debierne (1899) in pitchblende, is related to this thorium emanation. The products that are separated by precipitation from thorium compounds and that have this radiating property can be concentrated similarly to the methods used for the radioactive products from uranium combinations.

In 1905 Hahn separated the radio-thorium from 2500 kg. of the new mineral thorianite and recently has added the new product mesothorium. The thorium industry is of great interest for one reason, that metallic thorium has more radioactivity than radium and its price ought to be only one-third. Hahn hoped to recover annually enough radioactive thorium equivalent to 10 grm. of radium bromide which contains as much radium as does the total amount of radium salts that has been manufactured.

[1] *Am. Journal of Science*, August, 1911.
[2] *Chem. News*, CIV, 61.
[3] *Am. Jl. Sci.*, May, 1911.

Thorium has so far resisted all efforts to prepare in a pure state. The purest contains 96 to 97 per cent. thorium, and was prepared by heating thorium hydride in a vacuum of 10 mm. The metal thus obtained is black, does not oxidize in contact with air or oxygen at ordinary pressures, but oxidizes in oxygen under pressure and takes fire in fused potassium chlorate, burning with a luminous flame.

Basic Thorium Sulphate.—O. Hauser[1] finds that the transformation described by Barré,[2] which occurs so slowly at ordinary temperatures, takes place much more rapidly at 125° C., and that the salt, $ThO(SO_4)_2$-H_2O, is stable between 125° and 180° C. He has not investigated the possible formation of other hydrates, but finds that the above salt is obtained pure by heating a solution of anhydrous thorium sulphate (1 part) in water (140 parts) for some hours to 160° C.

Comparatively large quantities of thorianite, the mineral which was believed to occur only in Ceylon and where it was discovered by the Survey, has been profitably exported in recent years. Much remains to be done in discovering new localities where this mineral is present and also in devising better means of recovering it from the river beds.[3]

Zircon

Zircon occurs associated with monazite in many placers, either in the form of sand or crystals. This mineral is a silicate with 67.2 per cent. zirconia and occurs in the Carolinas. In 1911 the United States Geological Survey estimated a production of zircon of 3208 lb. valued at $802. This came from Henderson County, N. C. where W. E. Hidden of Newark, N. J. and M. C. and C. F. Tomes of Hendersonville, N. C. operated, the former the Freeman mine, and the latter two the Jones mine near Zirconia. Neither of these mines has been operated regularly. An interesting test is reported for auerlite, about 1 kg. of which was obtained at the Freeman.[4] The mineral is about 70 per cent. thoria. The concentrates are obtained by washing a decomposed pegmatite. The production of zircon is not regularly given though the following figures for previous years are from Mineral Resources of the United States Geological Survey: 1903, 3000 lb., value $570; 1906, 1100 lb. valued at $200. That in 1909 was from the Jones mine as in 1911. In 1909 also C. H. Hackney took out some zircon in development work at La Harpe, Kansas, and Wichita, Kansas.

Thomas L. Watson and T. L. Hess[5] report on occurrence of zircon in

[1] *Ber.*, 1910, XLIII, 2776–2778.
[2] *J. Indstrl. and Eng. Chem.*, 1910, 947.
[3] *Journ. Soc. Chem. Ind.*, Nov. 30, 1910.
[4] W. E. Hidden and J. B. Mackintosh, *Am. Jl. of Science*, Vol. XXXVI, p. 46.
[5] *Bulletin* 530-P, U. S. G. S.

sandstone near Ashland, Virginia. Ilmenite is associated with it. Zircon is a fairly good insulator both as to heat and electricity. Some clear colored varieties are used as gems, such as hyacinth.

Zirconium

Zirconium earth was long recognized as having pronounced incandescence, and was used in place of the original lime blocks of Drummond with the oxyhydrogen jet. Thorium earth was extremely rare when first its properties were realized but the discovery of the large monazite sand deposits allowed the chemists to obtain an abundance and without these deposits the gas mantle industry would have been impossible. Likewise, so long as the principal occurrence of the zirconium was the form of zircon or silicate, a difficult material to separate, it was not possible to obtain a clean oxide at a reasonable cost, and to this day there is no cheap and profitable method for the silicate.

Zirconia is now obtained from native Brazilian zirconia, containing 90 per cent. and is much more easily purified than when the mineral zircon was used.

In 1906 near Sao Paolo, Brazil, zirconium oxide was discovered in large quantities. Edward Rietz who worked this deposit deserves credit for his energy in interesting German chemists, notably Ludwig Weiss, and it was arranged that the material be manufactured and its uses extended. The zircon earth is in the trade in the powdered form. The percentage of Zr is about 85 and the cost per kg. about 15 cents. The firm of Wesenfeld, Dicke and Company, at Barmen, also makes high-grade products as follows:

Zirconium S. ore 90–92 per cent. ZrO_2, 1 per cent. Fe_2O_3, 8 per cent. SiO_2
Zirconium N. ore 90–92 per cent. ZrO_2, 7 per cent. Fe_2O_3, 1 per cent. SiO_2
Zirconium NS. ore 98 per cent. ZrO_2, 0.8 Fe_2O_3, 1 per cent.

the last costing 25 cents per kilogram. The Allgemeine Elektrizitäts Gessellschaft has used much zirconia for the Nernst lamp but the recent advent of metal filament lamps makes this use small. Zirconia had replaced beryllium as a hardener for this use.

There are many favorable properties of zirconium.[1] Zirconium carbide has not proved a good filament, as the recent lamps made by Hollefreund and Company, Berlin, were provided only with tungsten filaments for mantles. Zirconia is suitable for crucibles and hearths as its refractory qualities are excellent. Chemical utensils are manufactured by W. C. Heraeus in Hanau; and Elias Goebel and Son, Epterode in Hessen Nassau, manufacture crucibles of all descriptions. For enamel the oxide is not

[1] Translation from Dr. Rich. Böhm, *Chemiker Zeitung*, 14 Nov., 1911.

so suitable as the borates and a number of patents are taken out for processes. It can be predicted that the use for ceramic arts will be extended. The main difficulty to obtain a cheap iron free product will no doubt be solved shortly. Ferro-zircon, for use in steel manufacture, and also furnace linings especially adapted for electric steel furnaces, are sold by Wesenfeld, Dicke and Company. Further uses are for permanent moulds for castings, and for an insulating material. As an abrasive, zirconium carbide will also prove its worth on account of hardness; it is suitable for cutting glass and up to 7 mm. thickness it answers like a diamond. Zirconia is a good polishing material and could replace tin oxide. Mixed with good conductors, it makes a good furnace electrode. A suitable salt of zirconium could replace thorium nitrate or yttrium nitrate for preventing the loss of iridium by repeatedly coating the iridium bar with these substances in the Heraeus furnace. Zirconia would have sufficient fire-resisting properties for this purpose.

Zirconia is used in place of bismuth subnitrate, which is poisonous in some cases, for defining Röntgen pictures of the human stomach. Such a substance known as kontrastin is on the market.

Zirconia has an excellent covering power as a pigment and is permanent and white, not being altered by sulphide, acids or alkalies. It also has application in medicine. Zirconia has also been proposed as a non-irritant powder for covering the surface of wounds.

By Chas. Baskerville.—A Frankfurt a. Main, Germany, firm has recently placed on the market products ("siloxide") prepared by fusing silica with small proportions of refractory acidic oxides such as zirconia or titania (see French Patent 432,786, July 31, 1911, of F. Wolf-Burckhardt and W. Borchers), with the purpose of obtaining materials more resistant to the action of metallic oxides, especially alkalies, than quartz glass and having a lesser tendency than the latter to suffer devitrification on heating. Thomas (*Chem. Ztg.*, XXXVI, 25) has recently examined specimens of Z-siloxide, or zirconium glass, and T-siloxide, or titanium glass. While these did not exhibit the silky luster of quartz glass ("vitreosil"), they possessed distinct advantages over the latter with respect to strength, thermal properties (resistance to devitrification), and resistance to the action of alkalies. In the case of Z-siloxide, the best material with respect to strength contained 1 per cent. of zirconia, while that containing 0.5 per cent. possessed the best thermal properties. Zirconium glass has a softening point not much different from that of quartz glass, but it is said to resist deformation better at high temperatures, owing to its greater viscosity. T-siloxide is said to be superior even to Z-siloxide with respect to thermal properties. It is reported that both titanium and zirconium glasses may be worked more easily than pure

quartz glass, and that they may, in fact, be worked by the ordinary methods employed in glass manufacture. (See also under Quartz, p. 649.)

Nearly all attempts to replace tin oxide in the enamel industry[1] have failed. Titanium oxide, zirconia, etc., that can be manufactured even in great purity have failed; this is on account of the great amount required, thus causing porosity. Zirconia can be used[2] if it does not exceed 14 to 17 per cent. and with large relative amounts of cryolite. For example:

Feldspar	37.5	per cent.
Borax	18	per cent.
Cryolite	21	per cent.
Quarts	10.5	per cent.
Saltpeter	7	per cent.
Zirconia to fusion	8	per cent.
Zirconia to grinding mill	6	per cent.
Clay to grinding mill	7	per cent.

In this case the covering power depends not on the zirconia but on the cryolite. On account of the price of zirconia, however, it is better to work with borates and it is said that this zirconia borate is to be an article of commerce shortly. It could be added in either way as above, but if added in the mill it is likely to cause difficulty in applying the enamel even with small amounts of borates. Dr. Weiss has succeeded in producing zirconia free from iron but it cannot compete with tin oxide as yet for opaque glass and enamels.

Columbium oxide or niobium oxide and tantalum oxide, Ta_2O_5, are acids occurring in a large number of minerals such as columbite, tantalite, samarskite and euxenite. As is well known metallic tantalum is used like tungsten in filaments for electric lamps but the latter has more or less replaced it. The supply of tantalum is from Sweden and Norway. Specimens of native tantalum have been found in Siberia.

Struverite, a rare tantalum mineral, found in the Federated Malay States, is described in the *Bulletin of the Imperial Institute*, Vol. IX, No. 4, 1911. The production of tantalic acid from West Australia Fergusonite is given by Wedekind and Maass.[3] An indirect method for determining columbium and tantalum is given by H. W. Foote and R. W. Langley, in *American Journal of Science*, December, 1910.

Some Journal Articles Appearing During 1911

Anon. "Rare Earths, Their Occurrence and Uses." (*Aust. Mining Engineers*, 13, 87–114.)

Benner, R. C. "Technical Applications of the Rare Earths." (*Aust. Min. Stand.*, July 6 and 13, 1911.)

Böhm. "Technical Applications of Zirconia." (*Chem. Zeit.*, XXXV, 1261.)

[1] *Sprechsaal Coburg*, 1911, No. 5.
Kgl. *Technische Hochschule*, Munchen, 1910.
[2] *Zeitschr. Angewandte Chemie*, 23, 2314.

BÖHM. "The Rare Earths Considered with Special Reference to the Thorium Industry." (*Z. Ang. Chem.*, XXIV, 1203.)

BRONIEWSKI AND HACKSFILL. "Electrical Properties of the Alkali Metals, Rhodium and Iridium." (*Comp. rend.*, 153–814.)

HICKS, WILLIAM BROOKS. "Rare Earth minerals." (*Journ. Am. Chem. Soc.*, Sept., 1911.)

FRENZEL, A. B. "Growth of the Rare Metal Industry." (*Mining Science*, Jan. 25, 1912.)

HIRSCH. "The Preparation and Properties of Metallic Cerium." (Trans. Am. Elec. Soc., 20–57.)

JAMES and PRATT. "Separation of Cerium." (*Chem. News*, 104, 61.)

JAMES and PRATT. "New Rare Earth Compounds." (*Journ. Am. Chem. Soc.*, Aug., 1911.)

JAMES and ROBINSON. "Europium." (*J. A. Chem. Society*, XXXIII, 1363.)

LANCIEN, ANDRÉ. "The Application of the Reducing Metals, Aluminium and Calcium to the Preparation of Rare Metals." (*Rev. d'Econ Indus.*, June 16, 1911.)

MINER and WHITAKER. "Production and Application." (*Journal Ind. and Eng. Chemistry*, Vol. I, p. 235.)

STODDARD. "Stearate Separation of Rare Earths." (*J. A. Chem. Soc.*, XXXIII, 1076.)

NICKEL AND COBALT

Practically no nickel or cobalt is produced in the United States from native ores but both are made from imported ores and mattes, almost entirely from Canada. The deposit at Frederickstown, Mo., was not operated during the year, nor was that of the North Carolina Nickel Co., at Webster, N. C. At the latter plant alterations are being made and this silicate deposit will probably be worked again. A deposit of nickel ore exists in the San Poil mining district of Washington but it is of low grade and 30 miles from a railroad. There are also extensive ore bodies on Blackbird Creek in Idaho and some are said to contain 3 to 35 per cent. of cobalt in the form of sulphides associated with copper and gold. In Floyd and Roanoke Counties in Virginia, there exists a number of prospects which have recently been acquired by the United Chemical and Nickel Corporation. The ore is reported to contain about 1.75 per cent. Ni, less 1 per cent. of copper, and less than 0.4 per cent. Co.

Considerable nickel is recovered as nickel sulphate from the refining of copper by the electrolytic process, but probably the larger part of this nickel is of foreign origin. In 1909 this by-product, nickel, amounted to about 3,000,000 lb. valued at $62,000 but this has been largely increased

UNITED STATES IMPORTS AND EXPORTS OF NICKEL AND COBALT.
(In pounds, and tons of 2240 lb.)

Year.	Imports.							Exports.	
	Nickel Ore and Matte.		Nickel Alloys. (a)		Nickel Mnfrs.	Cobalt Oxide.		Nickel. (b)	
	Long Tons.	Value.	Pounds.	Value.	Value.	Pounds.	Value.	Pounds.	Value.
1898.....	26,826	$1,534,262	(c)	.....	.....	33,731	$ 49,245	5,657,618	$1,359,609
1899.....	19,857	1,216,253	(c)	.....	.....	46,791	68,847	5,004,377	1,151,923
1900.....	25,670	1,183,884	455,188	$139,786	.....	54,073	88,651	5,869,906	1,382,727
1901.....	52,111	1,637,166	635,697	209,956	$ 2,498	71,969	134,208	5,869,655	1,521,271
1902.....	14,817	1,156,372	752,630	251,149	30,128	79,984	151,115	3,228,607	925,579
1903.....	15,936	1,285,935	521,344	170,670	37,284	73,350	145,264	2,414,499	703,550
1904.....	8,548	915,470	589,555	203,071	2,950	42,352	86,925	7,519,206	2,130,933
1905.....	13,451	1,626,920	941,966	331,920	3,291	70,048	139,377	9,550,918	2,894,700
1906.....	15,156	1,816,631	210,000	77,373	8,963	41,084	83,167	10,620,410	3,493,643
1907.....	(d)16,888	2,153,873	180,025	80,994	9,159	48,013	74,849	8,772,578	2,845,663
1908.....	(e)16,322	2,396,217	241,868	91,388	10,010	219,098	17,077	9,770,248	3,297,988
1909.....	(f)18,578	2,927,975	277,911	104,019	4,279	12,132	11,096	12,048,737	4,101,976
1910.....	(g)28,519	4,085,076	323,239	125,013	14,759	6,124	4,806	15,244,937	4,704,088
1911.....	(h)23,993	3,918,556	293,301	104,160	.....	14,561	7,576	25,099,586	8,283,777

(a) Includes nickel oxide, and alloys of any kind in which nickel is the material of chief value, in ingots, bars, and sheets. (b) Comprises domestic nickel, nickel oxide, and matte. (c) Not separately enumerated; included in "Nickel Ore and Matte." (d) Contained 18,418,305 lb. nickel; not reported previous to 1907. (e) Contained 16,586,423 lb. nickel. (f) Contained 21,916,182 lb. nickel. (g) Contained 32,050,032 lb. nickel. (h) contained 29,545,967 lb. nickel.

as some of the refineries which were not recovering the nickel then have since put in plants for this work. Electrolytic nickel is now also made at at least one of the copper refineries, as a by-product.

The largest bulk of the exported nickel of the United States went to the Netherlands, viz., 9,101,150 lb. worth $2,835,465 in 1911. The United Kingdom received 7,196,259 lb. and France 5,463,358 lb. the same year. The imports to the United States included 23,993 tons of ore and matte, worth $3,918,556, containing 29,545,967 lb. of nickel.

The relative smelter capacity for nickel in the products was as follows, in recent years, as estimated by the "Metalgesellschaft":

North America	10,000
England	3,500
Germany	4,500
France	1,500

Prices.—The price of refined nickel in New York was 45 to 50 cents in large lots early in the year and 40 to 50 cents from May on. Cobalt oxide has been quoted uniformly at $.80 to $1.00 per pound throughout the year.

Nickel and Cobalt Mining and Metallurgy in Foreign Countries

Canada. Mining (by D. P. Schuler).—The most notable change in the Sudbury district has been the opening of the Froude mine by the Canadian Copper Company and an extension of the same deposit by the Mond Nickel Company. In both cases, the ore is covered with 500 to 800 ft. of rock, thus requiring much deeper shafts to reach the ore than any heretofore sunk in that district. Both companies are extending their power lines to the mines and the Canadian Copper Company is constructing a railroad to connect the Froude mine with its system around Copper Cliff. The Mond Company will utilize the old Stobie branch of the Canadian Pacific railroad by the construction of a short spur.

The Dominion Nickel-Copper Company has continued active development of the Whistler mine besides diamond drilling several of their other propeties.

Altogether about 150,000 less tons of ore were mined by the Canadian Copper Company and the Mond Company in 1911 than in 1910. However, the total nickel produced exceeds that of any year except 1910. To this might be added the nickel oxide produced at the Coniagas Reduction Works at Thorold, possibly $50,000 in value.

Metallurgy.—Considerable improvements have been put into operation at the Canadian Copper Company's plant during 1911. Three Smith-Pierce basic converters were installed and the last started operating in December. These converters are the largest in the world—37 ft. long

and 10 ft. in diameter and have 40 tuyères. In December also two new reverberatory furnaces were blown in. They are 120 ft. by 19 ft. and use pulverized coal for fuel like a cement kiln. The modus operandi will consist as follows: The ore is first smelted to matte which is converted using lean ore instead of quartz for flux. This affords considerable saving as the quartz formerly had to be specially quarried. The converter slags are then charged molten into the reverberatories together with dried dust and fines and the resultant matte is returned to the converters. So the new reverberatories and converters are mutually interdependent. This new procedure saves labor and losses all around and lower grade matte can be treated. So the works at Copper Cliff began the new year prepared for larger operations at greatly reduced costs to meet the constantly increasing demand for nickel.

The Canadian Copper Company also treats cobalt ores by smelting to speiss and bass bullion. The speiss is then subjected to wet treatments yielding silver, arsenic and a crude cobalt material. Over 1,000,000 oz. of refined silver are shipped monthly and the cobalt residues are bought by the Coniagas Reduction Works.

The Mond Nickel Company has been delaying in building their proposed new smelter at the junction of the Canadian Pacific and Canadian Northern Railroads. Undoubtedly they are watching the operations of the Canadian Copper Company and will adopt some of their plans if they prove successful. The Mond Company probably smelted approximately 60,000 tons as against 450,000 smelted by the Canadian Copper Company.

PRODUCTION, EXPORTS, AND IMPORTS OF NICKEL IN CANADA.

Year.	Production.		Exports.		Imports.
	Pounds. (b)	Value. (c)	Pounds. (d)	Value. (e)	
1900	7,080,227	$3,327,707	13,493,239	$1,040,498	$6,988
1901	9,189,047	4,594,523	9,537,558	958,365	12,029
1902	10,693,410	5,025,903	3,883,264	834,513	15,448
1903	12,505,510	5,002,204	9,032,554	878,159	26,177
1904	10,547,883	4,219,153	14,229,973	1,237,307	16,330
1905	18,876,315	7,550,526	11,970,557	1,185,056	19,076
1906	21,490,955	8,948,834	20,653,845	.....	(f)15,808
1907	21,189,793	9,535,407	19,376,335	.....	(f)19,511
1908	19,143,111	8,231,538	19,419,893	.....	(f)36,870
1909	26,282,991	9,461,877	25,616,398	2,676,483	(f)14,930
1910	37,271,033	11,181,310	36,014,782	4,030,040	(g)23,317
1911	34,098,744	.....	32,619,971	.....	(g)34,199

(a) Statistics for production and imports cover calendar years, and are taken from the Annual Reports of the Geological Survey of Canada. Figures for exports cover the fiscal years ending June 30, and are taken from the Statistical Year Book up to 1905 inclusive. Subsequent figures are for calendar years as reported by the Canadian Geological Survey. (b) Pounds metallic nickel contained in copper and nickel matte exported. (c) On the basis of refined nickel at New York, from the *Engineering and Mining Journal* average annual quotations. (d) Pounds of nickel contained in ore, matte, or speiss. (e) Spot value, to the producer, of the exported material; the variety of stages at which the material is shipped, as well as the different periods of time covered, lead to the apparent discrepancy in value when it is known that practically the entire production is exported. (f) Anodes only, fiscal year. (g) calendar year.

The Coniagas Reduction Works produced about 100,000 lb. of cobalt and nickel oxides in 1911. They pay for the cobalt in residues on a sliding scale, 8 cents per pound if over 6 per cent., 10 cents if over 8 per cent., and 12 cents if over 12 per cent.

ONTARIO NICKEL STATISTICS.
(In tons of 2000 lb.)

Schedule.	1905	1906	1907	1908	1909	1910	1911
Ore raised	277,766	343,814	351,916	409,551	451,892	652,392	612,511
Ore smelted	251,421	340,059	359,076	360,180	462,336	628,947	610,834
Per cent. nickel	(b)	(b)	(b)	(b)	(b)	(b)	(b)
Per cent. copper	(b)	(b)	(b)	(b)	(b)	(b)	(b)
Ordinary matte } Bessemerized }	17,388	20,364	22,041	21,197	25,845	35,033	32,607
Nickel content	9,438	10,745	10,602	9,572	13,141	18,625	17,049
Copper content	4,386	5,265	7,003	7,503	7,853	9,630	8,966
Value of nickel (a) } Value of copper (a) }	$4,019,814	$4,629,011	$3,291,355	$2,930,989	$3,913,012	$5,380,064	4,945,592
Wages paid	(b)	$1,117,420	$1,278,694	$1,286,265	$1,234,903	$1,748,153	.....
Men employed	(b)	1,417	1,660	1,690	1,735	.....	.....

NOTE.—The quantity reported in 1903 under "bessemerized matte" includes both bessemerized matte and high-grade matte, the former being the product of the Mond Nickel Company's works and the latter of the Ontario Smelting Works, which re-treat the low-grade matte produced by the Canadian Copper Company. (a) Value based on nickel and copper in matte and not on refined metals. (b) Not available.

NICKEL EXPORTS FROM CANADA.
(In pounds.)

	1907	1908	1909	1910	1911
To Great Britain	2,518,338	2,554,486	3,843,763	5,335,331	5,023,393
To United States	16,857,997	16,865,407	21,772,635	30,679,451	27,596,578
Total	19,376,335	19,419,893	25,616,398	36,014,782	32,619,971

New Caledonia.—The local smelting of nickel ores is known to be a great struggle between Canada and New Caledonia.[1] These two countries share the nickel production of the world and the control of the market is in the hands of Americans who have at the same time, considerable interest in the New Caledonian production. They held certain leases on the island, but have allowed these to lapse. The Kuen Thio lease alone has been estimated to contain from 60,000 to 350,000 tons of high grade ore. This territory has found its way into the hands of the Societe le Nickel. Recently this locality made a good shipment to America to show what could be done. Other territories have been worked as particularly the Pic mine along the Dumbea and another is the Poro, which were taken under the care of the Tao under the same conditions. The Societe le Nickel is expending much energy on the Julia and Melbourne mines at Poro where the average value of the ore is 7 1/2 per cent. and aggregates 120,000 tons. At the Mission and Thio mines are some 40,000

[1] *Australian Mining Standard*, November 16, 1911.

tons of high-grade ore. These mines are along the Plateau and belong to the same company. There appear to be ore reserves for years to come. The ore reserves at Houailou are yet untouched. The opposition metal company, Maison Ballande, have the full output of the Prise du Rivoa and the Prise d l'Alma mines, and the Prise de Odessa, the Ouli-Ouli mine at Kua. The first-mentioned company employs Japanese labor. The group of nickel mines belonging to the Consolidated Company at Voh, closed down in 1905, are expected to be reopened.

It has been expected that the hydroelectric blast furnaces on the Island to produce mattes locally, would make a good showing. The first furnaces were opened at Noumea in July, 1911, and belong to Maison Ballande and they are now constructing some electric smelters at Yate for the Societe le Chrome, one at Pt. Bouquet for the Societe le Nickel and others at Pointe Doniambo at Tao, Thio and Dilah. At the latter, it may be said that the ferro-nickel matte produced was absolutely free from sulphur.

The production of cobalt in New Caledonia has entirely ceased. The nickel ore carries about 6 1/2 per cent. nickel.

SHIPMENTS OF NICKEL AND COBALT ORES FROM NEW CALEDONIA. (*a*)
(In metric tons.)

	1902	1903	1904	1905	1906	1907	1908	1909	1910	1911
Nickel ore.	129,653	77,360	98,655	125,289	130,688	101,708	120,028	82,937	115,342	120,659
Cobalt ore.	7,512	8,292	8,961	7,919	2,487	3,943	3,405	979	54	*Nil*

(*a*) Reported by *Le Bulletin du Commerce*, Noumea.

Norway.—The production of nickel in Norway is progressing. In 1911 the Evje company produced about 25,200 metric tons of ore. The smeltery treated 24,000 tons from the mine at Flaa, 2480 from the mines at Faœ (a separate company), 2070 tons of ore imported from Greece, and about 120 tons of matte from France. The metal content of the matte produced was: 418.5 tons of nickel, and 185 tons of copper. The company employs about 250 workers.[1]

The old mines at Ringerike are being reopened and the ore will be smelted at the mines to matte, which will later be sent to the Christiansand nickel refinery. The Christiansand nickel refinery employs about 100 laborers, and workers on the Hybinette system. In 1911 it produced 285 tons of electrolytic nickel, and 80 of copper. This works is now being extended to produce 600 tons of nickel and 300 tons of copper per year.

Other Countries.—At the Martha Mine in Lower Silesia, 10,042 metric tons of nickel ore were mined in 1910. The production of Prussia was 10,053 tons valued at 203,104 marks. There are five smelters which pro-

[1] *Echo des Mines.*

duced 3186 tons in 1909 and 3496 tons nickel in 1910. Some ore is also mined in Saxony. The total imports of nickel ores into Germany in 1911 amounted to 14,900 metric tons. In Russia nickel ores have long been worked in small quantities at the Datsche Rewdirsk in the Urals. Lately excellent results have been obtained in deep workings. A new promising deposit has also recently been discovered at Rio Grande do Sul in Brazil.

Cobalt.

The production of cobalt from the mines of the Temiskaming district much exceeds the consumption of the finished oxide and new uses must be found before this industry can expand. It is reported that in January, 1911, the refining companies had the equivalent of 350 tons metallic cobalt in storage. The price has been cut in half within the last few years but few new uses have developed. At present the principal use is in making pottery and glass.

The world's consumption is estimated at 300 tons of cobalt oxide annually, while sufficient ore was mined in Canada to produce easily 1500 tons in 1910.

One suggested increased use for cobalt is for making pigments. At present they are used mainly in artists' colors, but it appears that at the present prices they might be used commercially as pigments for higher classes of work, as some of these compounds possess advantages over others of similar colors. At 30 cents a pound it is estimated that a pigment could be made which would compete successfully with second quality ultramarine blue and even at present price, a green could be made which could compete with those in common use.

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NITROGEN FIXATION

By E. F. Roeber

Since the publication of the full review of this subject in The Mineral Industry, Vol. XIX (1910, p. 59) nothing new of a revolutionary technical character has come up in this field.

The most interesting development in this country is the starting,[1] early in 1912, of the works of the Southern Electrochemical Co., at Nitrolee, S. C. The plant has a capacity of 4000 h. p., the power being supplied from the two large water power plants of the Southern Power Co. at Great Forks and Rocky Creek. The Pauling process[2] is used. The output is estimated as 8 tons of calcium nitrate per day of 24 hours. Calcium nitrate is in successful use in Europe as a fertilizer in competition with Chile saltpeter.

Since the ill-fated pioneer experiment of the Atmospheric Products Co. at Niagara Falls which used the Bradley-Lovejoy process, but stopped operation in 1904, this is the first attempt to make nitrates from atmospheric nitrogen on a large scale in this country. Evidently the availability of large water powers in South Carolina and the market for fertilizers which exists in the South have encouraged this new experiment. No information is yet available on the results obtained at Nitrolee.

In Europe the manufacture of nitrates from air has become a decided commercial success[3] under exceptionally favorable conditions, characterized by very low power cost ($3 to $5 per electric horse-power per year). Technically it is interesting to note in the friendly competition between the Birkeland-Eyde furnace (flame disk produced by magnetic deflection) and the Schoenherr furnace of the Badische Company (long arc around which the air passes in spiral form) that the former type has proved more efficient in large sizes. This is attributed[4] to the fact that large units of the Birkeland-Eyde type can be built very compact, while for the Schoenherr type they must be made very high.

The several electrical furnaces of the Notodden plant which were formerly of the Schoenherr type have all been replaced by the furnaces of the Birkeland-Eyde type. The capacity of the Birkeland-Eyde furnaces operating at Notodden is 35,000 kw.

In the large new Rjukan plant, which is an extension of the Notodden plant with 180,000 available horse-power, furnaces of both types are to be installed in order to determine definitely which of the two types gives the best results in commercial service.[5]

[1] *Metallurgical and Chemical Engineering*, Vol. X, p. 126.
[2] In commercial use in Tyrol and France; see description in *Metallurgical and Chemical Engineering*, Vol. IX, p. 102, 196.
[3] J. W. Beckman, *Metallurgical and Chemical Engineering*, Vol. IX, p. 340, also p. 364.
[4] J. W. Richards, *Metallurgical and Chemical Engineering*, Vol. IX, p. 545.
[5] A. S. Garfield, *Metallurgical and Chemical Engineering*, Vol. IX, p. 436.

PETROLEUM

By T. T. Gray

The production of petroleum in the United States during 1911, amounted to 213,381,884 bbl., as compared with 208,196,415 bbl. in 1910, and 182,205,030, bbl. in 1909. California suffered from overproduction, increasing her 1910 output by about 4,000,000 bbl. The Gulf field showed an increase amounting to about 4,000,000 bbl., owing to the opening of the Electra field in Wichita County, Texas, and the development of the Caddo field in Louisiana. The Illinois field fell off in production about 3,000,000 bbl., due to the decline of the old fields, and the lack of new development in the later pools. In the Lima field occurred a continuation of the decline which started in 1907. The Mid-Continental field was again a large producer, yielding nearly 58,000,000 bbl. In the Pennsylvania field was a decline of about 4,000,000 bbl.

Of the foreign countries, Russia, the largest producer, is estimated to have declined about 4,500,000 bbl. in 1911. Mexico is estimated to have produced over 8,000,000 bbl. in 1911, as compared with 3,500,000 bbl. in 1910 and 975,000 bbl. in 1907.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.
(In barrels of 42 gal.)

Field.	1905	1906	1907	1908	1909	1910	1911
California (a)	35,671,000	30,538,000	40,085,000	45,000,000	58,250,300	74,327,150	77,224,359
Colorado	(e) 550,000	600,000	400,000	411,836	(b) 310,771	(b) 239,794	(b) 250,000
Gulf Texas	30,354,263	12,666,000	12,350,000	11,206,464	9,256,000	8,181,580	(e)10,000,000
Gulf Louisiana	9,672,015	7,100,000	4,620,000	6,835,130	3,220,000	(b) 6,841,395	(e)10,000,000
Illinois	.....	4,900,000	24,540,024	33,685,106	30,898,339	(b) 33,143,362	(e)30,000,000
Lima Indiana	22,102,108	25,680,000	8,030,000	7,287,000	6,192,000	5,171,000	4,656,000
Lima Ohio							
Mid-Continental(c)	12,000,000	21,929,905	47,556,906	48,274,782	48,007,267	53,613,030	57,911,525
Ky.-Tennessee	(e)1,200,000	1,000,000	1,250,000	1,250,000	(b) 639,016	(b) 468,774	(e) 300,000
Pennsylvania (d)	28,324,324	27,345,600	25,500,000	24,240,000	25,394,200	26,089,900	22,275,000
Wyoming	(e) 12,500	13,000	13,000	(e) 13,000	(b) 22,137	(b) 115,430	(e) 360,000
Others	(e) 3,000	4,000	3,000	(e) 3,000	(e) 5,000	(e) 5,000	(e) 5,000
Total	139,889,210	131,771,505	164,347,930	178,203,345	182,205,030	208,196,415	213,381,884

(a) Reported by the California Producers' Association, except the statistics for 1907 and subsequent years, which are of our own collection. (b) Statistics of the U. S. Geological Survey. (c) Kansas and Oklahoma. Includes production from North Texas, in 1910. (d) Pennsylvania, New York, West Virginia, Eastern Ohio, and, until 1904, Kentucky and Tennessee. (e) Estimated.

Petroleum Production by States

Alaska (By W. T. Prosser[1]).—A noteworthy oil development is taking place in Alaska. The Katalla fields, once regarded as most promising,

[1] *Eng. and Min. Journ.*, June 3, 1911.

and then for half a decade almost forgotten, are coming to the front again. New owners have obtained possession, and find that high-grade oil is ready to flow at least 2000 bbl. per day. Steel tankage to contain 30,000 bbl. of oil was shipped from California to Katalla to be put in place at the harbor, where tank vessels can load direct. Two other steel tanks were put in place at the wells, about 9 miles from the harbor. One of these tanks is of 2000 bbl. capacity, and the other 5000 bbl. A 2-in. pipe line was laid to the harbor, but recent developments at the wells convinced the owners that this would be entirely inadequate and a 4-in pipe line was ordered. The flow of the wells will be sufficient to fill two vessels a month, even if no other gushers are found. A refining plant, to reduce about 500 bbl. of oil per day, is under construction near Katalla. About half the flow of the four wells would supply Alaska's entire needs, as the northern district now imports a little over 17,500,000 gal. annually, worth over $500,000.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (In gallons.)
(1 = 1000 in quantities and values.) (*a*)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating.		Residuum.		Totals.	
1897	121,864	$5,044	13,704	$1,020	804,446	$46,876	52,659	$6,732	(*b*)12,247	$335	1,004,920	$60,007
1898	120,436	5,016	17,258	1,071	764,823	38,895	65,526	7,626	(*b*)30,436	815	998,479	53,423
1899	117,690	5,958	18,210	1,597	733,382	49,172	71,116	8,658	(*b*)21,609	658	962,007	66,043
1900	133,161	7,341	18,570	1,681	739,163	54,693	71,211	9,933	(*b*)19,750	845	986,855	74,493
1901	127,008	6,038	21,685	1,742	827,479	53,491	75,306	10,260	(*b*)27,596	1,255	1,079,059	72,786
1902	145,234	6,331	19,683	1,393	778,801	49,079	82,200	10,872	(*b*)38,316	922	1,064,234	68,597
1903	126,512	6,782	12,973	1,519	691,837	51,356	95,622	12,690	(*b*) 9,753	282	936,697	72,629
1904	111,176	6,351	24,989	2,322	761,358	58,384	89,738	12,389	34,904	1,174	1,022,165	80,620
1905	126,185	6,086	28,420	2,215	881,450	54,901	113,730	14,312	70,728	2,128	1,220,513	79,641
1906	148,045	7,731	27,545	2,488	878,274	54,858	151,269	18,690	64,645	1,971	1,269,788	85,738
1907	126,306	6,334	34,625	3,676	905,924	59,635	152,029	19,210	75,775	2,528	1,294,660	91,383
1908	149,190	6,520	43,890	4,542	1,129,005	75,988	147,769	18,971	77,552	2,793	1,547,405	108,815
1909	170,338	6,028	68,759	5,800	1,046,401	67,814	161,640	20,016	121,966	4,180	1,569,103	103,839
1910	180,111	5,404	100,695	8,407	940,247	55,642	163,832	20,921	117,605	3,732	1,502,491	94,107
1911	201,843	6,165	137,295	11,483	1,112,295	61,055	183,320	23,337	133,979	3,882	1,768,732	105,923

(*a*) In addition to the above, the following quantities of paraffin and paraffin wax were exported (1 = 1000); 1897, 136,069 lb. ($5,284); 1898, 166,317 lb. ($6,363); 1899, 181,861 lb. ($7,650); 1900, 157,108 lb. ($8,186); 1901, 151,695 lb. ($7,960); 1902, 175,268 lb. ($8,398); 1903, 204,120 lb. ($9,596); 1904, 174,582 lb. ($8,273); 1905, 160,836 lb. ($7,873); 1906, 173,504 lb. ($8,463); 1907, 207,504 lb. ($10,209); 1908, 141,667 lb. ($6,923); 1909, 181,328 lb. ($7,609); 1910, 199,913 lb. ($7,329); 1911, 214,018 lb. (7,048). (*b*) Reported in barrels of 42 gallons.

The oil belt extends about 25 miles east of the Copper River's mouth, and back from 4 to 8 miles from the coast. Years ago seepages were noted there, by the Indians, who carried the information to traders. In 1897 much of the land was taken up. A number of American claimants, having some of the most promising locations, formed the Alaska Development Company, which later leased to the Pacific Coal and Oil Company, a Canadian corporation, which in turn leased to the Pacific Mines Company, Ltd. This last named company did the actual operating. Drilling outfits were set at work, and four of the first five bores resulted in oil. Then each of the companies sought to oust the others.

PETROLEUM OUTPUT OF THE WORLD. (*a*)
(In metric tons.)

	1905	1906	1907	1908	1909	1910	1911
United States	18,969,000	17,862,000	22,287,985	24,401,728	24,433,528	28,331,000	29,000,000
Russia	7,505,637	8,167,934	8,435,708	7,654,600	8,037,300	(*f*)8,952,793	8,290,000
Dutch East Indies	1,062,224	1,186,907	2,200,000	2,348,000	1,497,275	1,700,000	1,590,000
Galicia	794,862	739,885	1,176,000	1,754,002	2,150,000	1,700,000	1,300,000
Rumania	614,870	887,000	1,129,097	1,147,727	1,263,946	(*f*)1,352,300	1,540,000
India	581,519	564,470	587,000	568,000	905,336	872,000	1,043,000
Mexico	(*b*)	(*b*)	294,000	587,000	405,000	(*f*) 542,400	896,338
Other countries	350,000	367,000	296,000	293,000	505,000	(*f*) 786,480	1,000,000
Total	29,878,112	29,775,196	36,405,790	38,754,057	39,197,385	44,236,973	44,659,338

(*a*) In the above table the statistics for the United States are computed from the production reported in barrels as given in the first table of this article. As a gallon of the crude petroleum found in the United States varies in weight from 6.41 to 7.83 lb., the oil in a barrel varies from 269.22 to 328.86 lb. The arithmetical mean of these figures is 299.04 lb., which figure has been used as a factor in converting the output stated in barrels into metric tons. This is not strictly correct, because in the period from 1900 to 1910 the proportion of petroleum production of various gravities has altered materially, especially because of the largely increased production in California, Texas and Louisiana. However, the method adopted is as near an approximation as can be made at this time. (*b*) Included in other countries. (*f*) *Oil Investors' Journal.*

The Amalgamated Development Company quietly began the task of acquiring old ownerships, and in 1911 the task was completed. That company is going ahead with the development of its properties on an extensive scale. So far $200,000 has been expended, chiefly Canadian capital, in acquiring possession, and in providing equipment to handle the northern oil. Half a dozen other companies will be at work in 1912, and perhaps as many as 25 wells will be drilled.

The crude oil contains a high percentage of inflammable and illuminating products. Analyses from different wells show that the Katalla oil is from 30 to 40 per cent. gasolene, benzine or naphtha. An equal or greater proportion is kerosene.

Appalachian Field (By H. C. George[1]).—The Appalachian field, comprising the oil pools of New York, Pennsylvania, southeast Ohio, West Virginia and Kentucky, produces a high-grade petroleum with a paraffin base, rich in illuminating oil, benzine, and gasoline. In the last 5 or 6 years, nearly all of the oil pools in this district have shown a continuous decline in the number of wells drilled and the new production secured; 1911 declined to a greater extent than any previous year.

The total number of wells drilled in all pools of the Appalachian oilfield in 1911, was 4167 as compared with 5895 in 1910 and 8700 in 1909. The decline in new production was still more noticeable. The daily production from new wells in 1911 was 23,810 bbl., as compared with 62,047 bbl. in 1910, and 81,918 bbl. in 1909. The number of dry-holes increased from 25.6 per cent. in 1909 to 32.8 per cent. in 1910 and 39.2 per cent. in 1911.

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

PRODUCTION OF WELLS DRILLED IN THE APPALACHIAN OIL-FIELD IN 1909, 1910 AND 1911.

	Number of Wells Drilled.			Daily Production, Bbl.			Daily Production, Bbl. per Well Drilled.			Per Cent. of Dry-holes.		
	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.
Allegheny Co., N. Y.	468	303	201	838	446	201	1.8	1.4	1.0	7.6	16.2	33.3
Pennsylvania...	3,958	2,056	1,646	10,361	5,936	4,349	2.6	2.8	2.7	15.6	23.7	27.3
West Virginia...	1,810	1,735	1,228	35,872	35,347	10,556	19.1	20.4	8.6	36.9	40.6	48.4
Southeast Ohio.	2,285	1,682	1,178	25,239	19,321	9,363	11.0	11.5	7.9	36.2	36.6	41.2
Kentucky.....	179	119	114	2,108	997	1,541	11.7	8.4	13.5	44.7	62.2	32.4
Totals and averages.	8,700	5,895	4,167	81,918	62,047	25,810	9.4	10.5	6.2	25.6	32.8	39.2

The total production for the Appalachian oil-field in 1911 was 22,275,000 bbl., as compared with 26,089,900 bbl. in 1910, 25,394,200 bbl. in 1909 and 24,240,000 bbl. in 1908. From 1908 to 1910, each year showed an increase in production, but 1911 yielded a smaller production than any of those years. The price in 1911 was $1.30 per barrel, as compared with $1.33 in 1910, $1.62 in 1909 and $1.78 in 1908.

The only pool in the field that did not share the general decline was the small one of Kentucky which drilled about the same number of wells in 1911 as in 1910, but with a larger new production and a smaller percentage of dry-holes. In Pennsylvania, the question of drilling to greater depths in the Bradford pool in the northwest part of the state, and the finding of a few good wells at Canonsburg, Washington County, in the southwest part of the state, are the only new features for 1911.

In West Virginia, some good wells were found in Wetzel County. Considerable activity occurred in Roane and Kanawha Counties, with prospects of finding a large pool in the latter. But no large gushers, such as were secured in 1910, were found during 1911.

In southeast Ohio, as a result of the new operations of 1911, there are prospects of finding an extension to the deep-sand territory, to the south and east of its present limits. Many failures occurred in the shallow territory. In Harrison County the large gas companies were active in new operations.

In Kentucky and Tennessee much land was leased, but no increase was noted in drilling operations. Most of the work done in 1911 was in Wayne County, Ky. Central Ohio, the old Clinton Sand gas district, showed the most activity during 1911, resulting in the finding of many good wells.

California (By M. L. Requa).—Production in 1911 outran consumption and the surplus that began to manifest itself in the summer of 1910 continued more or less unabated throughout the following year. While the Lake View gusher ceased to produce, the bringing in of which

created the first signs of overproduction, this decline in output was more than counterbalanced by new wells in the Midway district. It is not probable, however, that this condition can continue much longer, because there is no incentive to drill additional wells, without which a decrease in production and a closer balance between consumption and production are inevitable.

While conditions as to overproduction remained about stationary in 1911, indications point toward a hardening of prices in 1912, and it would not be surprising if in the latter part of the year prices advanced sharply, with material gains early in 1913. On the whole, it is impossible for the oil producers in California to market their product at the existing price of 30 cents for fuel oil and 45 cents for refining oil.

CALIFORNIA OIL PRODUCTION, 1911,

Less Fuel Used.[1]

District	Bbl.
Midway	20,821,669
Coalinga	17,793,221
Kern River	12,225,273
Santa Maria	6,259,766
Fullerton	5,887,069
McKittrick	5,339,364
Maricopa	4,468,087
Salt Lake	2,815,097
Whittier-Coyote	1,106,966
Ventura	453,764
Los Angeles	436,200
Newhall	139,508
Summerland	64,875
Lost Hills	63,500
	77,874,359
Less loss and overestimates	650,000
Net for year	77,224,359
Shipments, 1911	66,665,000
Surplus	10,559,359

[1] Fuel used in the fields is estimated at 425,000 bbl. per month. The estimates of the U. S. Geol. Survey and those of the *Oil Investor's Journal* (Feby 1, 1912) are both higher than the above. The latter gives the total production as 83,743,044 bbl. for 1911 and 77,697,568 bbl. for 1910.

CALIFORNIA OIL PRODUCTION.

	1909.	1910.
	Bbl.	Bbl.
Coalinga	15,406,600	18,597,301
Kern River	14,508,292	14,725,096
Santa Maria, Lompoc and Arroya Grande	8,080,488	7,592,637
McKittrick	5,807,212	5,480,631
Fullerton-Brea Cañon	4,309,000	5,085,844
Salt Lake District	3,821,233	3,247,507
Midway	2,234,456	11,168,839
Sunset	1,999,800	9,149,159
Whittier	848,800	1,197,351
Los Angeles	529,965	465,799
Newhall and Santa Paula	516,778	653,923
Summerland	66,300	75,050
Watsonville and Sargents	63,799	36,490
Total	58,192,723	(a) 77,475,627

(a) Includes approximately 4,000,000 bbl. produced and used for fuel. Possibly 2,500,000 of total production lost by shippage, etc.

The speculative fever that was so acutely manifest during the summer and autumn of 1910 entirely subsided and there were practically no new flotations in 1911.

At the beginning of 1911 approximately 39,000,000 bbl. of oil was in storage, as compared with 28,000,000 at the end of 1910. The total production for the year was 77,224,000, as compared with 77,000,000 in 1910, an insignificant increase. The surplus for 1911 approximated 10,559,000 bbl.

The Midway field did not entirely fulfil the hopes of its most ardent supporters, but developed an important and rich field, the limits of which are now fairly well outlined; in the other developed fields of the State there were no material changes, and the new fields of Lost Hills and Bell Ridge are yet too young to hazard any estimate of their ultimate capabilities. Both of these fields, however, developed oil at comparatively shallow depths, and there seems to have been a complete upsetting of certain geological prognostications regarding this territory. The developments here indicated interesting possibilities for a large section of country between Coalinga and McKittrick which has not yet been proved or disproved, except in certain isolated areas. The Kettleman Hills fulfilled geological prophecy as to the depth at which the oil would be encountered, one well at present being over 4000 ft. in depth without any showing of oil. On the whole, the proved oil area was not increased to any large extent. It is also true that certain areas that were looked upon as extremely promising did not prove valuable.

Considerable territory in the Santa Maria field is showing a constantly increasing volume of water, and development in areas presumed to be oil-bearing indicated that the oil was at such depth as to be unprofitable at prices now prevailing. The Kern River field did not materially alter its position. Coalinga continued a reliable and steady producer, and the fields in the southern part of the state, as a whole, more than held their own.

The gas field developed in the Buena Vista Hills gave promise of large production, although no effort was made to pipe this gas to other parts of the state. The proposed pipe line to Los Angeles, which was much discussed, is as yet unconstructed. In fact, save for some construction by the Standard Oil Co., no new pipe lines were laid in 1911. Invasion of Southern California by the Standard, with a refinery at El Segundo, near Los Angeles, was the most important happening in the refining industry for the year.

As compared with a year ago, there is a marked decrease in the number of strings of tools running, notwithstanding a very large increase in drilling by the Southern Pacific. The activity of the Southern Pacific

dates from midsummer, 1911, at which time it appeared to have adopted a policy of line drilling against other producing wells, although it is understood that it will not drill lines until work has been started upon adjoining property. It is unfortunate that some equitable arrangement cannot be perfected whereby the Southern Pacific could keep its oil for future requirements. On the other hand, the draining of oil territory by independent producers unquestionably deprives it of oil beneath its territory.

The recent dissolution of the Standard Oil Co. effected no change in the dominating position of the Standard in California, and the oil producers of the state are asking themselves what real benefit was derived from this dissolution, and whether it would not have been much better if some form of regulation had been attempted that would have permitted agreements as to production and price. Certainly from a financial standpoint nothing was gained by independent producers.

The much discussed agreement between the Independents, the Union Oil Co. and the Associated Oil Co., whereby it was hoped that prices would be steadied, was apparently a complete failure. The hoped-for increased consumption which was to have benefited the Union-Independents was not forthcoming; on the contrary, there was a marked decline in consumption during some of the autumn months. Consumption by the Northern Pacific, Great Northern, and the Canadian Pacific did not materially affect the sales of the Union-Independent agency, although it is believed that, during 1912, these roads will greatly increase present consumption.

The most noticeable invasion of new territory in 1911 occurred at the plant of the Nevada Consolidated Copper Co. at Ely, Nev. This plant had been burning Utah and Wyoming coal, and had been presumed to be within the coal zone. The extension of oil burning on the Southern Pacific from Sparks to Carlin was also accomplished in 1911; this may be pushed still farther, to Montello, and possibly as far east as Ogden. It is, however, improbable that California oil can maintain a foothold in this field at the price at which producers can profitably afford to sell their oil.

Owing to a rapid increase in production and the unpreparedness of the companies to tank their oil, enormous waste occurred during the year. Well informed producers agree that 4,000,000 bbl. would be a minimum, and many believe that double that amount was lost.

Colorado.[1]—Florence, in Fremont County, and Boulder, in Boulder County, are the only two producing fields in this state. In 1911, the Florence field produced 210,000 bbl. of crude oil. There are two refin-

[1] *Min. and Eng. World*, April 6, 1912.

eries, one managed by the United Oil Company, and the other by a small independent concern. These refineries produced 2,850,000 gal. of refined oil, 180,000 gal. of gasoline, and 1,000,000 lb. of paraffin. Most of the oil is now obtained from a lower sand, the upper stratum being practically exhausted. The United Oil Company, of Denver, is said to control most of the Florence field, which covers an area of 81 square miles. The wells are fitted with storage tanks which are connected by pipe lines with the refinery. All of the 39 wells in this field are pumped. In 1911, the United Oil Company drilled 39,620 ft. at a cost of $51,647. This company helps independent drillers to prospect for oil by lending them tools and rigs, and then pays them $1 per barrel for all of the oil they produce. Four strings of tools are in operation by this company. The oils are all paraffin base. The refinery has a distilling capacity of 40,000 gal. per day.

The Boulder field, 25 to 35 miles north of Denver, has a production of 40,000 bbl. from a number of small wells. The refinery in the field manufactured 690,000 gal. of refined oil, 250,000 gal. of gasoline, and 25,000 gal. of gas and fuel oils. Very few wells were struck in 1911, and a good deal of prospecting was done in other parts of the state without success. The Rangely oil-field, in Moffat and Rio Blanco Counties in the northeastern part of the state, is promising. Seepage oil has been found, but as these are no facilities for storage or transportation, drilling has been stopped and the wells capped. In the San Luis Valley, 10 miles north of Alamosa near Mosca, the San Luis Valley Oil and Gas Co. has been doing a great deal of drilling. The Mesa Verde Oil and Gas Co. has a standard rig in operation in the Montezuma Valley, in the southwestern part of the state. Seepage oil has been struck, and the indications are very promising. Several other companies are prospecting in this field.

Illinois (By R. S. Blatchley[1]).—The production for 1911 was 30,000,000 bbl., as against 33,143,362 in 1910. The decline in 1911 is the second in the brief history of the oil business in Illinois. The first came in 1909 and was due to disturbed market conditions. Recovery was immediate in 1910 but the imminent decline of the early fields and the lack of new development in later pools in 1911 caused a second reversal. Unless new fields are discovered the decline must continue. The Ohio Oil Co. and the Tidewater Pipe Line Co. are the two pipe-line shippers; the tank-car shipments in the Illinois fields are those of the Sun Oil, Cornplanter Refining and Missouri-Illinois Oil Cos., and railroad shipments from Sandoval and Carlyle, Illinois.

The prices of the two general grades of oil remained steady from the previous year until May 2, 1911. Until this time oil of gravity over

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

30° Bé., commanded 60 cents per barrel, and under 30° Bé., 52 cents per barrel. The price increased in May for the two grades to 63 and 55 cents per barrel. A second increase for the first grade to 65 cents per barrel took place June 14. A third increase for the first and a second increase for the second grade to 67 and 57 cents took place on Sept. 15 and 19 respectively.

In the southeastern fields, the combined daily output of the Clark, Cumberland, and Edgar County wells was about 8000 bbl., as against 9000 bbl. in 1910. Crawford County showed the first step of a rapid decline in 1910. The average daily output of the county was about 18,000 bbl., as against 30,000 bbl., in 1910 and 100,000 bbl., in 1907. Lawrence County is the richest producing area in the state. There are seven sands from 750 to 1900 ft. that yield steadily. In order they are the Bridgeport Nos. 1, 2 and 3 lenses, 750 to 1000 ft.; the Buchanan sand, 1100 to 1400 ft.; the Kirkwood sand, 1350 to 1650 ft.; the Tracey sand, 1550 to 1750 ft.; and the McClosky lime, 1750 to 1900 ft. The Kirkwood and McClosky sands, especially the latter, are the richest developed in Illinois. The accumulation of oil and gas in these sands is governed by a long double plunging anticline. Lawrence County maintained her usual production without new additions to the field. The average daily yield of the field was about 42,000 bbl. Both sour and sweet oils were produced, which were handled separately.

The Sandoval field of Marion County was clearly defined in 1911, and was found to be limited to about three-fourths of a square mile. There are about 66 producing wells that have a daily yield of 1800 bbl. The oil comes from the Stein and Benoist sands in the basal portion of the Chester rocks of the Mississippian series of formations and is equivalent to the Kirkwood sand of Lawrence County. The Sandoval field lies along an irregular structural terrace upon the broad and gentle western flank of the Illinois basin. A new field was opened 3 miles northwest of Carlyle in Clinton County, in April, 1911, which proved to be the best new field for the year, east of the Mississippi. The area was described by the State Geological Survey[1] previous to opening the field. The field is governed by an elongated dome on the western flank of the Illinois basin and has an areal extent of about 1 1/2 square miles. The gravity of the oil is about 35° Bé. There are about 120 producing wells yielding about 4500 bbl. daily. The initial production of the first Carlyle wells was about 200 bbl.

In Macoupin County, two gas wells of about 100,000 cu. ft. daily, and two oil wells of 5- and 10-bbl., daily yield were completed about 3 miles south of Carlinville. The production comes from the base of the

[1] Ill. Geol. Surv., *Bull.* No. 16, pp. 87 and 167.

Pottsville rocks immediately overlying the massive St. Louis limestone. The structure, according to F. C. Kay of the Survey, seems to be a small elongated dome. The Midvalley Oil Co. of St. Louis found a showing of oil in its prospect four miles east of Pinckneyville County, late in November. A conspicuous north and south anticline has been pointed out between Duquoin and Pinckneyville by the State Geological Survey, and much drilling is contemplated along this area.

The Carlyle development caused the average initial production to rise in April and July, to the highest point in two years. Of the 1256 wells drilled in 1911, Lawrence and Crawford Counties, with 479 and 449 respectively, headed the list. On Jan. 1, 1911 it was estimated that 18,618 wells had been drilled in Illinois. Of these 2847 or 15.2 per cent. were barren. In the first 11 months of 1911, 1256 wells were drilled, with 287 or 22.8 per cent. barren. The total up to Dec. 1, 1911 was 19,874 wells drilled with 3134 or 15.7 per cent. barren. The greatest number of wells, 4988, were drilled in 1907, and in 1911, the smallest number.

Kansas and Oklahoma (By Erasmus Haworth[1]).—The production of oil in 1911 by the Mid-Continental field was the greatest since its development began, the total aggregating 57,911,525 bbl. The Prairie Oil & Gas Co. was again the greatest purchaser of oil. The Texas Oil Co. and the Gulf Oil & Pipe Line Co. were the next two largest purchasers. The independent refineries did not consume so much as they did in 1910. Compared with 1910, each of the three large purchasers handled a little more. The Prairie Oil & Gas Co. increased its storage about 1,250,000 bbl., although during a number of different months it delivered considerably more than it bought.

No developments of a startling character were made within the limits of the Mid-Continental field proper, but to the southwest just across the line in Texas, about Electra, great interest arose. Drilling was well kept up during the year throughout the entire field. A new pool was discovered north of Cleveland in Oklahoma, commonly known as the West Osage field. Here a number of strong wells were obtained, a few of which rivaled the big wells in the Glenn Pool years ago. At present, however, the field seems to be fairly well circumscribed. Oil was also found about Jenks, near the north end of the Glenn Pool which at the present time is attracting a number of drillers. It seems that the oil will be developed on both sides of the river. Recently, the Prairie Oil & Gas Co. brought in a good well reported at about 600 bbl. natural, situated near the northern edge of its tank farm at Jenks. Beyond these comparatively new pools development has been confined almost

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

OIL PRODUCTION, MID-CONTINENT FIELD.
(Kansas and Oklahoma. In barrels.)

	1908. (a)	1909. (a)	1910. (a)	1911.
Prairie Oil and Gas Company	33,266,282	28,773,727	31,597,313	34,691,298
Texas Oil Company	5,371,000	7,329,500	7,513,000	8,934,800
Gulf Oil and Pipe Line Company	5,638,300	6,728,490	7,941,500	8,669,827
Independent refineries	3,999,200	5,175,550	6,387,500	5,365,600
Fuel oil, crude				250,000
Total	48,274,782	48,007,267	53,429,313	57,911,525

(a) From *Oil, Paint and Drug Reporter.*

entirely to drilling here and there throughout the areas already exploited at the beginning of the year. Every area received considerable attention, with fairly good success, showing that the oil supply is by no means exhausted.

Almost no developments of pipe lines were made during the year. The National Refining Co. at Coffeyville, offices at Cleveland, Ohio, extended a line into the shallow field from Coffeyville. The term "shallow field" now includes all of the old time shallow field around Alluwe and Coody's Bluff, and also the northeastward extension commonly known as the Delaware field lying principally to the west of Delaware. The Gulf Pipe Line Co. and the Texas Pipe Line Co. extended their lines into this shallow pool area early in the year. Each of the purchasing companies continuously extended its gathering lines, so that oil could be marketed from all the wells throughout the entire year.

No startling developments of gas were made during the year. Drilling was kept up reasonably well both in Kansas and Oklahoma, and many new wells were found, but no distinctly new pools. A number of gas wells were obtained in Oklahoma both north and south of Cleveland, particularly in the neighborhood of the West Osage oil-field, already mentioned. Likewise throughout the well-known gas territory, many new wells were obtained. The gas pressure over the entire area is seriously reduced. The Kansas Natural Gas Co. extended its pipe lines into the Hogshooter gas field early in the winter of 1910 and 1911. During the latter part of the summer months the Kansas Natural Gas Co. applied to Kansas City for the privilege of increasing the retail price of gas from 25 cents per 1000 cu. ft. to 35 cents. No action has yet been taken upon the subject. The Kansas Natural also gave notice of an increase in the price of gas to consumers in a number of towns in Kansas.

Early in September the Public Utilities Commission of Kansas City, Mo., employed me to investigate the gas situation. The conclusions reached were substantially:

(1) That all the companies are now drawing gas from substantially the same areas. These are the Kansas Natural Gas Co.; the Oklahoma

City Pipe Line Co.; the Quapaw Pipe Line Co. carrying gas to the Joplin mining district; the Caney River Pipe Line Co., carrying gas to Wagner and Muskogee; and the Wichita Pipe Line Co., carrying gas to Winfield, Wichita, Hutchinson and Newton, Kan., and the Portland Gas Company. (2) That more than 1,000,000 people are now being supplied for heating, lighting and manufacturing purposes with natural gas from the Mid-Continental field, about 45 per cent. of which is used by factories. (3) That at the present rate of consumption, Kansas City need not hope for a supply of gas sufficient to meet its wants longer than 2 or 3 more years. (4) It seems probable that towns and villages within the gas fields may reasonably expect to have their supply continued for a decade or more after the big pipe lines practically have gone out of existence.

Barring a small supply which may yet be obtained from the Vilas field, and the Independence field, Kansas can no longer supply the pipe lines. Across the line, in Oklahoma, the pressure is low and most of the wells are waterlogged within 10 miles of Kansas except some new developments in the Osage. The area around the erstwhile famous burning Caney well is substantially exhausted. Southward in some places where the gas sands are irregular many new wells come in about as strong as ever, but throughout the great Hogshooter area, the pressure is reduced from 470 lb. to 350 lb., and the volume from an original average of 15,000,000 cu. ft. to 8,500,000 cu. ft. per well. The Hogshooter pool occupies a nicely marked anticlinal area and a number of wells along the limbs of the anticline are already waterlogged, which looks bad for the field.

Louisiana.—The production of petroleum in 1911 was about 10,400,000 bbl., an increase of 3,000,000 bbl. over the production of 1910. The large increase was due to the opening of a deeper stratum. Conservative operators estimate that the production for 1912 will double the output of 1911. A 100-mile belt in the western part of the state, from the Arkansas line to the Gulf of Mexico, is an enormous oil-field. The Caddo field, in addition to being a large producer of petroleum, is the largest gas-producing field of the United States. Owing to a lack of market for the gas, the waste has reached large proportions in the last few years. Steps have recently been taken to close the larger wells until a market can be found for the gas.

Ohio (By J. A. Bownocker).—The petroleum industry in Ohio dates from 1860 when a well was drilled on the bank of Duck creek in northern Washington County. Oil was found at a depth of less than 100 ft. and the high price of the fuel made the well very profitable. Others were sunk as rapidly as possible and a year later producers were found

at several places. From that time the drill has been at work in southeastern Ohio and the territory has become a large producer. As the work progressed, deeper and deeper sands were found until the Berea grit was penetrated; efforts have been made to find the still lower formations such as yield oil in Pennsylvania and West Virginia, but without success, and it may be accepted as proved that when the Berea is reached it is useless to go deeper.

The opening of the Sistersville pool in 1891 gave new life to the field. In less than 10 years many new pools had been located from Columbiana County on the north to Washington and Morgan on the south, making the territory one of the most prominent to oil men. By 1902 the limits of the various pools had been pretty definitely determined and since that time no really large pools have been found, save that in the "Clinton" sand to be considered later. No notable results were accomplished in this territory during 1911 and none need be expected hereafter, so extensively has the territory been explored.

During many years years drillers sought oil in the "Clinton" sand in Fairfield and counties to the north and south, but no large pool was located until 1907, when the Bremen field was discovered. First, the territory near Pleasantville, Bremen and Junction City was exploited, and while this was being done many tests were being made north and south.

During 1911 the principal development was near Gore in northeastern Hocking County and in the adjacent part of Perry. Wells having an initial production of 100 bbl. or more have been drilled but their number was small. One well was completed near Crooksville in northeastern Perry County, the "Clinton" sand being found at a depth of about 3400 ft. The production of oil started at 25 bbl. per day, a rate so small as to render the well of doubtful value.

Efforts to find oil in the "Clinton" in the western part of Muskingum County have been only partly successful. Four tests have been made near Gratiot, but only two of these were successful; one began yielding 60 bbl. of oil per day and the other 50 bbl. Nearly 40 ft. of sand was reported but its depth was about 3100 ft.

Results of tests in southwest Coshocton County were similar to those in Muskingum. Several small oil wells were secured but the "Clinton" sand lies very deep, about 3200 ft., making the expense of drilling heavy.

During 1911 a half dozen oil wells were drilled in Hopewell Township, Licking County, the territory being near that of Muskingum County referred to above. The "Clinton" sand is reported as regular and of good thickness, but the production of oil is uncertain. Two of the six wells started at only 5 bbl. per day and hence can scarcely be rated

among the producers. The best well began yielding from 50 to 75 bbl. per day. Drilling is still actively in progress.

Wayne County was tested at several places but the results as a whole were unsatisfactory. Of three wells just south of Wooster, one was dry, one yielded a good flow of gas, while the third produced about 35 bbl. of oil at the start per day, but is now yielding nearly twice that volume. All these are "Clinton" sand wells. Further tests are now being made.

Two or three years ago a small oil well or two was gotten in the "Clinton" sand in Medina County and hopes were high that a large pool would be found. Later tests, however, have been discouraging and but little work was done in 1911.

Cuyahoga County had its touch of oil excitement in 1911, the place being near Newburg, the location of deep wells drilled 25 years ago. Oil was gotten in the "lime," that is one of the great formations of the Upper Silurian, at a depth of 2508 ft. The production is reported to have started at 30 bbl. per day, the oil being of 42 degrees gravity. Information concerning this well has been difficult to get and the above figures may not be correct. A second well in this locality was completed early in 1912 and a show of oil reported. Shooting may make it profitable.

From what has been said it is clear that the results from the "Clinton" sand in 1911 were unsatisfactory so far as the production of oil is concerned. Nor is the outlook for 1912 better.

The report for the Trenton limestone field of Northwestern Ohio is the same as that for 1910, and that for 1912 will be similar to that for 1911. Some drilling was done in Hancock and Wood Counties during 1911 and the same is true for Seneca County. Occasionally a fairly satisfactory well was gotten but generally the result was disappointing. In the language of an operator of wide experience, "the principal occupation has been pulling old wells." It must be borne in mind that the limits of this field were determined about 15 years ago, and that practically every promising spot has had a drill on it.

The Lima Field (By H. C. George[1]).—Production in the Lima oil field of northwestern Ohio and eastern Indiana showed a continuation of the decline in production which has been noticed since 1907. The total production in 1911 was 4,656,000 bbl., as compared with 5,171,000 bbl. in 1910, 6,192,000 bbl. in 1909 and 7,287,000 bbl. in 1908. The average price paid for North Lima oil in 1911 was 82 cents per barrel, as compared with 83 cents in 1910, 91 cents in 1909 and $1.03 in 1908. South Lima oil has constantly brought 5 cents per barrel less.

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

PRODUCTION OF WELLS DRILLED IN THE LIMA FIELD IN 1909, 1910 AND 1911.

Year.	Number of Wells Drilled.			Daily Production Bbl.			Daily Production, Bbl. per Well Drilled.			Per Cent. of Dry Holes.		
	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.
Northwest Ohio.	917	573	561	7,771	6,344	6,471	8.4	11.1	11.3	9.6	13.4	9.5
Northeast Indiana	304	363	151	3,852	8,010	1,528	12.6	22.1	10.1	27.3	19.0	35.8
Total.....	1221	936	712	11,623	14,354	7,999	9.5	15.3	11.2	14.0	15.7	14.6

Almost the entire decline in the wells drilled and in the new production secured, is from the Indiana part of the field. In the Ohio part, the old gas end of the Finely pool in Hancock County furnished the best producers. Years ago this district was not worked, on account of the large amount of salt water associated with the oil. Years of pumping has reduced this, and has produced conditions which led to the developments of 1911.

A number of new wells were put down in Wood County in northwestern Ohio in 1911, but there as in most other parts of the Lima field, most of the wells drilled were on inner locations on old leases, which give a small but sure production.

Texas (By William B. Phillips[1]).—The production of petroleum in Texas during 1911 exceeded 10,000,000 bbl., a considerable increase over 1910 and preceding years. Activities in the older and better known fields, such as Spindle Top, Sour Lake, Saratoga, Batson, Humble and Corsicana were continued.

The most noteworthy event was the opening of the Electra oil field, Wichita County, 140 miles northwest of Fort Worth. At the close of 1911 the production in this new field was close to 10,000 bbl. per day and a pipe line was being laid to the refineries near Dallas, a distance of 160 miles. Two new oil refineries will be built at Fort Worth. The oil from the Electra field varies in gravity from 38 to 42° B., the general average being close to 41°. It is, therefore, more like the Corsicana oil than the oils from the Gulf coast. The deepest well in the region is now down 3985 ft., but is not productive. Most of the oil is now coming from a depth of 1000 to 1200 ft. although the deeper sands, 1700 to 1900 ft., are also good producers.

Notable extension occurred in the use of natural gas in the state. By far the greater production came from Clay County, 100 to 120 miles northwest of Fort Worth. Many cities and towns in north Texas are now supplied with this fuel. The price varies according to consumption and may be taken at about 25 cents per 1000 cu. ft., on the average. On

[1] *Eng. and Min. Journ.*, Jan. 6, 1912.

what appears to be reliable authority it is stated that the town of Wichita Falls will offer natural gas to industrial establishments at 1 1/2 cents per 1000 cu. ft. This gas is also obtained from Clay County, the piping distance being from 15 to 20 miles.

The natural gas fields of Clay County are north of Henrietta and around a little station known as Petrolia. This area also produces oil, but not in such amounts as the Electra field. A considerable amount of natural gas is used in and around Corsicana, but is not piped to any distance. On the Rio Grande the town of Laredo is supplied with natural gas from Webb County. Texarkana and Marshall get their gas from the Caddo field, Louisiana. Some natural gas is used at Pittsburg, Camp County, at Santa Anna, and Trickham, Coleman County, at Moran, Shackelford County, and south of San Antonio, in Bexar and McMullen Counties.

Utah.[1]—Several companies are prospecting in the Green River fields of east-central Utah. Gas has been found by the Moob Oil Co. at 460 ft., but not in large volume. It is claimed that good showings of paraffin-base oil have been found, but no producing wells have yet been brought in.

In the San Juan oil field, according to the U. S. Geological Survey report on petroleum resources of Utah, a gusher was struck on March 4, 1908, and although some wells were put down to validate the title to claims, nothing noteworthy has developed, owing to the scarcity of water and the inaccessibility of this field.

Petroleum in Foreign Countries

Australia.[2]—As a result of the government bonus paid on kerosene and paraffin wax, amounting to 2d. per gallon on the kerosene and 2 s. 6 d. per hundredweight on paraffin wax, kerosene is being produced from shale at the rate of about 3,000,000 gal. per year. Three companies are operating: The Commonwealth Oil Corporation, at Newnes; the British-Australian Co., at Temi; and the Tasmanian Oil and Shale Co., at Latrobe.

Canada.—The decline in the production of the petroleum wells of Ontario continued during 1911, and the yield was only 10,188,219 gal. as against 11,056,337 gal. in 1910. Twenty years ago the output was 35,000,000 gal. The great bulk of the crude product of Ontario has come from the Petrolia, Oil Springs and Bothwell fields, which have been yielding oil for over 40 years. Here the oil-bearing rocks belong to the Corniferous formation (Onondaga limestone), and the wells are

[1] *Salt Lake Min. Review*, May 30, 1912.
[2] *Journ. Soc. Chem. Ind.*, Jan. 15, 1912.

not of great depth, say 400 or 425 ft. In the newest field of Tilbury and Romney the oil is more deeply seated, being found mainly in the Guelph formation. This field is showing a much more rapid falling off than the older ones. The latest pool, in the township of Onondaga, has not so far made a large production. The oil here is found in the White Medina, a still earlier formation. There is a little flurry at Delaware, where shows are obtained from the Corniferous. The thickness of the oil-bearing rocks from the Trenton up to the Corniferous is great, as is also their superficial extent, and these conditions provide a fair chance of striking new oil-fields. So far all the Ontario oils have been of the paraffin-base variety.

The natural-gas area of Ontario is steadily spreading, and now covers a large part of the northern shore of Lake Erie. There are two fairly distinct fields —one in the counties of Welland, Haldimand and Norfolk, and the other in the counties of Kent and Essex. In 1910 there were 828 producing wells in all, of which 781 were in the former and 47 in the latter field. A population of about 250,000 in southwestern Ontario enjoys the advantages of natural gas both for fuel and lighting purposes, the gas being piped to the cities, towns and villages for some considerable distance outside of the gas fields. It is also used to some extent in manufacturing industries. The production of Ontario in 1910 was valued at 1,346,471 at an average of 20.5 cents per 1000 cu. ft. and the output for 1911 was worth $1,724,258. Alberta, in 1911, also yielded gas to the value of $96,665.

Japan.[1]—The total production in the fields of Echigo province, or Niigata prefecture, in 1910, amounted to 42,736,480 gal. a decrease of 2,369,480 gal. from 1909. The output of illuminating oils for 1910 was 16,871,640 gal., a decrease of 5,641,280 gal. As there was quite an increase in machine and heavy oils, the total decrease in refined products was not over 2,400,000 gal. The conservative policy followed by the Hoden Petroleum Co. and the Japanese Petroleum Co., which have enlarged cautiously as their capital increased, accounts for the unsatisfactory condition of the industry. Active search is being made by all the companies in Echigo for new oil territory.

Mexico (By S. H. Ball[2]).—The Tampico oil-field, at which development began 10 years ago, is on the shore of the Gulf of Mexico, extending 35 miles to the west and 130 miles south from the city of Tampico. The country is marked by asphalt seepages.

The Huasteca Petroleum Company's wells at Juan Casiano, seven in number, include four producing wells reported to have flowed when brought in, respectively 35,000, 12,000, 800 and 600 bbl. per day. From

[1] *Journ. Soc. Chem. Ind.*, May 31, 1911.
[2] *Eng. and Min. Journ.*, May 13, 1911.

September, 1910 (when the pipe line was completed) to the middle of March, 1911, 2,860,000 bbl. of oil, about 16,000 bbl. daily, had been delivered by pipe line at Tampico. Production is limited to the capacity of the pipe line. The wells are from 1700 to 1900 ft. deep. The oil has an asphaltic base, is 20 degrees B. and in part is being refined at the Waters-Pierce refinery at Tampico, in part stored at Tampico, in part going to the Mexican National Gas Co., Mexico City and in part to Ebano, where it is mixed with the Ebano oil and delivered to the National Railways of Mexico. All of these wells have been brought in during the last 2 years.

The Ebano wells of the Mexican Petroleum Co., a sister corporation to the Huasteca Petroleum Company, both being Doheny & Canfield properties, are 20 in number which at present market about 3700 bbl. of fuel oil (9 degrees B.) per day. The No. 3 Chijol well, 6 miles south of Ebano, the largest, is reported to flow 800 bbl. per day of 14 degrees oil. The oil is more or less mixed with water and pumping is necessary in most wells, the product being afterward settled. These wells, which are said to be from 1650 to 1900 ft. deep, are the oldest in the district and have been producing about 9 years.

The new Pearson (Mexican Eagle Company) gusher, at Potrero del Llano, came in on Dec. 27, 1910, and before it was successfully controlled, approximately 1,000,000 bbl. of oil escaped. The well is now under partial control, being capped but not shut in. The oil is being discharged into an earthen tank of 5,000,000 bbl. capacity. Early in February, 1911, it was gushing an 8-in. column of oil to a height of 640 ft., the daily flow then being probably 35,000 bbl.; while on March 16 it was estimated to be making from 75,000 to 80,000 bbl. per day. The oil is 22° and has an asphalt-paraffin base. There are two other older wells nearby producing 600 and 800 bbl. (No. 1 and 2) respectively. The wells are reported to be 2000 ft. deep, and No. 4 is said to be 200 ft. into the oil-bearing limestones.

The Dos Bocas gusher of the Mexican Eagle Oil Co. is situated near Tamiahua lagoon. This was a huge gusher which, at its zenith, is said to have spurted oil 1500 ft. high, and at one time to have flowed at least 90,000 bbl. per day.

The East Coast Oil Co. (Southern Pacific) well at Topila had flowed about 50 bbl. per day and was being deepened when on March 19, 1911, a steady flow was developed. It was capped, but through a small leak made about 200 bbl. per day. On March 25 it was opened and this intermittent gusher made from 5000 to 6000 bbl. per day. The same company 200 ft. away has a 50-bbl. well and near by an unsuccessful hole, while the Waters-Pierce company across the creek has a small well of about

the same capacity and one crooked hole. The oil is 20° B. The big well is a little over 2300 ft. deep, the other Southern Pacific well 2693 ft. deep, yielding more or less oil from 2400 ft. down.

The field is too young to determine the average life of the wells. Some wells, like Dos Bocas, have been short lived, but in its 90 days this gusher is estimated to have thrown out almost 9,000,000 bbl. of oil, equivalent to a life of 25 years at 1000 bbl. per day. The Ebano field alone has been producting for more than two years. About 4 years ago its product was 10,000 bbl. per day; at present from 2500 to 5000 bbl. per day. Taking 3700 as the present production, this field appears, after reaching its maximum production, to have lost yearly about 25 per cent. This would indicate that in 10 years only the largest wells will be producers. Few fields have had so many large gushers, and as a result the average yield of the wells struck to date is high. A normal well appears to give from 500 to 1000 bbl. of oil per day.

The oil is not of high grade, and for the most part has a predominant asphaltic base. The oil, black in color and viscous, in a general way becomes better from north to south, that of the Ebano fields being low grade, while that of Potrero del Llano is much better, having a mixed asphalt and paraffin base. While the oil of this well, as well as that of a part of the Juan Casiano field, is being refined, the Tampico field is believed to be a fuel-oil producer.

The estimated oil production of Mexico in 1907 was 975,000 bbl. and 3,500,000 bbl. in 1910. In January, 1911, about 17,000 bbl. were being marketed daily, and in March, 1911, over 25,000 bbl. The 1911 shipments were 6,586,916 bbl.[1] If all of the wells continued to produce at the present rate, and if all were marketed (neither of these conditions, however, is probable), the output this year should exceed the local demand for oil; hence the future depends largely on creating a market in Europe for fuel oil, and in supplying not only the British Admiralty, but also the big steamship lines with fuel oil.

In March, 1911, three of the Tampico wells daily gushed a quantity of oil greater than the daily production of the 1500 Illinois oil wells. On Dec. 17, 1910, a contract for 2 1/2 million barrels of crude oil at 92 1/2 cents per barrel was made between the Ebano (Mexican Petroleum Co.) and the Waters-Pierce Co. Such favorable contracts will, however, not be made in the near future, 50 cents per barrel being considered a fair price.

Drillers get $7 per day and their American helpers, usually four in number, $4, in addition to board and lodging; they drill 11 hours. For ordinary labor the Mexicans, fairly efficient, receive 1.25 to 1.50 pesos per day. As wells may go over 3000 ft., a heavy rig is necessary, a com-

[1] *Oil Investors Journ.*, Feb. 22, 1912.

bined standard and rotary being preferable. The rotary is used for 1500 to 1800 ft. in the soft layers, and the standard rig to finish the hole. Holes will take from 3 to 6 months, the Huasteca wells, 1800 to 1900 ft. deep, having required about 4 months. Drilling is not particularly difficult if the wells do not encounter basalt. Wells in the latter rock go crooked and those in shale should be cased immediately, as the shale swells. The greatest danger appears to be from excessive gas pressure, which may blow out the casing and ruin the well. Drilling costs to date have been exceedingly high, and probably the majority of the wells have cost from $9 to $12 per ft. With careful management companies doing a large amount of drilling should be able to drill wells to 2700 ft. at about $6 per foot. Wood for fuel is abundant.

Tampico is situated on the Panuco River, about 6 miles from the coast, and steamers of from 8000 to 10,000 tons and of 22-ft. draft, regularly call at the port.

At the present time there are no custom pipe lines in the district. The Huasteca Petroleum Co. has an 8-in. line 105 1/2 killometers long between its wells at Juan Casiano and Tampico. A branch line, now abandoned, goes to Presidio. This pipe line, with its six pump stations, is said to have cost 14,300 pesos per kilometer. This same company has surveyed a railroad from Cerro Azul through Jaun Casiano to San Geromino, near Dos Bocas.

The Mexican Eagle Oil Company has laid a 6-in. line from its wells at Potrero del Llano to the Tancochin River and has projected a second pipe line (8-in.) from the former point to Tuxpan. The same company has a 6-in. pipe line from Tanguijo to the lagoon and Furber has a 6-in. pipe line (86 kilometers long) and a narrow-gage railroad from his field to a point on the coast east of Tuxpan. The East Coast Oil Company has a pipe line 8000 ft. from its Panuco wells to the river of that name. It is rumored that the Huasteca Petroleum Company is soon to start a pipe line from the Juan Casiano wells to Mexico City, a concession having already been granted. Presumably the first stage of this pipe line will merely reach the Mexican Central Railroad.

At Tampico, the Huasteca Petroleum Company has 30 steel tanks, each of 55,000 bbl. capacity, and in addition to other similar tanks, intends building a concrete reservoir at Tampico of 1,250,000 bbl. capacity, another at Geromino, where its pipe line reaches Tamiahua lagoon, of 750,000 bbl. and still another at Juan Casiano of 500,000 bbl.

Some of the better oil is being refined, the refineries having the following daily capacities: Minatitlan (S. Pearson & Sons), 4000 to 6000 bbl.; Veracruz (Waters Pierce), 3000 bbl.; Tampico (Waters Pierce), 8000 bbl.; total, 15,000 to 17,000 bbl. The Minatitlan refinery is on

the gulf, 350 miles from Tampico. It is being supplied by the Potrero del Llano, Furber and San Cristobal fields. The Waters-Pierce refineries treat Juan Casiano oil. At Ebano there is an asphalt plant, the product being largely used in Mexico, although a little is exported.

The majority of oil wells have been put down on land leased from the Mexican farmers. The leases are taken ordinarily for 30 years; a yearly rental, commonly paid 2 or 3 years in advance, and ordinarily from 5 to 10 per cent. of the oil being the consideration; 10 per cent. royalties prevail in the northern, while 5 per cent. is the ordinary royalty in the southern part of the field, except near the best asphalt seeps. The money consideration varies widely, according to the situation of the land, and is rapidly rising. In the early days many thousands of acres were leased as low as 50 cents per acre.

Peru.—The principal oil development has occurred in four fields. The Negritos, or Talara field lies on the coast about 40 miles northwest of Paita, the port for the department of Piura. Talara, where the refinery and wharves are situated, is a fine harbor connected by a 16-mile narrow-gage railway with Negritos. After the volatile constituents of the oil have been extracted, the residue is sold as fuel oil, mostly in Chile, whither it is taken in tank steamers for use in the nitrate works. The crude oil produced in these fields in 1909 amounted to over 100,000 tons. Some 25 miles inland from Negritos is La Brea, where a black and viscid substance like asphalt is found. So far this has been evaporated and used for coating the inside of tubs for holding the spirits called "pisco," or "aguardiente," and in waterproofing the company's pipes, etc. It possesses good lubricating properties.

The Zorritos field lies about 25 miles south of Tumbes in the extreme northern part of Peru, and is on the ocean. At the end of 1910 there were about 45 producing wells, three of which were flowing, and producing about 15,000 tons per annum. The new wells that have been sunk into the deep sand continue to flow freely without pumping, and the same results are being obtained from experiments made in adjoining territory.

In 1910, 27,000 acres of oil land at Cabo Blanco, north of the Lobitos field, were added to the 750 acres of the original Zorritos property, and an addition to the distilling plant increased the producing capacity by 10,000 gal. of kerosene per month. Italian natural-gas engines for well boring have been introduced, and the use of natural gas permits a great saving of fuel oil, for which the demand is already much greater than the supply. The Lobitos field is situated in the department of Piura and is owned by the Lobitos Oil-fields, Ltd., a British company. At the end of 1910 the company was pumping 92 wells. The quality

of the oil is high, and it is all sold at good prices, chiefly in Peru, Chile, and the United States. The total output of crude petroleum of the Lobitos Oil-fields, Ltd., for 1909 and 1910 was 57,226 and 53,344 tons, respectively.

The fourth important field is that in the department of Puno, near Huancane. The oil of Puno has a paraffin base, while that of the departments of Tumbes and Piura has an asphalt base. Petroleum is also found in the districts of Pirin, Chimbote, Jauja, Huancavelica, and Ica. In fact, it seems to extend through the mineral belt of Peru at intervals from Tumbes in the north to Lake Titicaca, on the Bolivian frontier, in the south. The Peruvian wells along the coast run from out in the sea to many miles inland, varying in depth from 250 to 3048 ft. The total production for 1910 was: Crude oil, 175,000 metric tons; kerosene, 1,050,000 gal.; gasoline and benzine, 155,000 gal.

Roumania.—The petroleum production is reported as 10,199,144 bbl. in 1911, as compared with 9,722,958 bbl. (1,352,289 metric tons) in 1910. The increase was due to the development of new fields and new wells. The older established pools showed a decline for the year, which was particularly noticeable in the first half-year about Moreni and Tzintea in the Prahova district. Later some heavy wells were brought in on the Moreni pool, and this pool with its very few wells, maintains the lead in Roumanian production. The Dambovitza, Buzau and Bacau districts showed increases. Prices for crude were 35.5 to 37 Lei per ton for Bustenari and 25 to 27.5 Lei for Tzintea.

Russia.[1]—The statistical bureau of owners of oil fields in Baku, Caucasus, estimates a shortage in 1911 from all Russian oil fields of 4,617,226 bbl. The Tcheleken Island production is about the same as in 1910, and Maikop shows an increase of 659,604 bbl. The Maikop oil is considered the best quality, yielding more hydrocarbons of low boiling-points; the Grozny oil ranks next while the Baku is the heaviest. The Maikop oil field in South Russia is being developed chiefly by British capital. This field covers an area of 39 square miles, but the possible oil-bearing land is estimated to cover 660 square miles. The principal groups of English capitalists are: The Anglo-Maikop Oil Corporation; The Maikop Trust, and The London Maikop and Australian Maikop. The Nobels and Rothchilds are supposed to be in secret combination with a great many of the other interests.

Several good wells have been struck in this field since the original one came in on Sept. 12, 1909, at 240 ft. The Anglo-Maikop Oil Corporation brought in the second gusher on Feb. 27, 1910, at 214 ft., which is said to be producing 300 bbl. per day. The Spies Company brought

[1] P. C. A. Stewart, *Eng. and Min. Journ.*, Feb. 25, 1911.

in a well in November, 1910. The oil ranges from 0.924 to 0.864 in specific gravity.

Trinidad.—The first shipments of petroleum from Trinidad are said to have reached 7,282,617 gal. during the first 6 months of 1911. American contractors are erecting large storage tanks at Point Fontin and a refinery is to be erected at Tabaquite. Guapo is also being developed.

The Refining of Petroleum

Distillation.—It is only within the last few years that methods of refining petroleum have taken an impetus, and we are now in the midst of a revolutionary period in this regard.

Perhaps the most noteworthy development has been due to the so-called "tower still." In former years, and even now to a great extent, most crude stills were fitted with one water condenser, and the separation of the different raw products, naphtha, kerosene, etc., was made by running the distillate into a different tank when the "stream" reached a certain predetermined gravity. The original charge of crude oil was distilled until there remained in the still a residue, known as tar, representing 10 to 15 per cent. of the charge. This tar was then transferred to a tar storage tank, where considerable heat was lost. It was finally charged into "tar stills" of similar construction, and distilled to coke. The distillate, known as lubricating distillate, was then chilled and filter-pressed to remove the paraffin wax, while the pressed distillate was reduced in the reducing stills in a similar manner, except here steam was always introduced to prevent decomposition of the oil. The distillate from the reducing stills was separated into fuel and gas oils, and light lubricating oils, which were again often distilled in the same way. This continual distilling naturally broke down the hydrocarbons, yielding hydrocarbons of lower and higher specific gravities than existed originally in the oil, therefore producing more stock, merchantable only as fuel oil.

The new tower stills vary in construction, and in the number and size of the towers, according to the ideas of the management of different refineries. The vapor lines from the stills are connected with aerial condensers, called towers, having connections at the top and bottom for the flow of the vapor. The vapor from the still passes through the goose neck (vapor line) into the bottom gas chamber of the first tower, then from the top gas chamber into the bottom of the second tower, and so on through all the towers, until the very lightest uncondensed gas passes through an ordinary water-cooled condenser. By this method, fractional condensation takes place in each tower, giving different products in

each case. The streams of condensed distillate from each tower are further separated by turning into different tanks according to the specific gravity or other tests made in the receiving house, from which the distillations are always controlled.

The advantages of these stills are necessarily obvious, viz.: closer separations of the different products are made; redistillation, with a breaking down of the oil, and a resulting loss as gas, is minimized; a large saving of fuel is effected; better control of the process is obtained; and most important of all, better products can be made.

Fuel Economy.—By the installation of waste-heat boilers, exchangers, and the collection and utilization of the uncondensed gases formed during distillation, a great saving in fuel has been accomplished. Usually one small waste-heat boiler is placed in back of every two stills, deriving its heat from the flue gases produced by them. They effect a considerable saving in pounds of coal per barrel of oil distilled. Wherever it is found practicable, the heat from hot, reduced oils is exchanged to the cold oil being charged into stills. Exchangers vary somewhat in construction, but are usually closed tanks containing a coil of pipes. They are particularly adapted to continuous distillation, where the hot oil pumped out of the last still gives up its heat to the cold oil charged into the first still.

Continuous Distillation.—In the larger refineries, stills are connected in batteries and run continuously. The charging line runs through an exchanger into the first still of the battery, near the front, at the hottest part of the still; the overflow at the back of the still runs into the next still in a similar manner, and so on through the entire battery. The pumping-out line from the last still runs back through the exchanger, where the hot, reduced oil meets with cold oil being charged into the first still, giving up its heat to it and at the same time cooling the reduced oil; heretofore, the reduced oil was always cooled by pumping through a coil cooled by water. All inlet and outlet lines are on a level, a little above the center of the ends of the stills, except in the last still, where in order to avoid contamination with the vapors, the pumping-out line is on the inside, near the bottom. In other respects, the continuous stills are fitted with goose necks and other features the same as batch stills.

Each successive still is kept at a higher temperature than the preceding one. The oil is slowly and continually charged into the first still, from which it flows through the entire battery, meeting with higher temperatures and giving off a certain proportion of distillate in each still. The advantages of the continuous system are fuel economy and greater capacity.

Decolorizing.—The two general methods of treating oils to color are by sulphuric acid, and filtration through fullers earth. In the treatment of oils with sulphuric acid, considerable action takes place and the reaction is a very complex one. The acid removes the asphaltic matter, and attacks and combines with many hydrocarbons forming "sulpho" compounds. After agitation with the acid, a semiliquid asphaltic mass, known as sludge acid, settles out and is drawn off. Formerly this sludge was carted away, but to-day most refineries of any size are recovering it.

The sludge acid is thoroughly agitated with hot water in lead-lined tanks, and the acid water separated from the asphaltic mass. The recovered weak acid is then concentrated in lead pans and finally in cast-iron stills to 66° B. gravity, when it is equal to the original new commercial acid for treating oil. The asphaltic residue is charged into stills, the water is distilled off and the residue melted down into a merchantable asphaltic product. By this method it is possible to recover 60 per cent. of all acid used, obtaining the asphalt as a by-product.

The old method of filtering oils was to charge the oil into the top of vertical tanks containing bone black or fullers earth. The treatment removed the asphaltic matter and therefore the color of the oil. After filtering the oil through this medium for some time, it was found that the bone black or fullers earth lost this property of decolorization. The earth, therefore, containing about 25 per cent. of oil was thrown away. The method to-day, in so far as the operation of filtering is concerned, is substantially the same, except that formerly where is was thought that the filters must be of rather small diameter, in order to get efficient service, they are now many times larger. Then again, most filtering is done under pressure, whereas before, it was done by gravity.

After a certain quantity of oil has passed through the filters, and the color begins to run dark for that particular grade, it has been found that the fullers earth retains considerable decolorizing power for oils of a darker grade. Either the first kind of oil is filtered through until the life of the earth is spent, or another grade of about the same test, but to be of a darker color, is charged in, and so on.

The oil held by the earth is extracted by filtering some solvent, such as naphtha, through the filter, the naphtha being recovered by distillation. The naphtha left in the filter is recovered by passing steam through the earth. The steam and naphtha gas pass through a condenser where the naphtha is separated. Even after this treatment, the earth retains about 3 per cent. of asphaltic matter (insoluble in the naphtha under these conditions) which render it valueless. The earth is passed through a retort, where by careful heating the asphaltic matter is

burned off and the earth revivified. In this manner fullers earth can be used over and over again, but the continual heating finally destroys its property of decolorization.

The property of fullers earth of decolorizing oils, is known as "adsorption." The action is physical, the asphaltic matter merely being held by the earth. A study of different earths has shown that an earth good for one oil may not be good for another. By diligent experimentation, a great deal of valuable data has been collected in this field, and most refineries are familiar with the fullers earth best suited to their products.

General.—The large number of gas engines in use has increased the demand for gasoline. Considerable progress has been made in increasing the yield of gasoline from petroleum, by what is known in oil refinery parlance as "cracking," a term given to the process of breaking down high boiling-point hydrocarbons, obtaining thereby, low boiling-point products, like gasoline. Several methods have been set forth, such as heating to elevated temperatures under pressure, and exposing to high temperatures in the presence of catalytic agents, but as yet, large yields have not been obtained and considerable fixed gas is formed during the operation.

Lubricating oil has received a great deal of attention, and many new excellent products have been put on the market. Paraffine base lubricating oils of high viscosity and low cold tests, that are well up in gravity, which were formerly almost impossible to make, can now be obtained at almost any garage.

Particular attention has been given to the manufacture of oils that will not carbonize in the gas engine, where they are exposed to very high temperatures and pressures. Improvements in this direction have been very gratifying, and to-day there should be no reason, providing the consumer knows where and how to obtain good oil, for complaining of the cylinders being choked with carbon.

In Russia they are recovering the naphthene acids from the waste "lye" obtained in treating oil. The acids, like sodium salts, possess emulsifying properties, and are therefore used in soaps.

Some small refineries are "skimming" their crude, a term applied to the process of distilling off the light fractions, such as gasoline, naphtha, etc., and selling the reduced oil. This cannot, however, be classed as an improvement in refining, and is used only by the small plants, which lack capital for the installation of proper refining facilities.

To sum up the improvements in refining petroleum, not many startling things of a progressive nature have been accomplished. Too much attention has been paid to quick practical results, and too little to careful investigation.

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PHOSPHATE ROCK

The marketed output of phosphate rock in 1911 amounted to 3,123,584 long tons, worth $12,237,111, an increase of 16 1/2 per cent. in quantity, but of only 2 1/3 per cent. in value, as compared with 1910. The principal gain occurred in the Florida land pebble field, assisted by a slight increase in Tennessee brown rock. Market conditions were reported as generally unsatisfactory throughout the producing districts, and prices fell from an average of $4.11 for all grades of rock in 1910 to $3.90 in 1911.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1908		1909		1910		1911	
Phosphate.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Florida hard rock...	642,259	$6,262,025	527,582	$4,136,243	392,068	$2,940,660	474,094	$2,953,606
Florida land pebble .	1,026,392	5,131,960	1,334,569	4,751,066	1,637,709	6,550,836	2,020,477	6,809,007
Florida river pebble.	5,000	17,500	.....	.....	.....	.....	.....	.....
Total, Florida.....	1,673,651	$11,411,485	1,862,151	$8,887,209	2,029,797	$9,491,496	2,494,571	$9,762,613
S. Car. land rock....	250,000	$1,687,500	197,000	$648,838	185,000	$786,250	(b)169,156	$673,156
S. Car. river rock...	40,000	240,000	9,500	33,250	16,347	57,314	*nil*	.....
Total S. Carolina..	290,000	$1,927,500	206,500	$701,088	201,347	$843,564	169,156	$673,156
Tennessee.....	403,180	$1,673,197	388,380	$1,456,425	440,699	$1,586,516	449,352	$1,761,460
Other States.....	8,200	28,700	6,735	23,500	10,095	40,380	(c) 10,505	39,882
Total, U. S.....	2,375,031	$15,040,882	2,463,766	$11,068,222	2,681,938	$11,961,956	3,123,584	$12,237,111

(a) Statistics collected by MINERAL INDUSTRY, except as otherwise noted. (b) Reported by the U. S. Geol. Survey. (c) Idaho, Utah and Wyoming.

STATISTICS OF PHOSPHATES IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production.	Imports.	Exports. (b)	Consumption.	Year.	Production.	Imports.	Exports. (b)	Consumption.
1901...	1,483,723	180,714	729,539	934,898	1907..	2,251,459	25,896	1,018,212	1,259,143
1902...	1,600,813	145,793	802,086	944,250	1908..	2,375,031	26,734	1,196,175	1,205,590
1903...	1,581,576	153,972	785,259	950,289	1909..	2,463,766	11,903	1,020,556	1,455,113
1904...	1,874,428	166,090	842,484	1,198,034	1910..	2,681,938	19,319	1,083,037	1,618,220
1905...	1,933,286	82,072	934,940	1,080,418	1911..	3,123,584	16,153	1,246,577	1,893,160
1906...	2,052,742	46,228	904,214	1,194,756					

(a) Production statistics of 1901 and subsequent years, except 1905–1911, are those of the U. S. Geological Survey and are based on marketed products. (b) Neglecting the insignificant re-exports of foreign product.

IMPORTS OF FERTILIZERS INTO THE UNITED STATES.
(In tons of 2240 lb.)

	1908.		1909.		1910.		1911.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Guano	31,469	$420,724	37,766	$734,636	44,268	$699,949	35,869	$774,313
Crude phosphates	26,734	175,365	11,903	97,277	19,319	235,040	16,153	157,394
All other fertilizers (a)	.....	4,394,434	.....	5,770,454	.....	6,870,164	.....	5,169,694

(a) Except potash salts.

Phosphate Mining in the United States

Arkansas.—The phosphate deposits of Arkansas are not generally regarded as of great economic importance. Compared with the product of the Tennessee and Florida fields the rock is rather low grade. The deposits are well situated to supply the growing demand for fertilizers west of the Mississippi River, and, though much of the material is too low in phosphoric acid and too high in iron to make it desirable for the manufacture of super-phosphate, the increasing consumption of ground rock phosphate for agricultural purposes will no doubt hasten further development in these fields.

The portion of the phosphate fields of Arkansas now being worked lies in the northwestern part of Independence County, along Laferty creek, north and east of the White River. The deposits, however, extend over a considerable area in the north-central part of the State, and the phosphate horizon has been recognized in Stone, Izard, Searcy, Marion, Baxter, and Newton Counties.

Only one company has mined Arkansas phosphate to any extent. At a point 12 miles northwest of Batesville, near the junction of East and West Lafferty creeks, the phosphate has been opened by nine tunnels run into the hill on the west side of the creek. Numerous rooms branch out from these main tunnels, and 50,000 tons of rock have been taken out. The stratum of high-grade phosphate here has an average thickness of 3 1/2 to 4 ft. As the phosphate is traced southward on the west side of the creek the beds dip rather sharply. On the east side of the creek the beds are nearly horizontal and considerably thicker, a 6-ft. stratum being in evidence for 1/2 mile along the hillside.

A fertilizer company of Little Rock, Ark., is preparing to enlarge an already extensive plant and is contemplating the erection of a sulphuric-acid factory. A number of other companies and individuals have large interests in these fields, and although some of them were bought primarily to develop the manganese deposits, they will no doubt handle the phosphate rock as the demand for this material increases.

Florida (By E. H. Sellards).—The production of phosphate rock in Florida during 1911 was again in excess of that of any preceding year. The production for 1910 was approximately 2,029,797 long tons, while the output for 1911 was 2,494,571 long tons, an increase of nearly a half million tons. The increase occurred in both the hard rock and the pebble fields, although the percentage of increase was greater in the pebble field. The amount of phosphate rock consigned for use in the United States showed a further decided increase. The total domestic shipment of phosphate rock during 1910 from Florida was 1,014,473 tons, while during 1911 the domestic shipments amounted to 1,290,779 tons, from which it appears that slightly more than one-half of the phosphate produced in Florida is consigned to points within the United States.

Hard Rock Phosphate.—The year 1911 witnessed renewed activity in the mining of hard rock phosphate in Florida, with the result that the output was appreciably increased over that of the preceding year. The total output of hard rock phosphate in Florida for the year 1911 was 474,094 tons. Comparative figures are contained in the accompanying table. Fourteen companies operated in the hard-rock section during 1911.

A notable feature in the development of the hard-rock industry was the construction of a dam across the Withlacoochee River, as a result of which it has been possible to supply electric power to a number of the phosphate plants in Citrus and Marion Counties. With the aid of electric power and light some of the mines have been able to work day and night shifts, thereby materially increasing the output. The practice of removing overburden by hydraulics has also been extended during the year in the hard-rock section, and has further facilitated production.

Domestic shipment of hard rock during 1911 was 16,723 tons, as against 18,745 tons during the preceding year. The amount of hard

STATISTICS OF FLORIDA PHOSPHATE.
(In Long Tons.)

Year.	Production.			Total Shipments	Domestic Consignments	Exports
	Hard Rock	Pebble	Total			
1908	768,011	1,150,000	1,918,011	1,531,520	431,681	1,101,271
1909	527,582	1,334,569	1,862,151	1,843,203	837,217	1,005,986
1910	392,088	1,637,709	2,029,797	2,081,936	1,014,473	1,067,463
1911	474,094	2,020,477	2,494,571	2,456,440	1,290,779	1,165,661

rock exported during 1911, as reported by the producers was 462,072 tons, as against 461,353 tons during the preceding year. The total amount of hard rock phosphate shipped is in excess of the amount mined, indicating a further reduction of the stock on hand.

Pebble Phosphate.—The production of pebble phosphate, which has steadily increased year by year, shows a further increase during 1911. The total output for 1910 was 1,637,709 long tons. The output for 1911 was 2,020,477 long tons, thus verifying the prediction of a year ago that the production of pebble rock during 1911 would exceed 2,000,000 tons. Seventeen companies operated in the pebble phosphate fields, although only sixteen were actually producing rock during 1911.

The domestic shipments of land pebble rock during 1911 amounted to 1,274,056 tons, as against 703,589 tons exported, thus contrasting with the hard rock phosphate, practically all of which is exported. The domestic shipments of pebble rock during 1910 amounted to 995,728 long tons, while the amount exported was 606,110 long tons. These figures indicate an increased percentage of rock consumed in the United States.

The statistics on the production and shipments of phosphate rock have been supplied direct by the producers, and are complete for all plants operating in Florida. Thirty-one companies in all were engaged in mining phosphate in Florida during 1911. In addition to these the Amalgamated Phosphate Company began the construction of a plant for the production of pebble phosphate.

Idaho.—The only locality at which phosphate deposits have been developed on a commercial basis is the Waterloo mine, near Montpelier, at which the San Francisco Chemical Co. has been maintaining, for the past 5 years, an annual output of about 5000 tons of rock containing 70 per cent. calcium phosphate. This is shipped to the company's plant near San Francisco for the manufacture of superphosphate. The deposit on this property averages 5 ft. thick and is estimated to contain 20,000,000 tons; the rock is exceptionally free from iron and alumina, and as it separates freely from the enclosing shales and limestones no sorting is required. The deposit is mined exactly like a coal seam pitching at 35 degrees.

In two other localities in Idaho, phosphate rock has been developed. Near Soda Springs, Bannock County, occurs one of the largest and most readily accessible deposits yet discovered. The other is found in eastern Fremont County, along the lower slopes of the Teton range; also along the southwest side of the Teton valley and on the west slope of the Big Hole Mountains, particularly among the tributaries of Canyon creek. These deposits lie 2 miles west of the Horseshoe Basin coal properties. Details of these occurrences will be found in the Thirteenth Annual Report (for 1911) of the Idaho State Mine Inspector, and in *Bulletin* 470 of the U. S. Geological Survey.

The Utah Fertilizer and Chemical Co. did some development work near Georgetown during 1911, but did not begin to ship that year.

Kentucky.—Within the last few years considerable interest has been manifested in the phosphate deposits of Kentucky, but conflicting rumors concerning the value of these fields have confused the prospective investor and discouraged mining developments. The phosphate area of the State so far examined lies in Woodford, Fayette, Scott, and Jessamine Counties, but the most thoroughly prospected properties lie in Woodford County. The Kentucky phosphate region forms part of the great Cincinnati anticline, extending from Nashville, Tenn., in a northeasterly direction through Lexington, Ky., almost to Cincinnati. South of this city it divides into two broad domes, one culminating near Nashville, and the other in Jessamine County, Ky. The phosphate rock itself varies somewhat in its physical properties. Its color ranges from a light gray to a rich chocolate brown and in texture from a compact, close-grained plate rock to porous cellular fragments and disintegrated phosphate. Most of the rock is in thin, close-grained plates, brownish-gray in color and fairly hard.

Up to the spring of 1911 work on the Kentucky phosphate area had been confined to prospecting. A small plant is now in course of construction which will start operations in 1912. The owners of phosphate lands are holding their property at high figures. This is partly due to the fact that shortly after the rock was discovered large sums were paid for options on several farms in the vicinity of Midway. These options were renewed upon payment of other large sums, but were finally allowed to lapse, owing to lack of capital to develop the properties. The farmers therefore have a somewhat exaggerated idea of the value of their farms.

Recently 2400 acres were under option or leased by companies and individuals interested in the phosphate industry. The land under lease is to be mined on the royalty basis, 25 to 50 cents being paid on every ton of rock produced, with a guaranty of a certain tonnage each year.

It is only a question of time before the Kentucky phosphate fields will be developed. The value of the deposits has not as yet been sufficiently well established to encourage the outlay of much capital, but the erection of the plant cited above will draw attention to this area, and it seems probable that its favorable location and the character of the output will put mining operations in the area on a sound footing.

Montana.—An occurrence of phosphate rock was found by the U. S. Geological Survey (described in *Bulletin* 470-A) south of Butte, in the canyon of the Big Hole River, and in the hills northwest of Melrose and south of Divide, on the Oregon Short Line. Analyses of float specimens indicated an average of 70 per cent. bone phosphate. The outcrop is

buried in talus, but appears to indicate a bed of workable thickness; only one bed has been found. The formation is probably identical with that containing phosphate beds in Idaho, Wyoming and Utah; the most advantageous features of the Montana locality are its traffic facilities and its proximity to sources of sulphuric acid at Butte and Anaconda.

In regard to the western phosphate deposits in general, it may be said that their immediate value has probably been overestimated. It has been suggested that a solution of the smeltery-smoke question is now at hand, by utilizing the Butte sulphur fumes in the preparation of fertilizers. It is necessary, however, to bear in mind that even if fertilizer were made, there would remain the important question of disposing of it. Much of the land in that region is not tilled and water is needed more than fertilizer. Transportation expense in the Northwest is high, limiting the commercial usefulness of such a product. It is cetain that adjacent agricultural operations would not consume fertilizer in sufficient quantity to make an adequate market at present.

Geologists of the United States Geological Survey have been engaged in classifying these portions of the remaining public lands. Pending their reports, great areas have been withdrawn from entry, and Congress is expected to adopt new legislation covering the disposal of such lands as are more valuable for their minerals than for agriculture. In following this policy, the following areas have been withdrawn, by reason of their phosphate resources. In Idaho, 1,167,137 acres; in Wyoming, 1,266,668 acres; in Utah, 107,745 acres; in Montana, 33,950 acres; and in Florida, 35,640 acres; a total, up to early 1912, of 2,611,140 acres.

Tennessee (By John Ruhm, Jr.).—Total shipments during 1911 were 449,352 long tons as compared with 440,699 tons in 1910, and 338,380 tons in 1909, showing that since the panic of 1907–8 there has been a steady increase and shipments have now about got back to normal. Mt. Pleasant shipments increased materially, but a falling off in the Hickman County field held down the total.

Prices were very steady at $3.50 to $4 for 72 per cent. rock and $4.25 to $4.50 for 75 per cent. rock. Rock of 78 per cent. for export commanded $5 to $6, and 62 1/2 per cent. ground rock $3.50, the ground-rock prices being on short tons, while the others are on long tons.

The Charleston Mining and Manufacturing Co. added to its holdings in Giles County and started operating their new plant at Wales. Hoover & Mason completed their plant and began regular operation on the Griffith place, near Mt. Pleasant; they also purchased a tract of phosphate land between Mt.Pleasant and Scotts Mill on the Mt. Pleasant Southern Railroad. The old Blue Grass Phosphate Co. holdings having previously been sold to the I. A. C., G. W. Killebrew, together with Jarecki Chemical

Co., Rauh & Sons, and others, undertook the development of some new properties near Mt. Pleasant and began the erection of an electrically operated plant about a mile south of Mt. Pleasant.

A large amount of prospecting was done on the properties in the Mt. Pleasant district belonging to M. E. Wheeler, the work being done under the direction of the American Agricultural Corporation which organization later took over the Wheeler properties comprising about 2000 acres. While owning lands in Giles and Hickman Counties, this is the first purchase of the American Agricultural Chemical Company in the Mt. Pleasant District (Maury County).

The Federal Chemical Company added to its improvements a grinding plant and embarked in the business of selling ground phosphate rock for direct application. This is a strong admission of the value of this material for use without treatment, as up to 1911 the Federal Chemical Company openly and strenuously opposed its use, and made every effort to discourage its application, as being worthless unless acidulated.

At the close of the year 1911 the concerns actively engaged in producing phosphate rock in Tennessee were as follows:

In Maury County—International Agricultural Corporation; Charleston Mining and Manufacturing Co.; Federal Chemical Co.; Ruhm Phosphate Mining Co.; Hoover & Mason; Central Phosphate Co.; Petrified Bone Mining Co.; International Phosphate Co.; Williams Phosphate Co.; Jackson & Cooper. In Hickman County—American Agricultural Chemical Co.; Meridian Fertilizer Co.; Volunteer State Phosphate Co.; Cornbelt Phosphate Co. In Giles County—International Agricultural Corporation; American Agricultural Chemical Co; Charleston Mining and Manufacturing Co. In Summer County—American Fertilizer Co.

Wyoming.—The deposits operated in this State are part of the same field to which the deposits in adjoining states belong. Shipments were made in 1911 from Cokeville by Duffield & Jeffs, and by the Union Phosphate Co. (Stauffer Chemical Co.), and from Sage by Bradley Bros. The combined output from Wyoming in 1911 was about 6000 tons.

Phosphate Mining in Foreign Countries

Australia.—The Mt. Lyell Mining & Railway Co., of Tasmania, operates three phosphate works, Yarraville (Victoria), Port Adelaide (South Australia), and Freemantle (Western Australia). The profit from all sources for the first half of 1911 amounted to £69,033, but in view of the strike the dividend was passed, for the first time since 1897.

PRODUCTION OF PHOSPHATE ROCK IN THE WORLD.
(In metric tons.)

	1905	1906	1907	1908	1909	1910	1911
Algeria	334,784	333,531	315,000	452,060	351,491	319,069	(d) 332,897
Dutch W. Indies	22,940	26,138	(c)	(c)	.....	.....	.....
Belgium	(b)193,305	(b)152,140	181,230	198,030	205,260	202,880	.....
Canada	1,180	930	680	1,622	998	1,341	506
Christmas Island	97,052	90,561	290,000	109,097	105,481	137,700	.....
France	476,720	469,408	375,000	485,607	397,908	333,506	.....
Norway	2,522	3,482	1,830	1,771	.....	.....	.....
Ocean and Nauru Islands.	108,615	140,000	190,000	301,160	197,922	310,625	.....
Russia	20,585	13,891	16,774	19,660	21,313	24,867	(d) 25,486
Spain	1,370	1,300	3,078	4,483	1,387	2,840	.....
Sweden	(c)	(c)	5,317	(c)	(c)	.....	.....
Tunis	559,645	796,000	1,069,000	1,300,500	1,223,512	1,286,262	a)1,446,633
United States	2,135,449	2,085,586	2,251,459	2,413,032	2,503,186	2,724,849	3,173,551

(b) Metric tons of phosphate of lime; in addition there were 315,200 cu. m. of phosphatic chalk in 1902, 350,250 cu. m. in 1903, 311,640 cu. m. in 1904, 80,380 cu. m. in 1905, and 119,450 cu. m. in 1906. (c) Statistics not available. (d) Figures furnished by Charles Michel, Paris.

Christmas Island.—This island, which is in the Recherche Archipelago, 20 miles southeast of Israelite Bay, contains phosphatic limestone. The known area is very small. Near Dandarraga, west of Moora, phosphate deposits are known to exist for a distance of 22 miles. Among them are: Weathered rocks with 0.54 to 2.10 per cent. phosphoric acid; ferriginous sandstone with iron phosphates, carrying 7 to 15 per cent. phosphoric acid; and fossil bone and coprolite rocks with 15.32 to 39.30 per cent. phosphoric acid; the last two are 7 ft. or more thick. Shipments from Christmas Island amounted to 137,700 tons in 1910, and 105,400 tons in 1909.

Egypt.—Phosphate deposits occur near the Red Sea, but the product is shipped by the Nile, not the Suez Canal. A large shipment from this region reached Rotterdam late in 1911, an average analysis of which was: P_2O_5, 31.21 (equivalent to 68.19 bone phosphate); CaO, 49.92; Al_2O_3, 0.28; Fe_2O_3, 1.50; SiO_2 (insol.), 4.26; other matter, 12.83 per cent. On a minimum basis of 64 per cent. bone phosphate, the price was 10 1/2 cents per unit, c.i.f. London. Total exports from Egypt in 1910 amounted to 2186 tons, of which the larger part went to France.

French West Indies.—Grand Connetable Island, off the coast of French Guiana, contains a deposit of phosphate rock about 7 1/2 miles from the mouth of Approuague River. The phosphate is found within a radius of 1 1/2 miles. The deposit has been worked about 40 years. No new deposit has been reported. The American company which has the concession to work the deposit is taking out 4000 to 5000 metric tons annually. A small quantity of guano is picked up on the island but the quantity is insignificant.

It may be added that an English company started to work a deposit of phosphate of alumina in the small island of Redonda, near the island

of Montserrat, British West Indies, in 1907. The better grades average 35 to 36 per cent. phosphoric acid.

Russia (By E. de Hautpick[1]).—Phosphates are produced in the Podolia, Bessarabia, Kostroma, and Kursk governments. The Podolia phosphates are considered the best in the world, and this explains the great demand for them from abroad and from Poland. A detailed description of them was given in *The Mining Journal* of May 21, 1910. The extraction of phosphates from the Podolia mines fell away in 1911, for they were exploited only in the old mines that had been worked for a long time, and the search for new phosphate deposits has not yet yielded satisfactory results. Of all the known Podolia phosphate deposits the richest are those of the Novo Ushitsky district, whence comes about two-thirds of the total yield of the region. Some famous mines here are the Zhurzhevo, the Antonovska, the Grigorovska, the Kolnachevska. After Novo Ushitsky the richest districts in the order of their importance are the Letichevsky, the Mogilievsky, and the Proskurovsky.

Amongst Kostroma phosphate mines some well-known ones are the Gorki, near the station of Kineshma, and the Komilovska, in the Emelyanskoi district. In the government of Kursk phosphates are produced in the Oboyansky district from the Kazatsky and Irubezhsky mines, in the Phatezhsky district from the Putchinsky and the Shakovsky mines, in the Shtchigrovsky district from the Kritovsky mines. In Bessarabia we have the Ozhevsky phosphate mines in the district of Xotinsky.

The statistics of the production of phosphates in Russia are to be found in the following table (in long tons):

District.	1907.	1908.	1909.	1910.	1911.
Podolia	10,520	12,870	14,125	15,545	10,485
Bessarabia	870	920	952	980	1,200
Kostroma and Kursk	5,120	5,560	5,900	7,950	13,400
Total	16,510	19,350	20,977	24,475	25,085

For the acid treatment of Podolia phosphates two new factories have been opened in the towns of Vinnitza and Mohilieff. Now, because of the insufficiency of phosphates at the local mines for those works, phosphates are imported from abroad, even from Canada. The treatment of foreign phosphates is very costly, because they require a larger quantity of sulphuric acid than the Podolia phosphates. A considerable quantity of Podolia superphosphates is used as fertilizer in Poland, not only on large estates, but also on medium-sized farms, and even on peasant lands. The price of superphosphates, quoted per unit of 1 per cent. of soluble phosphoric acid in 100 lb. delivered free at any railway station in Poland, is 2.375d. Since the production of superphosphates at the Podolia works is insufficient for the demand for fertilizers in Poland, a considerable

[1] *Mining Journal*, June 8, 1912.

quantity is imported, chiefly from Silesia and Posen. A large supply of superphosphates in 1911 came to North Russia, not only from Germany, but also from Belgium and Sweden. Germany has the advantage of cheap sulphuric acid as a by-product of the production of zinc in Silesia. The price of sulphuric acid in Russia is 5s. 7d. per 100 lb. By selling superphosphates at a low price Germany has put Russia into a difficult position in respect to the development of her phosphate industry. Last year, nevertheless, the first chemical works was opened on the Volga for making superphosphates from Kostroma phosphates with sulphuric acid from Ural pyrites. In order to stimulate the Russian phosphate industry, the Government must establish a high import duty on foreign phosphates and superphosphates, since the young Russian chemical industry is not yet capable of struggling with the Germans, notwithstanding that the requirement of the superphosphate market in Russia is rising every day, and last year alone it increased by 3000 tons.

British superphosphate houses cannot compete with the Germans because of the heavy freight charges. It is reported that a group of English capitalists proposes to open a chemical works at Odessa to convert Podolia phosphates into superphosphates. What has been stated above will show that such an enterprise is not practicable, not to mention that South Russian soil does not greatly require fertilizers. On the other hand, the erection of such a factory in East Russia, far from Germany, and disposing of the cheap Kostroma phosphates and of sulphuric acid from the Ural pyrites, should prove to be a complete success, for the land in that part of Russia needs fertilizers, the production of which, at the only existing chemical works on the Volga, is not sufficient.

Society Islands.—A French company, now exploiting the phosphate deposits at Makatea, an island of the Society group, about 120 miles northeast of Tahiti, has over 600 persons at work on the island. It is estimated that the island contains 10,000,000 tons of high-grade phosphate, averaging 82 per cent., with many million tons more of inferior quality. The force now at work in Makatea can furnish and load about 100 tons of phosphate a day, but this cannot be regularly kept up at present, owing to the almost constant rains during the summer months and to the fact that all the drying is dependent upon the sun and the crushing upon human force. An English engineer from Ocean Island has been at Makatea installing an electric plant of 240 h. p. for lighting the island, for propelling the cars carrying phosphate, and also to run the crushing and drying plant now under construction. When this undertaking is completed, early in 1912, it will be possible to turn out 15 tons each working hour. The present intention of the company is only to extract, dry, and crush the phosphate at Makatea. In 1910,

exports amounted to 10,000 tons of phosphate; in 1911, they reached 30,000 tons; and it is hoped that the 1912 shipments will reach 100,000 tons, which is the amount estimated as necessary to make the investment pay. Shipments during 1911 to the United States included 1160 tons to California and 2700 tons to Honolulu, all of which was declared at $9.65 a long ton. An export duty of 15 cents per ton is collected by the local government.

Tunis.—The combined output of Tunis and Algeria in 1911 amounted to 1,779,530 metric tons, as compared with 1,605,331 tons in 1910. The products of the several operations are brought down to the coast and shipped from the ports of Bône and Bougie, in Algeria, and from Tunis, Sousse, and Sfax, in Tunis. The Tebessa mines are tributary to Bône; the Tocqueville and Cie. Centrale mines to Bougie; while the tonnage shipped from the Tunisian ports originates at the mines at Gafsa, Kalaa-Djerda, Kalaa-es-Senam, Floridienne, Bir el Afou, Gouraya, etc. The entire output of the Gafsa mines went through Sfax until 1910, but in 1911, Sousse handled almost half their product.

SHIPMENTS OF PHOSPHATE ROCK FROM ALGERIA AND TUNIS.
(In metric tons.)

Year.	Algeria.	Tunis.	Year.	Algeria.	Tunis.
1894	49,693		1903	301,112	360,621
1895	104,603		1904	344,969	457,133
1896	142,524		1905	347,747	529,645
1897	227,870		1906	302,262	747,303
1898	269,572		1907	343,085	956,998
1899	286,681	65,209	1908	362,890	1,270,020
1900	277,896	171,288	1909	351,491	1,224,822
1901	278,185	178,019	1910	319,069	1,286,262
1902	265,964	266,553	1911(a)	332,897	1,446,633

(a) Figures furnished by Charles Michel, Paris.

Of the combined products of these fields in 1911, shipments to France amounted to 587,592 tons; to Italy, 370,167 tons; to Germany, 212,666 tons; to England, 180,627 tons; and to Spain, 102,307 metric tons. The rest went to other European countries, except 41,081 tons to Japan. None of it came directly to the United States.

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PLATINUM

By Frederick W. Horton

In 1911, as in previous years, the production of crude platinum in the United States continued to be derived chiefly from the placer deposits of California and Oregon, where it was recovered as a by-product in mining gravels and beach sands for gold. The output of metal from this source in California may be estimated at 400 oz., of which probably 85 per cent. was recovered by gold dredges in Butte, Yuba and Sacramento Counties. The remainder came from hydraulic-mining and dredging operations in Trinity, Shasta, Humboldt and Siskiyou Counties, and a few ounces from the beach-sand workings of Del Norte and Humboldt Counties. About 70 oz. of crude platinum were also derived from Oregon placers. Approximately two-thirds of this output was obtained from beach sands in Curry and Coos Counties, and one-third, from gravel deposits in Josephine and Jackson Counties.

The total domestic production of crude platinum (nuggets and dust) in 1911 may, therefore, be estimated at 470 oz. The average platinum content of this crude material was probably about 70 per cent., which would make the production in refined metal equivalent to 329 oz. as compared with 273 fine ounces from the same source in 1910. To this amount must be added the quantity of primary platinum of domestic origin derived by the Bureau of the Mint and private refineries from the treatment of platiniferous bullions. This amount is estimated at 600 fine ounces. It is impossible to give accurate figures as most of the platinum obtained by the Mint is from the refining of dental and jewelers' scrap, and that institution keeps no record of the separate sources from which it derives platinum. Further, it is impossible to state what proportion of the platinum obtained by private refineries is from domestic sources. In the estimate given above, the Mint is credited with an output of 250 oz. and the private refineries with 350 oz. of fine metal, the latter including about 60 oz. obtained from shipments of platinum ore from the Rambler mine at Holmes, Wyoming.[1]

The total production of platinum in this country in 1911 may, therefore, be reckoned as 929 fine ounces, worth $40,058 as compared with an output of 773 fine ounces, value at $25,277 in 1910. The slight increase in the production may be attributed to increased activity in

[1] The geology of the Rambler Mine and the occurrence of platinum in its ores is described by Prof. J. F. Kemp, in Mineral Resources of the U. S., 1902, pp. 244–250 (U. S. Geol. Survey).

gold dredging in the platinum-bearing areas of California and to a smaller extent to the entrance of the Rambler mine as a factor in the production. Shipments from this mine, while on a small scale, are of interest as they give the Rambler property the unique distinction of being the only lode mine in the world producing what is primarily a platinum ore on a commercial scale. In 1911 the mine was leased for a term of years to the Platinum Mining and Milling Company, which was formed to operate the property. Mr. Wells M. Ruggles, manager of the mine, reports that in 1911 several car loads of ore were shipped to the Baltimore Smelting and Rolling Company at Baltimore, Md., with gratifying results; further, that the lessees expect to build a new mill on the property in 1912 to supersede the one erected last year. This first mill was built largely on experimental lines and was not successful.[1]

While the domestic production of platinum is sufficient to supply only about 1 per cent. of the home consumption, the refining of crude platinum and the attendant manufacture of platinum wares of all descriptions is a business of considerable importance in the United States. During the last 6 months of 1911 alone, 31,306 oz. of crude platinum, valued at $1,225,292, were imported and at least 25,000 oz. more were probably brought in during the first 6 months of the year, although no exact figures can be given covering this period because up to July 1, 1911, crude platinum was grouped with ingots, bars, and other unmanufactured forms in the reports of the Customs. However, it is safe to say that at least 45,000 oz. of foreign crude platinum were refined in this country in 1911. In addition to this at least 20,000 to 25,000 oz. of platinum were refined from scrap of domestic origin. With the exception of a few concerns which refine scrap in a small way, the refiners of platinum in the United States are: The Bureau of the Mint; Baker & Co., Inc., Newark, N. J.; Croselmire & Ackor, Newark, N. J.; S. S. White Dental Mfg. Co., Prince's Bay, N. Y.; American Platinum Works, Newark, N. J.; Roessler & Hasslacher Chemical Company, Perth Amboy, N. J.; and J. Bishop & Co., Malvern, Penn.

The consumption of platinum in this country has steadily increased during the past 5 years, and to-day, it is probably larger than in any other country of the world. Without taking into account hundreds of thousands of dollars worth of scrap that annually finds its way back into the market, the consumption of platinum in this country in 1911 amounted to approximately $5,000,000 worth. Imports of unmanufactured platinum including crude, ingots, bars, sheets, wires, etc., showed

[1] The method of treating the ore in this mill is described in an article entitled "Concentration of Plantiniferous Copper Ore at the Rambler Mine, Wyoming." (*Met. and Chem. Eng.*, Feb., 1911; 2 3/4 pp.)

nly a slight gain in weight but a very substantial increase in value, mports in 1911 being 122,390 oz., valued at $4,722,752 as compared with 120,478 oz. valued at $3,320,699 in 1910. In fact, the value of hese imports was over a million dollars in excess of those in any previous year. Imports of manufactured metal which were almost wholly n the form of jewelry, were valued at $135,842 as compared with $333,965 n 1910.

With the exception of a few thousand ounces of platinum received rom Colombia, imports into this country were almost entirely of Russian netal. Nevertheless, only a few hundred ounces were received directly rom Russia, shipments being made through firms in France, Germany nd England. A comparative statement of the production, imports and consumption of platinum in the United States for a series of years, is given in the accompanying table.

STATISTICS OF PLATINUM IN THE UNITED STATES.

Year.	Production. (a)		Imports.			Consumption.
			Unmanufactured.		Manufactured.	
	Troy Oz.	*Value.*	*Troy. Oz.*	*Value.*	*Value.*	*Value.*
1896	163	$ 944	83,080	$926,678	$106,338	$1,033,960
1897	150	900	83,080	960,299	43,921	1,005,120
1898	225	3,375	101,018	1,178,142	52,283	1,233,800
1899	300	1,800	187,778	1,462,157	55,753	1,539,710
1900	400	2,500	118,919	1,728,777	36,714	1,767,991
1901	1,408	27,526	85,438	1,673,713	24,482	1,725,721
1902	94	1,814	105,450	1,950,362	37,618	1,989,794
1903	110	2,080	114,521	1,921,772	135,889	2,059,741
1904	200	4,160	103,802	1,812,242	105,636	1,920,478
1905	318	5,320	104,196	1,985,107	188,156	2,176,263
1906	1,439	45,189	137,556	3,601,021	187,639	3,797,460
1907	357	10,589	74,208	2,509,926	175,651	2,696,166
1908	750	14,350	50,768	1,096,615	134,119	1,244,984
1909	638	15,950	118,851	2,557,574	410,997	2,984,521
1910	773	25,277	120,478	3,320,699	333,965	3,679,941
1911	929	40,058	122,390	4,722,752	135,842	4,898,652

(a) Statistics of the U. S. Geological Survey, except for 1911.

Market.—As usual after the holiday season, trade at the beginning of the year was dull and there was a slight falling off in prices. During the first week in January refined platinum was quoted in New York at $38.50 per ounce and some lots were reported as offered for resale at as low as $38 as compared with an average quotation of $39.10 for the preceding 2 months. Prices remained practically steady until March 1, when there was a sharp advance to $41 per ounce with a good volume of business. The demand at the advanced price was quickly supplied, and by the last of March business was again quiet and the market weaker in tone. However, under the influence of higher prices and a strong market abroad, the market strengthened and domestic quotations gradually advanced, reaching $43 per ounce on May 17,

$44 on Aug. 16, $45 on Aug. 23, and $46 to 46.50 on Sept. 27, at which point they held with little variation until the end of the year when closing quotations were at $46 per ounce. The record of prices is one of a steady and almost unchecked advance throughout the year. The lowest average monthly price of refined platinum in New York was $39.06 per ounce in January; the highest was $46.25 in October. From that point the decline to the end of the year was only 25 cents per ounce. It may be noted by reference to the accompanying table which gives the average monthly prices of platinum in New York for a series of years, that the average for 1911 was $10.42 above that of the previous year, an advance of 32 per cent., and an increase of $18.25, or 73 per cent. over the average price in 1910.

AVERAGE MONTHLY PRICES OF PLATINUM AT NEW YORK.
(In dollars per troy ounce.)

	1906	1907	1908	1909	1910	1911
January	20.50	38.00	25.50	24.10	29.00	39.06
February	25.00	38.00	25.50	24.00	28.75	39.31
March	25.00	37.00	25.50	23.75	29.13	41.00
April	25.00	32.50	23.50	23.50	29.25	41.35
May	25.00	29.50	22.50	23.25	29.55	42.81
June	25.40	26.20	23.50	22.75	31.38	42.88
July	26.00	26.75	20.00	22.43	33.00	42.95
August	26.00	28.13	18.75	22.65	33.00	44.38
September	32.10	28.70	20.00	25.31	33.63	45.31
October	33.00	27.13	21.50	27.75	37.50	46.25
November	35.50	26.31	24.00	29.50	39.44	46.13
December	38.00	26.00	24.00	29.50	38.75	46.00
Year	28.04	28.18	22.85	24.87	32.70	43.12

Hard metal containing 10 per cent. iridium was variously quoted at prices ranging from two to three dollars per ounce above that of refined platinum. On Jan. 4, 1911, the price was $40.50 to 41 per ounce, and on Dec. 27 it was $48.50.

The unquestioned superiority of hard platinum as a setting for diamonds, pearls and other light-colored precious stones, both from an artistic standpoint and from that of durability and safeness of setting, has created a demand for it by the jewelry trade that is now firmly established. In 1911, this demand was unprecedentedly large, and was principally responsible for the steady rise in price of the metal and for the firmess of the market.

While there has been a very large increase in the demand for platinum for jewelry, the present high price of the metal will undoubtedly restrict its use in other directions. For example, in certain chemical wares and physical instruments in the manufacture of which platinum has been used extensively, its place can, in some cases, be taken by gold with little disadvantage and at less than half the cost. However, any re-

striction of demand due to using substitutes for the metal is likely to be small and will not seriously affect prices. On the contrary, in view of the limited production and the discovery of no new sources of supply, a still further rise in the price of metal may confidently be expected.

Platinum in Foreign Countries

Australia.—In New South Wales a small quantity of platinum is annually derived from gravel deposits at Fifield near Parkes and from the beach sands of the north coast. In 1910 the output was 332 oz. valued £1418. The total production to 1911 was 11,910 oz,

In Tasmania, 120 oz. of osmiridium valued at £530 were produced in 1910. In 1911 the output was 272 oz. valued at £1888.

Canada.—The platinum output of the Tulameen and Samilkameen districts of British Columbia which 20 years ago produced several thousand ounces of the metal per year, has declined to almost nothing as the more accessible and richer gravels of the district have been worked out. In 1911 the production of this region is estimated at not more than 20 oz. However, some little prospecting was carried on, and the results obtained from 40 drill holes on a property on the Tulameen River are said to warrant the installation of a dredge. The only other locality in Canada from which platinum is derived, is the Sudbury district of Ontario. Here small quantities of sperrylite, the arsenide of platinum ($PtAs_2$), occur with the pyrrhotite and chalcopyrite ores of the district. No estimate of the amount of platinum obtained from this source can be made as a large proportion of the mattes produced from these ores are exported to the United States and Great Britain for refining, and the electrolytic muds obtained from them are often mixed with muds from other lots of metal. The total production of platinum in Canada in 1911, is estimated at 50 oz.

Colombia.—The recent high prices of platinum and the gradual depletion of the Russian deposits have of late drawn special attention to the Colombian localities from which the metal is derived.[1] This country, while ranking next to Russia in its platinum production, at present supplies less than 10 per cent. of the world's output of the metal.

The production in 1911 was approximately 10,000 oz., about half of which was exported to the United States. The operation of two dredges in the Choco district is reported to have been successful but does not seem to have resulted in any large increase in the production. During the year it was reported that the Consolidated Gold Fields of South Africa and the firm of Johnson, Mathey & Co., of London, were jointly to

[1] The occurrence and methods of mining platinum in Colombia were described in some detail in Vol. XIX of the Mineral Industry.

engage in the operation of gold and platinum placers in Colombia on a considerable scale. Later, this report was in part verified by the purchase of the La Lozana and some adjoining platinum properties near Buena Ventura by the Consolidated Gold Fields of South Africa. This property will probably be systematically worked and it is confidently expected that through this and other systematic exploitations, the platinum output of the country will greatly increase during the next few years. The accompanying table shows the imports of platinum into the United States from Colombia for a series of years.

UNITED STATES IMPORTS OF PLATINUM FROM COLOMBIA.
(In troy ounces.)

Year. (a)	Quantity.	Value.
1906	2,316	$45,763
1907	5,496	100,206
1908	1,092	25,576
1909	2,392	36,440
1910	1,600	31,383
1911	5,503	147,820

(a) Fiscal year ending June 30.

Russia.—A slight decrease in the platinum output of Russia in 1911 is attributed to an insufficient labor supply in some distrtics and to the fact that some of the mines have been worked out. The shortage in labor was especially felt during the early part of the year. Later short crops in eastern Russia sent many laborers back to the mines and the number of *starateli*—prospectors and tributors—in the placers increased. According to figures reported to the Imperial Mint the production of platinum amounted to 187,008 oz. in 1911, as compared with 176,344 oz. in 1910, a gain of 10,674 oz. However, these figures are far from correct, for in addition to the reported output there is a great deal of metal that is produced and marketed surreptitiously to avoid the payment of the Government tax. The quantity thus mined and disposed of through irregular channels is rapidly decreasing due to the syndication of the mines and the greater difficulty of finding buyers. This accounts for the discrepancy between the official returns which show a considerable gain in production and estimates of actual output which show a decrease. That the production is greatly in excess of the amount shown by the official figures is indicated by custom-house returns of exports in 1911 which show that 420 poods (221,201 oz.) of crude platinum valued at $7,902,106 were exported. Further, that the amount of platinum produced in 1911 was actually less than in 1910, may be inferred from the fact that in 1910 the exports were greatly in excess of those in 1911, amounting to 518 poods (272,815 oz.) valued at $6,932,415. The exports in 1907, 1908

and 1909, are reported at 156,948, 155,908, 259,648 oz., respectively. The accompanying table gives the output as officially reported and the probably actual production for a period of years.

PRODUCTION OF PLATINUM IN RUSSIA.

Year.	Official.	Actual.	Year.	Official.	Actual.	Year.	Official.	Actual.
	oz.	oz.		oz.	oz.		oz.	oz.
1894	167,268	203,250	1900	163,060	212,500	1906	185,546	210,318
1895	141,757	290,900	1901	203,257	315,200	1907	172,064	310,000
1896	158,326	200,000	1902	197,024	380,806	1908	156,792	250,000
1897	179,879	395,200	1903	192,976	226,000	1909	164,513	275,000
1898	193,213	203,100	1904	161,950	290,120	1910	176,334	300,000
1899	191,464	380,900	1905	167,950	200,450	1911	187,008	280,000

The following table gives the production of platinum in the Urals according to districts for a number of years:

PRODUCTION OF PLATINUM IN THE URALS.
(In troy ounces.)

District.	1907.	1908.	1909.	1910.	1911.
South Verchotur	121,665	97,747	100,986	111,070	121,317
North Verchotur	7,554	13,032	14,727	11,858	11,367
Perm	36,082	36,735	39,867	46,071	46,382
Teherdinsk	4,894	7,452	7,161	6,361	(a)6,400
South Ekaterinburg	1,869	1,826	1,772	974	(b)1,042
Total	172,064	156,792	164,513	176,334	187,008

(a) Estimated. (b) Includes 2 oz. from North Ekaterinburg.

Notwithstanding the falling off in the production, the platinum-mining industry in the Urals prospered as it never has before due to the high prices received for the metal. Whereas two years ago every small operator in the Urals was either out of business or on the verge of bankruptcy, these same operators are now doing well. In place of demanding Government intervention to prevent monopolistic control, they are questioning whether such intervention is needful. The industry is so flourishing that it certainly needs no aid at present. The threatened use of alloys alleged to be satisfactory substitutes for platinum, and the reported discovery of other sources of supply in remote parts of the world are of little consequence and any substantial reduction in the prices received will only be brought about by manipulation. It is possible that the small producers in the flush of their prosperity may dread Government control, and of their own volition join the larger houses to save the industry from such regulation. These small operators will, however, promptly discover that they must either accept Government control or the control of the large European houses, for if the Government measure forbidding the exportation of unrefined platinum and providing for the

establishment of a Government refinery is withdrawn, such action will be promptly followed by a collapse in the market manipulated by the syndicated interests. Such a collapse would either force the small producers out of business or oblige them to join the syndicate at the syndicate's terms. Whether the above-mentioned bill which would involve the breaking of contracts of producers with the refiners, will be adopted by the Government is doubtful, especially so as it is strongly opposed by the large producers.

The course of the platinum market at Ekaterinburg and St. Petersburg in 1911, is shown in the accompanying table, as is also the course of the market in New York. The Russian prices which have been reduced to American weights and currency are for crude metal—83 per cent. platinum. The New York prices are for refined metal. Ekaterinburg is the primary market where purchases are made from the small producers; the output of the large operators is shipped directly to St. Petersburg which is the wholesale market.

AVERAGE PRICES OF PLATINUM IN 1911.
(In dollars per troy ounce.)

	New York Refined Platinum.	Russia, Crude Metal—83 Per Cent. Platinum.	
		St. Petersburg.	Ekaterinburg.
January	$39.06	$31.57	$31.90
February	39.31	31.96	31.85
March	41.00	32.43	32.25
April	41.35	34.69	34.27
May	42.81	34.55	35.25
June	42.88	34.71	35.06
July	42.95	35.47	34.71
August	44.38	36.78	35.44
September	45.31	37.02	37.23
October	46.25	37.39	37.60
November	46.13	37.98	37.89
December	46.00	37.78	37.60
Average for 1911	$43.12	$35.21	$35.09
Average for 1910	32.70	26.96	26.37

At the beginning of the year the Russian market was quiet. Opening prices were rather nominal at 8.50 rubles per zolotnik at Ekaterinburg and 32,500 rubles per pood at St. Petersburg, $31.96 to $31.85 per oz. respectively—for crude metal 83 per cent. platinum. Early in March a strong demand developed from abroad and there was a scarcity of the metal. The price advanced sharply to 9.10 per zolotnik at Ekaterinburg, and to 34,500 to 34,750 rubles per pood at St. Petersburg ($34.21 and 33.93 per oz. respectively). The market continued strong until early

in June with a slight advance in prices to 9.30 rubles per zolotnik at Ekaterinburg and 35,500 rubles per pood at St. Petersburg. At this time the small producers pressed sales in order to get money for their summers' work. Throughout the remainder of the year the market remained firm with a further slight advance in prices. On Dec. 24 quotations for small lots of crude metal, 83 per cent. platinum, at Ekaterinburg were 10 rubles per zolotnik—equal to $37.60 per oz.—and for large lots at St. Petersburg, 38,500 rubles per pood—$37.73 per oz.—for the same grade.

In 1911 there was practically no production of osmiridium in the Urals, the entire output amounting to only five ounces, which came from the Miassk deposits, in the Government of Orenburg. The total production of osmiridium in Russia during the past ten years is officially reported as 308 oz.

Platinum in the Russian Empire, 1911

By C. W. Purington

The product is officially stated to have been 343 poods of 83 per cent. platinum dust, worth on January 1, 1912, Rs. 38,400 or $19, 968 per pood of 526.6 oz. or $37.80 per troy ounce. At this same date pure platinum was quoted at $46.00 per ounce in New York. This product of 83 per cent. platinum and 2 per cent. gold mechanically mixed represented a money value of $6,849,000. It is very unlikely, however, that this was all the platinum produced, and probably an estimate of 280,000 oz. all told, is none too high for 1911. At this rate the value of the cruke product at Ekaterinburg was well over $10,000,000 at present quotation.

The districts from which the platinum came are all within a comparatively small area. The *Iss* River and some neighboring streams account for all the platinum that comes from the Shuvalof Estate (89 poods recorded) and the patented claims on this river which are mostly in the hands of the French Trust, namely the Société Anonyme d'Industrie du Platine. It is safe to say that 80 per cent. of the Ural platinum comes from the Iss and Pura Rivers. The Demidof or Tagil Estate 50 miles to the south of the Iss, produces the bulk of the remainder. Smaller producers are the Nikolai Pavdinsk, Polyakof and Syssert Estates, and numerous smaller producers on the Iss, Tura, Lyalya, Nyasma, and some other streams make up the output.[1]

Platinum mining is hampered by several obstructions and it is interesting to consider what the future product is likely to be in view of the peculiar conditions which obtain.

[1] See *E. and M. Journal*, May 5 and 12, 1904, for list of localities.

The Iss River heads in an immense tract of land known as the Shuvalof Estate, and flows for 15 miles through this Estate. The Estate contains great reserves of alluvial platinum, and numerous attempts have been made to buy the property from the owners, so far without success. The French Trust has a contract to take all the product of the Shuvalof property for a term of years.

The lower course of the Iss, some 25 miles, is mostly located under patented claims, averaging about 200 acres each. Most of these belong to the French Company, but some belong to private individuals. I estimated[1] that the product of the Iss up to 1897 had averaged 60 cents (at $10.18 per oz.) per cubic yard, for an average width of 350 yards, with a depth of 3 1/2 yards from grass roots to bed-rock. This was at the rate of $10.18 per troy ounce for the product, the value at that time. The value per ounce was $37.80 at the end of 1911. An average figure for the past 14 years may be taken at the value of pure gold. On this basis gravel of equal tenor for the past 14 years would have averaged $1.20 per cubic yard including overburden and pay gravel. Allowing for the gradual impoverishment of the alluvial by exploitation, and on the other side making an allowance for the increased value at the present time (50 per cent. above the average assumed above), it is safe to allow $1 per cubic yard for the entire section of the alluvials being worked to-day.

How much longer these phenomenally rich gravels will hold out, which gravels are capable of being worked by the present laborious and expensive methods, I have no way of knowing, but I should place a limit of ten years. Then the Iss may be said to have been exhausted except for such portions as may be re-worked at a profit.

There will remain in the 40 miles of its course between 75 and 100 million cubic yards of material in which there may be expected values corresponding to 20 per cent. of those previously extracted from the rich pay gravel. That is judging by experience in various parts of Siberia. How much of this gravel will be workable by dredging is only determinable by close examination. Much of the ground referred to will be of undisturbed character and the balance will represent scattered heaps and masses of tailings, difficult to handle by dredging.

The Tura River for 50 miles below the mouth of the Iss is platiniferous. The cultural conditions, however, namely the existence of peasant agricultural holdings, are a hindrance to work.

The Tura is a promising dredging field and contains an immense and practically untouched store of alluvial platinum. But existing laws, rights of the peasant owners are an effectual bar to the development of

[1] Trans. American Institute of Mining Engineers (Vol. XXIX, 1899). Platinum Deposits of the Tura River System.

dredging on this river. It is practically impossible to buy them out on any terms.

The Tagil and other small fields will probably be exhausted in the same proportion as will the platinum reserves of the Iss. Unless some undiscovered field of platinum is opened up in the North Ural, it is safe to say that the Ural alluvial platinum will be exhausted in 25 years.

The native source of the Iss platinum is well known, namely a basic rock known as dunite. The presence of platinum has been frequently determined by assay in this rock, which occurs on the Shuvalof Estate and on the Tagil Estate covering considerable areas.

So far as I know there has been no well-defined attempt to extract the platinum from this rock, and the general opinion is that it is not workable under present conditions, as the platinum does not occur in well-defined leads, but scattered irregularly through the rock in association with nodules of chromic iron.

There are two dredges working on the Shuvalof Platinum Mines, and the following information was kindly furnished by Mr. Leon Perret, the manager. The ground is extremely hard to dredge, which fact undoubtedly accounts in part for the small capacity of the two dredges. The material consists principally of stiff clay and a small admixture of gravel. The two boats were Werf-Conrad, open-connected bucket, steam driven dredges. In a season which lasted from May 11 to November 27, or in other words 200 days, the two dredges handled 246,542 cu. yd., being an average of 616 cu. yd. per dredge. The recovery in 83 per cent. platinum dust was 37,010 zolotniks or 5070.37 oz. troy, worth at Ekaterinburg Rbls. 370,100, or at $37.80 per ounce $191,660, or $.777 per cubic yard. The complete cost of handling was $.168 per cubic yard. In the early part of the season, from May 11 to June 14, the annual frost in the ground was thawed by steam points at a cost of $.054. This is the lowest cost for steam thawing either in Siberia or elsewhere of which I have any record. This is the best information obtainable from the Ural, and represents the work of a man working under many difficulties.

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POTASSIUM SALTS.

In 1911 a great deal of effort was made by the Government, and is still being made, to determine any possible commercial sources of potash in the United States. Hope is held out by the United States Geological Survey that any one of a number of possible sources will become available. The users of fertilizer containing potash are at present dependent upon foreign sources, namely the potash minerals mined in other countries. The chief source is kainite mined both in Germany and Austria where it occurs in very large deposits. In Austria these are being developed but the supply in Germany has been the principal one for the world. Another source of potash is the Indian saltpeter, containing also nitrogen, a valuable ingredient in fertilizers.

The German potash deposits were discovered by accident, the Germans being, in fact, drilling to develop a deposit of common salt and it is stated that the potassium salts as brought to the surface, were wasted for some time until their value was discovered. In the Stassfurt-Egeln district, it was found that the ordinary rock salt deposits may be above or below deposits of potash salts. The activity in other countries to discover sources of potash minerals, is largely due to the controversy between consumers and the German syndicate. The American manufacturers of fertilizer had acquired independent sources of supply in 1909 and the output of the several mines was contracted for. In the recent law passed by the German Reichstag in 1910, practically all of the potash mines were required to belong to the syndicate and to share in the deliveries. After negotiations extending over nearly two years between the Kali Syndikat and the American potash interests, the American buyers agreed, on May 20, 1911, to pay the prices fixed by the Syndikat. In addition they will pay a large sum to abrogate the contracts besides heavy legal costs and apparently receive in return only a permission granted to some of the mines they had tied up to mine a small surplus over their fixed quota without paying the extra tax. It is stated, however, that the expense to American consumers was well worth the privileges gained and not a loss as commonly supposed. The agreement affected the packers and nearly 50 firms or Independents. The contract is pronounced satisfactory and is binding until 1917. Among the independent mines were the Sollstedt which furnished the International Agricultural Corporation with potash for its own use and which was allowed to mine the equivalent

of 20,000 tons of pure potash in excess of its allotted amount in 1911. The Aschersleben potash works also had agreed to a cancellation of its five years contract with the same company. The Virginia Carolina Chemical Company entered into an agreement with the syndicate in the spring. About the middle of 1911, the Armour Fertilizer Works also entered into an agreement. The mines of the Schmidtmann group are thus deprived of their independent consumers and will, no doubt, make arrangements with the syndicate. The situation, therefore, is practically the same as that existing before the various attempts to secure independent supplies were made. The prices for concentrated

PRODUCTION OF POTASSIUM SALTS IN GERMANY. (a)
(In metric tons and dollars; 1 mark = $0.238.)

Year.	Kainit.		Potassium Salts other than Kainit.		Potassium Chloride.		Potassium Sulphate.		Potassium Magnesium Sulphate.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
		$		$		$		$		$
1898	1,103,643	3,835,856	1,105,212	3,576,628	191,347	6,380,220	18,853	763,397	13,982	259,485
1899	1,108,159	3,838,250	1,384,972	4,202,000	207,506	6,801,250	26,103	1,027,500	9,765	195,000
1900	1,178,527	4,134,000	1,874,346	5,643,750	271,512	8,793,750	33,853	1,249,250	15,368	280,500
1901	1,500,748	4,327,250	2,036,326	5,443,250	282,750	8,782,250	27,304	1,460,000	15,612	286,500
1902	1,322,633	4,571,980	1,962,384	4,949,448	267,512	7,507,710	28,279	1,079,092	18,147	334,390
1903	1,557,243	5,208,154	2,073,720	4,993,478	280,248	8,125,320	36,674	1,389,444	23,631	441,252
1904	1,905,893	6,322,470	2,179,471	5,305,972	279,238	8,425,676	43,959	1,564,572	29,285	545,972
1905	2,387,643	7,976,808	2,655,845	6,396,250	373,177	10,580,528	47,994	1,804,040	34,222	614,754
1906	2,720,594	8,918,574	2,821,073	6,538,336	403,387	11,034,632	54,490	2,032,520	35,211	644,028
1907	2,624,412	8,579,206	3,124,955	7,314,930	473,138	12,639,704	60,292	2,216,494	33,368	631,652
1908	2,715,487	9,196,082	3,383,535	7,720,006	511,258	13,369,174	55,756	2,037,518	33,149	663,068
1909	3,181,349	10,646,792	3,860,685	8,787,198	624,994	16,245,642	68,539	2,574,684	38,722	697,816
1910	4,249,667	12,712,056	4,062,004	9,034,004	741,259	17,371,144	84,583	2,989,756	37,439	668,066
1911	4,425,497	13,757,452	5,181,379	11,794,328	838,420	19,851,342	107,631	3,967,460	42,253	780,974

(a) From *Vierteljahrshefte zur Statistik des Deutschen Reichs.*

IMPORTS OF POTASSIUM SALTS. (a)
(Tons of 2000 lb.)

	1907		1908		1909		1910		1911	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Bicarbonate	155	$ 6,787	109	$ 11,500	171	$ 16,915	167	$ 16,633	163	16,428
Carbonate (crude)	3,732	266,502	3,903	269,318	4,814	324,540	4,204	263,643	5,340	327,400
Carbonate (refined)	9,326	583,730	8,428	575,731	5,505	379,392	4,518	303,917	5,572	395,728
Chloride	115,664	3,863,311	118,475	3,885,419	148,669	4,758,907	195,927	5,640,541	214,283	6,449,575
Chlorate	10	1,486	7	1,052	10	1,655	207	26,608	12	2,096
Chromate	5	685	59	8,447	86	12,330	439	43,037	11	1,699
Cyanide	1,535	483,789	1,644	494,915	1,376	386,354	1,211	328,146	1,163	326,973
Hydrate (crude)	3,732	266,502	2,971	241,995	3,521	294,409	4,392	358,855	3,713	309,965
Hydrate (refined)	20	4,545	23	4,967	64	9,289	72	11,095	41	9,254
Kainit	344,005	2,347,695	329,467	2,008,555	344,526	1,974,165	484,155	2,313,150	656,842	2,637,106
Nitrate (crude)	.....	.....	8,374	517,334	8,735	512,473	12,573	675,765	4,645	282,835
Nitrate (refined)	.....	.....	163	17,487	208	14,421	1,306	115,457	426	38,284
Prussiate (red)	29	11,811	26	10,697	39	13,522	33	12,460	27	10,230
Prussiate (yellow)	727	148,580	410	88,637	877	178,744	1,165	229,907	889	180,584
Sulphate	29,153	1,013,045	28,276	1,030,470	30,508	1,148,607	42,485	1,399,915	53,135	1,952,371

(a) For the fiscal years ending June 30.

salts correspond to the German law. For 20 per cent., 16.4 per cent. and 12.4 per cent. crude salts, however, the prices are higher. After January 1, 1914, the syndicate can raise the price 3 per cent. and no

EXPORTS OF SALTPETER FROM INDIA. (a)
(In tons of 2000 lb.)

Year.	Quantity.	Value.	Value per 100 lb.	Year.	Quantity.	Value.	Value per 100 lb.
1901.....	17,721	$1,189,400	$3.22	1906....	19,446	$1,352,735	$3.48
1902.....	21,882	1,359,335	3.11	1907....	20,025	1,337,000	3.34
1903.....	23,105	1,450,980	3.14	1908....	21,627	1,424,000	3.29
1904.....	21,894	1,331,745	3.04	1909....	22,677	1,442,633	3.18
1905.....	17,535	1,178,615	3.36	1910....	17,246	1,123,361	3.26

(a) From "Mineral Production of India," by H. H. Hayden, B. A., F. G. S., Director G. S. of India.

more, and on January 1, 1916, another 3 per cent. There are 94 firms in the syndicate varying in their interest from 1 1/2 per cent. to 1/3 of 1 per cent[1]. The syndicate will endeavor as one of its first objects to prevent overproduction. In the course of 1911, the number of independent shafts has grown from 67 to 77 and a large number of properties are in the course of development. It is only through an increase in the production and sales therefore that the quota of the existing members of the syndicate will remain the same. The value of the sales have increased 16,000,000 marks in the first 9 months of 1911 of which increase 6,000,000 marks falls to new works. It is estimated that the total sales in 1911 were 170,000,000 marks as compared with 148,000,000 marks in 1910. New money invested in new potash concerns was estimated at 97,000,000 marks in 1911, nearly three times the amount for 1910. In round numbers the total investment in this industry would approach 776,000,000 marks. The question of over capitalization does not seem to worry the new enterprises. The value for 1911 is estimated at 8,860,000 quintals. The statement made by Dr. Ochsenius in the Kunstdunger Industrie[2] showed that, allowing an operating loss of say one-third of the mineral, every square meter of potash territory estimated will still yield 56 tons of raw salt of which 8 tons is potassium chloride. A square kilometer yields 56,000,000 tons. It is estimated that the tract between the region of the River Weser and the cities of Berlin, Segeberg and Halberstadt contains 62,160 square kilometers, and supplies 5,000,000 tons per annum; therefore, it would appear that this area will last 600,000 years. The Government had fixed the amount of the various potash salts to be produced from May to December 31, 1910, in cwts. of K_2O as 7,921,000. Of this about 4,000,000 was to be supplied to foreign consumers. For

[1] *Bergw. Mittheilungen*, 1912, p. 261.
[2] Vol. XI, No. 9, through the *American Fertilizer*, Feb. 11, 1911.

the calendar year 1911, the amount of potash to be sold in Germany has been fixed as 901,940 metric tons of which 460,060 is for export. The main portion of this is for crude potash salts containing from 10 to 15 per cent. K_2O.

In upper Alsace, throughout an area of 200 square kilometers, there are two strata of potash salts known to exist at a depth of 2100 feet near Wittelsheim. The mining will be difficult on account of high temperature but the thicknesses, 3 and 16 ft., would appear to pay and make a formidable addition to the potash industry if the mining can be developed. There are said to be no faults as in the Stassfurt region.[1]

In Prussia the total production of 1910 was 6,205,242 metric tons exceeding that of the previous year by 845,933 metric tons, an increase which is due to a larger recovery of kainite in which a production of 3,119,400 metric tons shows a gain of 28 per cent. Other potassium salts had a production of 2,584,565 metric tons, an increase of something like 7 per cent. in a year. The recovery of potassium salts is now restricted to the Halle and Clausthal districts.

There were exported in 1911 from Germany 11,549,730 metric quintals of raw potash such as kainite, kieserite, etc., valued at $7,601,720. Of this quantity 6,425,734 quintals went to the United States. Potassium sulphate was exported to the extent of 1,095,548 quintals, valued at $4,462,500; of this quantity 569,695 quintals went to the United States. The amount of potassium-magnesium sulphate exported was 2,825,742 quintals valued at $4,764,760; of this quantity 1,437,746 quintals went to the United States. Potassium chloride exported amounted to 3,297,342 quintals, valued at $11,735,780; of which 2,292,398 quintals went to the United States. The production of manufactured products in 1909 is given in the *Zeitschrift für Praktische Geologie* 1911.

It would appear to be established that no immediate foreign competition threatens the German industry. If foreign countries supply the demand without discovering minable beds, that is, by new manufacturing processes, it will necessarily mean a large increase in prices.

Phonolith.—Phonolith is a finely ground volcanic stone containing 9 to 10 1/2 per cent. potash, 8 per cent. soda, 0.05 per cent. phosphoric acid, and 50 per cent. silicic acid, of which about 25 per cent. is soluble in hydrochloric acid. One great advantage of this new fertilizer[2] is that it does not contain any chloride, and is specially suitable for potatoes, tobacco, grapes, fruit trees, etc. Another advantage is that the potash in the phonolith is not immediately soluble in water, and therefore any loss of potash by washing away is out of the question. It must be used for top-soil fertilizing after the crops are planted.

[1] *Chemiker Zeitg.*, Aug. 22, 1911.
[2] *Chem. Trade Journ.*, Nov. 25, 1911.

There is an unlimited supply of this material in the Eifel Mountains, Germany; it is sold for $35.70 per car. It is not known to what extent this material will compete with potash salts as a fertilizer, but a strong movement to organize the extensive sale is noticeable. The potash syndicate states that the merits in proportion to the prices are exaggerated.

Austria.—The Austrian Potash Syndicate controls the Galician potash fields in Kalusz. This syndicate is composed of the Austrian Government and capitalists. Its operations have not extended the imports of potassium salts during the past year materially. The Austrian Government is expected to fix the prices of the production for potash material mined in the country. The development of these mines is likely to extend the facilities for selling German potash in this field but the value of these deposits, thus far, does not appear established. Estimates made in 1910 state that a supply of 33,000,000 lb. per year could be obtained from the four shafts when completed for 18 years, but it is believed that this hypothesis is based upon the geological features which are similar to those at Stassfurt, Germany. From 1904 to 1908, inclusive, the Galacien Company produced only 24,840,000 lb.

Chile.—It is considered that the saltpeter deposits in Chile may be exhausted at some date more or less near, and the capital invested could be advantageously used in new auxiliaries, such as the potassium salt deposits located in the province of Tarapaca and on Huasco Lake in the province of Atacama. In 1905 several small companies were formed to exploit these deposits. It is a well-known fact that the caliche of Chile also contains a certain percentage of potassium and that it can be utilized as a by-product of the Chilean saltpeter. It is also doubtful whether the potassium of Huasco Lake could be profitably prepared and exported.

Italy.—Deposits of alunite have been worked for some centuries near Tolfa.

Spain.—In an article by Nordal Preuss[1] the composition of a deposit found near the Benahabux railway, 10 km. from the port of Almeria, is given as follows: So_3, 77 per cent.; Al_2O_3, 37.98 per cent.; K_2O, 9.64 per cent.; H_2O, 17.61 per cent. This is an aluminum potassium sulphate called calafatite. Alumina, potassium sulphate and alum are produced from this ore by calcination but most of the sulphuric anhydride is eliminated leaving the alumina in the residue and potassium sulphate in solution. The average contents of the calcined ore is alumina, 65 to 70 per cent., and potassium sulphate, 25 to 30 per cent. The veins in which this mineral occurs are 5 to 6 meters thick, and the contents are such that

[1] *Engineering and Mining Journal*, Feb. 4, 1911.

it should compete with the German products. The mines are the property of the Compania Minera de Benahabux.

From experiments made by P. Uguet[1] the treatment should be on ore reduced to 1 1/2 in., and the temperature for calcining should make the color light brown and should be maintained 10 to 12 hours. It will be necessary to use long flame coal such as that from Newcastle. Bauxite will be a fair by-product.

Russia.—A peculiar source of potash as a by-product is that in the Caucasus where potash is obtained as a by-product to the sun-flower seed oil industry. This oil was marketed and sent to Denmark. The industry now has spread northward and consists in burning the sun-flower stalks for ash, from which potash is extracted, a process carried on in a number of factories. A syndicate is said to have been formed controlling this industry at Moscow.[2]

United States.—The uses of potassium salts are chiefly for fertilizer manufacture. While it does not appear to be as essential as phosphate, it is nearly so and potassium is far more abundant in nature than is phosphorus. According to F. W. Clarke[3] it constitutes 3 per cent. of the surface of the earth.

The possible sources outside of the discovery of mineral to be mined are discussed in the following paragraphs and may be enumerated as follows: from alunite, feldspar and igneous rocks, brines, marls, seaweed, wood ashes and other organic materials. According to the United States Geological Survey there is a good chance of discovering mineral that can be mined in the United States.

A preliminary report of the fertilizer resources of the United States is given in Doc. 190, United States Senate, Sixty-second Congress, second session. The identification of potash in the fields is by no means easy but a number of tests are described in this bulletin. Secretary Wilson of the Department of Agriculture has established at Reno, Nevada, a laboratory for the examination of material thought to contain potash. The laboratory has been established under a coöperative agreement between the Mackay School of Mines, the Bureau of Soils of the United States Department of Agriculture and the United States Geological Survey. It is in charge of George J. Young of the Mackay School of Mines and A. R. Merz of the Bureau of Soils as chemist. The laboratory will be at the service of the public free of charge.

The insoluble compounds of potash such as alunite, leucite, and various feldspars have not yet been treated commercially by any process in the United States. The soluble potash compounds include potassium

[1] *Engineering and Mining Journal*, June 24, 1911.
[2] W. Jacob. *Chemical Trade Journal*, Jan. 20, 1912.
[3] Data of Geo. Chemistry, *Bull* U. S. G. S. No. 491, 1911.

nitrate, or niter, potassium sulphate, potassium chloride and a number of double salts, namely, kainite which is a potassium chloride and magnesium sulphate and manure salts, a variable mixture of chlorides, and sulphate of potassium, sodium, magnesium, etc. containing between 15 and 40 per cent. potash. All such materials are now obtained from Germany except potassium nitrate which comes from India. The prices along the Atlantic seaboard are approximately as follows per short ton: potassium nitrate, $65; sulphate, $45; chloride, about $35; kainite, $8, and manure salts, $13. Material that does not contain too large a proportion of other salts than those of potassium, will be worth between 50 cents and $1 per unit per ton.

Alunite.—The treatment is similar to that of calafatite and in European practice the roasting is done in furnaces like lime kilns being heated in large pieces by the flame without direct contact with the fuel until sulphur dioxide begins to escape. This requires about 6 hours when the material is extracted and exposed for several weeks being moistened occasionally. The muddy product is agitated in boilers with water at 90° C. and the clear decanted liquid 10 to 12 per cent. Bé is evaporated to 32° Bé. Any ferric oxide is separated in the recrystallization.

Another process of treating this alunite is by treatment with sulphuric acid and addition of potassium sulphate. The method given by Guyot is to ignite at 800° C. for 3 hours into a clear oven, with 10 sulphuric acid to dilute to 30° Bé, and heated to 90° C. Eight tons of calcined material are then added with stirring and left for 2 hours when it is evaporated to 39° Bé and treated with 2.7 tons of potassium sulphate. After 13 hours the liquid is decanted. After further concentration, crystals of alum are removed for recrystallization. The total yield of alum is about two and one-tenth times the original weight of the ore. The insoluble matter contains 3 per cent. alumina, 2 per cent. potassium sulphate in addition to silica, etc.

The Australian treatment at Bullah-Delah consists in grinding the mineral and calcining it in a reverberatory furnace treating it in lead-lined tanks with a weak solution of sulphuric acid, heated to the boiling point by steam jets. The liquor drawn off is crystallized in cooling tanks, the sulphate of alumina remaining in solution. The residue is re-treated with water. The liquor containing sulphate of alumina is then returned to the vat and sufficient of the calcined material added to neutralize the free acid. By boiling, reversion takes place and ferric hydrate is precipitated. The alum after cooling is again crystallized for market. The sulphate of alumina solution finally yields the lower grade of sulphate of alumina by running the liquor on to lead tables where it solidifies.

The alunite deposits near Marysvale, Utah, are now being worked with a very favorable prospect. They are described in a report by B. S. Butler and H. S. Gale, *Bul.* 511 of the United States Geological Survey.

Another source of potash is the rock containing the mineral leucite especially where it occurs in Leucite Hill, Wyo., in the Rock Springs gold district. It is a lava containing about 53 per cent. silica, 12 per cent. alumina, 11 per cent. potash and 6 per cent. magnesia and lime. It is estimated there are, in the various deposits on Leucite Hill, 1,973,000,000 tons of the rock which will therefore contain 197,000,000 tons potash. Many chemists are working on the problem of extracting the potash. The only grant of land in the territory covered by these deposits were the original odd sections given the Union Pacific Railroad as its original subsidy. The railroad has agreed to waive the royalties after September 1, 1912, but there is other Government land open to location.

Among the possible places of occurrence in the United States the Geological Survey points out a number of enclosed basins in the Western deserts.[1]

Government work has been in progress near Fallon, Nev., since the summer of 1911 and drill holes are sunk in the ancient lake bottom. The prospecting of Railroad Valley, 75 miles southwest of Ely, is expected to be undertaken by the Government. The presence of potash became known there when the sodium salts gathered for the chloridizing process was found to contain too much potash to be used for this purpose. The salt in Railroad valley runs from 1 to 8 per cent. potash and the surface water samples give results equivalent to 20.5 KCl. It is hoped to determine whether this lake bottom has underlying deposits similar to those in Germany as it is thought that the geology is similar. The Railroad Valley Potash Company of Tonapah is building roads and preparing to prospect the country with drill holes.

This prospecting[2] is being done on the basis of a careful geological study. Lake Lahontan flooded the valleys of northwestern Nevada in recent geological times but is now evaporated. Its area was 8422 square miles and depth 866 ft. As it is known never to have overflowed, it must have been charged with mineral salts as this is the common rule, and yet no deposits have been found in this area. The lake sediments examined show salts presumably the same as the waters from which they came, but the main deposits must exist below these. The predominance of potash rocks around this old lake, together with other factors, low points, etc., favor the location for drilling as at present. There are other areas that are surrounded by rocks predominantly limestones and sedimentaries which would not furnish potash or soda. The well

[1] H. S. Gale, *Bul.* 530-A. U. S. Geological Survey.
[2] H. S. Gale, *Bull.* 530-A. U. S. G. S.

at the end of 1911 had reached a depth of 400 ft. and will be continued to 1000 ft.

The geological work in other regions led to negative conclusions. The Bureau of Soils also determined that surface deposits of commercial value are not likely to occur near Otero Basin, New Mexico.

Possible Deposits of the Gulf of California.—Dr. H. Erdmann[1] in reviewing the possible sources of potash salts, dismisses those of France, Austria and the Caspian region also those of Thibet, because the potassium chloride in all of these deposits is too small for profitable extraction. Across the Gulf of California, the Colorado marshes were dammed in such a way that in the depression resulting from the drying up of this gulf end, he estimated 200,000,000 tons of potassium chloride deposited. It cannot be inferred, however, that this mass is in a form ready to be mined. He points out the unfavorable conditions as to wages and transportation and then considers the flooding of this area by the American engineers as making prospecting impossible for a number of years to come.

The greensand marls of New Jersey are rich in glauconite, formed by taking up potash from waters. The composition of the glauconite or greensand is a calcium carbonate, sulphate, phosphate, with quartz sand, iron phosphate and shells, and the marl ranges from 1 to 3.8 per cent. phosphoric acid and from 3.5 to 6.3 per cent. potash. Thirty years ago the material was used locally as a fertilizer and proved very valuable. Its use is now on a very small scale, and it cannot compete with concentrated fertilizers.

In Kentucky the Leitchfield marls contain, according to a sample analyzed by C. Palmer, 3.73 per cent. K_2O and .13 per cent. phosphoric acid. In McNairy County, Tennessee, greensand mentioned in 1835 by G. Trost analyzed as follows: Silica 48 per cent., alumina 7 per cent., ferrous oxide 20.70 per cent., K_2O 10.1 per cent., calcium carbonate 5.70 per cent., and water 8.0 per cent. But phosphoric acid was not determined.

As a source of potash, a number of methods have been suggested for treating these greensands. One consideration in their favor would appear to be the ease of mining them as compared with that of feldspars.

The patent by E. C. Eckel (No. 1,011,172, Dec. 12, 1911) outlines a process by which the greensand is heated, with or without purifying, in a suitable kiln to a temperature of 900° C. The potash is volatilized and condensed in a chamber as potassium hydrate. The addition of substances like pyrite, gypsum, lime or limestone is also suggested, as they assist in the chemical action and reduce the temperature necessary.

[1] Abstract from Kuxen-Zeitung, Berlin, in *Engineering and Mining Journal*, May 27, 1911.

R. Peter suggested[1] to set free alkalies by the addition of calcium chloride and carbonate.

Potash from Seaweeds.—In the opinion of the Government officials, one of the most promising sources of potash at present is from the large areas of kelp groves along the Pacific coast. It is estimated that the kelp can be gathered from the 100 square miles already surveyed, and that 1,000,000 tons of chloride of potash can be obtained annually. The value of this source of potash remains to be determined. Besides the potash, laboratory tests have shown that iodine will be a by-product. A plant has been established in Southern California, about 30 miles north of San Diego, for the extraction of potash, etc., from seaweed, on a large scale. The plant is owned by the Corinado Chemical Company which has been working on the kelp proposition for some years. It consists of four furnaces capable of treating 6000 lb. of dried kelp per 24 hours. Additional furnaces are under construction and it is proposed to have a total capacity of 36,000 lb. of dried kelp per day. The kelps were at first interesting on account of being a source of iodine. It has been found that the kelp of the Puget Sound area, when dry, contains as much as 30 per cent. potassium chloride, and .02 per cent. iodine. In the South they contain 20 to 30 per cent. potassium chloride and rather more iodine than in the North. Should these kelp beds of Puget Sound prove commerically profitable, they should be protected and not cut until after the middle of July as they are annuals. The kelp of the South are perennials and will grow indefinitely. It is possible that, like all other sources of potash that require a manufacturing process, a number of by-products will be needed to sustain the industry. The importations of iodine and iodine products are worth $1,000,000 annually.[2] The seaweeds of the Atlantic apparently do not contain iodine.[3]

A number of analyses both of the ash and seaweeds from various localities are given. The data show that in a ton of the dried stems of Pacific weed there is approximately 1/2 ton of potassium chloride of 89 per cent. purity. The present cost of this article is about $42 per ton. A ton of the same contains 4 lb. of iodine the value of which is about $2 per pound. This gives total value of $29 per ton. A second Pacific weed shows that a ton of the dried material contains about 600 lb. of potassium chloride of 80 per cent. purity and is worth $40 per ton. Hence the whole weed, in this case, is worth—potassium chloride $12 and 2 lb. of iodine, or $16 per ton. The distillation products, in all cases, were worthless. The Atlantic weed examined in these tests is

[1] Kentucky Geol. Survey, 1884.
[2] F. K. Cameron, *Chemical Engineer*, March, 1912.
[3] G. V. Green and H. S. Johnson, *Chemical Engineer*, Feb. 1912

not considered a commercial proposition and it is not yet possible to state that the Pacific weeds will be a commercial success.

H. S. Firman of San Diego, California, has taken out a United States patent No. 1,010,403, November 28, 1911, for treating seaweed. Seaweed is made to undergo a slow combustion with the admission of air to burn the carbonaceous matter. Water is added to dissolve potassium salts while the temperature is still high enough to cause the residue to clinker.

Brines and bitterns are being carefully examined by the Survey and during 1911 many of those east of the Rocky Mountains were examined. From analyses made in the Bureau of Soils, Department of Agriculture, by R. F. Gardner and A. R. Merz, many of these solutions were found to contain more potash than expected. The results from a natural bittern from Fairport Harbor, Ohio, occurring over a large area is given as better than any other such material and contained 3.9 grms. per liter potassium and 43.4 grm. per liter sodium; the percentage of KCl in the solids therefore is 2.5. Other constituents of the bittern are Ca 48.6 grm., Mg 11.2, Cl 188.4, SO_3.2, and ferric chloride 3 per cent.

Waters of some lakes as Owens and Mono in California, contain potassium and indicate that beds resulting from evaporation of similar lakes in the desert regions of California may include much potash.

That potash is included in the waters of Borax Lake or Searles Lake has long been known and was recently investigated by the Government. The area containing wells sampled is owned by the Foreign Mines Development Company which it, or its predecessors, were working for borax, now no longer profitable alone.

A number of chemists have been working on the problem of extracting potassium salts from feldspars. The process patented by Firman-Thompson, United States Patent No. 995,105, June 13, 1911, is as follows: The feldspar is ground to pass through a 100-mesh sieve and five parts of the powdered rock mixed with 5 parts of an alkali by-sulphate and 1 8/10 parts of an alkali chloride, are heated to a bright red, cooled and mixed with water. The potassium and sodium sulphate are separated from the solutions. The sodium by-sulphate reacts with chloride to produce hydrochloric acid which at high temperatures acts on the feldspar. It is claimed that 80 to 90 per cent. of the potash contents of the rock can be recovered. It is probable that the gas liberated in the process could be recovered and utilized. A company has been organized and the Spar Chemical Company also proposes to operate a plant at Baltimore.

The process patented by Edward Hart of Easton, Pennsylvania,

United States patent No. 997,671, July 11, 1911, provides for the recovery of potassium salts by fusing the feldspar with potassium sulphate or barium sulphate, together with carbon. This can be done in the electric furnace. It is then ground and treated with sulphuric acid, a residue of pure silica and a solution of sulphate being obtained. From this, alum and potassium sulphate are crystallized out. This product is then fused with carbon yielding a carbonate through a solution of which waste furnace gas containing carbonic acid is passed to precipitate alumina. Some potassium sulphate remains in solution.

The patent granted to Franklin R. Carpenter is dated May 31, 1910, and consists essentially in treating feldspar, heated with or without a flux and then suddenly chilling, reducing the material to a fine powder.

The number of processes invented for the purpose of extracting potash from silicates depend for their success largely upon the by-products. It is said that on a small scale, a few processes are being developed that practically use all the material handled in the shape of commercially salable by-products.

Ascham[1] finds acids useless for decomposing feldspar. Where calcium chloride is cheap, fusion with the anhydrous salt or with lime and common salt is possible.[2] Ascham also finds that the potassium becomes soluble by treatment with hot lime and water under pressure.

The work of extracting potash from feldspar has been reviewed by Cushman and Hubbard.[3] Several years ago, N. A. Martin found that above 100° C. under pressure, hydrous sulphuric acid acted as a catalytic agent and caused in a dilute solution of sulphuric acid a great deal of decomposition of feldspar. At the Sheffield chemical laboratory, Yale, recently also S. R. Scholes, carried on experiments with electric current and hydrosulphuric acid and got some good results.

In the manufacture of cement, the potash is said to be objectionable and the dust contains an appreciable amount due to volatilization in the kilns. It is stated by Cottrell that, at one plant in California where this dust is precipitated by high tension currents, it is proposed to leach out the potash before returning the dust to the feed.

Potassium carbonate was for a long time the best-known potash salt, being the residue from boiling lye from wood ashes in iron pots, whence the name potash.

The following table shows the "potash" or K_2O in the several potash salts and minerals:

[1] *Z. Anorg. Chem.* 74, 55, 1912.
[2] Rhodin, *Journ. Soc. of Chem. Ind.*, 1901, 431.
[3] *Jl. Am. Chem. Soc.*, 30, 779, 1908.

Element.	Symbol.	Percentage of potassium (K).	Chemical equivalent in terms of "potash" (K_2O).
Potassium	K	100	.....
Potassium salts or "potash" salts—			
Potassium chloride or muriate (mineral sylvite).	KCl	52	63
Potassium sulphate	K_2SO_4	45	54
Potassium nitrate (saltpeter)	KNO_3	39	47
Potassium hydrate or caustic potash	KOH	70	84
Potassium carbonate	K_2CO_3	57	68
Stassfurt minerals—			
Carnallite	$KMgCl_3 \cdot 6H_2O$	14	17
Kainite	$MgSO_4 \cdot KCl \cdot 3H_2O$	16	19
Sylvite (potassium chloride)	KCl	52	63

PRECIOUS STONES

By George F. Kunz

It is remarkable that in the face of a universal financial depression the prices of diamonds and pearls have advanced considerably since 1906, and this after only one year's shutdown of the great group of diamond mines and with no attempt to regulate the price of pearls. The imports of precious stones have remained constant, but were less than in 1906, a banner year and the result of overbuying. The effect of the small importation in 1908 was the selling down of the old stocks, which in turn led to the greater imports of 1909, 1910 and 1911.

However, the upward trend of prices, is clearly apparent when we compare the average prices of the uncut, rough diamonds for 5-year periods for the last 20 years: 1891 to 1895, average price per carat 26s. 9.45d.; 1896 to 1900, 29s. 1.15d. per carat, increase in price, 8.6 per cent.; 1901 to 1905, 47s. 3.36d. per carat, increase, 62.5 per cent.; 1906 to 1910, 55s. 7.93d. per carat, increase, 17.7 per cent. The diamond syndicate advances on the price of rough diamonds when sold to the diamond cutters have been as follows: June, 1906, 4 per cent.; May, 1907, 3; June, 1909, 5; June, 1910, 2 per cent.

The value of the precious stones, cut and uncut, brought into the United States in 1911, is about $42,163,164 as against $42,315,830 for 1910. From fiscal year figures, 1912 will exceed this.

Standard of Weight.—The general adoption of a uniform standard for dealings in precious stones, based upon the metric carat of 200 mg. is a matter much to be desired. The carat has heretofore varied somewhat in different countries, resulting in some confusion and inconvenience in business. The metric carat has now been formally adopted in most of the countries of Europe, and its use made compulsory. Our own country and Great Britain, however, have thus far taken no action in this matter. Many of our dealers and importers recognize the theoretical advantage and inconvenience of the change, but do not take any active interest in it, apparently, as a practical reform. This matter is giving Germany and France a great lever in international trade. But the change must come ere long. A single standard for all countries, and the substitution of decimal for common fractions in the carat, are advantages so plain that they must surely soon be realized.

A striking illustration of the defectiveness of the system hitherto in

use, as compared with the standard metric unit, is given in a recent article by Mr. L. S. Spencer, of the British Museum, on "The Larger Diamonds of South Africa."[1] Mr. Spencer tries with care to clear up some published errors and misstatements as to the great diamonds obtained in recent years from the African mines. But it proved impossible to obtain the precise weights of some of these notable stones, especially those of the earlier discoveries, on account of the uncertainty as to what carat-weight was employed. The present English "Board of Trade" carat is 205.304 mg., while the "old" English carat, prior to 1888, was 205.409 mg., and the French "international" carat is 205 mg. Hence with some of the stones, only an approximate statement of their original weight is now possible.

The writer first suggested[2] dividing the carat into 100 parts, the carat to be 200 mg. each; that is, five diamond carats to the gram, or 20 pearl grains, or 5000 carats to the gram, and 20,000 gr. in a milligram.

The subject of various diamond carats, their incongruity and the confusion of values of historic gems is fully treated by the writer.[3]

DIAMOND

According to recent figures[4] the 1911 production and values of the South African Union are: De Beers, 1,924,225 carats, valued $24,690,430; Premier, 1,774,206 carats, $7,169,850; Jagersfontein, 338,831 carats, $4,968,895; Koffyfontein, 123,933 carats, $1,473,165; other mines and the Voorspoed will bring the Union product to 4,891,998 carats valued $43,743,620.

In the De Beers mines the year may be considered fairly successful, although the profit on all operations amounted to £2,270,291, against £2,690,726 in the preceding year; but the decrease, as noted by the chairman of the De Beers Company is due to the fact that in the year ending June 30, 1910, many sales were made out of the accumulated stock, thus unnaturally increasing the apparent output. In the actual output there was a gain in 1911, and the prices increased 3 per cent. or 4 per cent. over those secured in 1910. Dividends were paid to the amount of £800,000 on the preference shares and £1,000,000 on the deferred shares —equivalent to 40 per cent on each class of stock. The Reserve Fund, which amounted to £968,905 in 1900, stood at £1,337,953 on June 30, 1911.

The cost of production increased slightly in 1911 in the De Beers

[1] *Mineralogical Magazine*, London, 1912.
[2] Before International Congress of Weights and Measures, World's Columbian Exposition, 1893.
[3] George Frederick Kunz and Charles H. Stevenson, "The Book of the Pearl," New York, 1907, pp. 321–329.

and Kimberley mines, but decreased considerably in all the others of the group—the figures per load being as follows:

	1910.	1911.
De Beers—Kimberley	8s. 5.29d.	8s. 7.67d.
Wesselton	5s. 5.82d.	4s. 9.09d.
Bultfontein	4s. 4.48d.	3s. 11.45d.
Du Toitspan	5s. 5.34d.	4s. 7.09d.

The margin of profit per load was greatest (10s. 10.235d.) in this latter mine, as in the preceding year.

The yield of diamonds per load has continued to decrease, except in the Bultfontein mine, where it showed a slight advance. The data on these points are as given below. It should be noted, however, that the DeBeers has not actually been worked since 1908, the mine being in course of re-equipment, and all the diamonds accredited to that mine came from the deposit-floors.

THE OUTPUT OF THE DE BEERS CONSOLIDATED MINES, LTD.

Mine.	Loads Hauled. (a)	Loads Washed.	Yield per Load. Carat.	Value per Carat.	Value per Load.
1911.					
De Beers Kimberley	445,169	1,402,894	.28	46s 7.79d	19s 7.11 d[1]
Wesselton	2,422,487	1,423,117	.27	37s 9.6 d	10s 2.47 d
Bultfontein	2,457,412	1,866,212	.38	35s 0.52d	13s 3.79 d
Du Toitspan	2,780,070	2,335,240	.21	73s 6.5 d	15s 5.325d
	8,105,138	7,027,463			
1910.					
Be Beers. Kimberley	431,339	1,808,734	.38	47s 9.25d	18s 1.83 d
Wesselton	1,702,237	2,139,738	.32	34s 3.58d	10s 11.7 d
Bultfontein	2,099,173	1,818,509	.37	33s 1.62d	11s 10.68d
Du Toitspan	878,755	917,175	.23	68s 1.46d	15s 8.02 d
	5,111,524	6,684,156			

[1] Figures are different from those in printed statement.
(a) One load = 1600 lb.

The total number of carats of diamonds secured in the 2 years from these mines as computed from the average per load washed is:

	1910.	1911.
De Beers. Kimberley	687,319	392,810
Wesselton	684,716	384,241
Bultfontein	672,848	709,160
Du Toitspan	210,950	409,400
	2,255,833	1,895,611

The actual result of the year's production was £4,938,086, and the revenue from all sources, £5,928,830. Expenses of all kinds reduced this to a net profit of £2,998,616. Out of this were paid the dividends above mentioned, and Government taxes for the previous year, £310,137, and in part for the current year, £265,458, leaving a final balance of £623,019.

In one important respect the figures for 1911 differ from those for 1910; while in 1910 there were washed 1,572,632 loads more than were hoisted; in 1911, the excess of loads hoisted over those washed was 1,077,675. This has increased the stock of blue ground to 8,416,372 loads. Including 604,654 loads of lumps, there were in all 9,021,026 loads, entered on the balance sheet, at a cost price of £1,414,209. During the year ending June 30, 1911, as many as 2,359,021 loads of tailings were washed, yielding 256,631 1/4 carats of diamonds. This added to the production of these mines as given above, makes a total of 2,152,242 carats of diamonds produced in the year.

There was but little change in the depths of the shafts in the De Beers group of mines during the financial year 1911, except in No. 10 Prospect shaft of the Kimberley mine, which was deepened 400 ft. The main rock tunnel on the 3520-ft. level of the Kimberley mine was holed April 15, 1911, its total length being 1,154 ft.

DEPTH OF DE BEERS MINES.

De Beers	Main rock shaft	2,466 ft.
	No. 1 shaft	1,670 ft.
Kimberley	Main rock shaft	3,601 ft.
	No. 10 Prospect shaft	3,520 ft.
Wesselton	Main rock shaft	1,119 ft.
Bultfontein	No. 1 Main rock shaft	780 ft.
	No. 2 Main rock shaft	1,084 ft.
Du Toitspan	No. 1 Main rock shaft	860 ft.
	No. 2 Main rock shaft	768 ft.

The De Beers Company seems again coming into the lead of the diamond-supply, as the richness and quality of the Premier mine appear to be declining, and the output of German South Africa has not continued to increase. The company has also won an important success in a recent suit against the Rhodesian Chartered Co., the Privy Council having decided that the former has a right of claim to all diamonds found in Rhodesia.

The *total* output of the De Beers group of mines up to 1911 may be estimated at the enormous figure of 67,000,000 carats, worth over £100,000,000, or half a billion of dollars. If to this be added the yield of the Premier and other less extensive mines, and that of German Southwest Africa, the total may be placed at $600,000,000, which will

be doubled again by the costs of cutting, etc. Yet notwithstanding this vast production, the prices continue in a steady advance.

The 89,676,316 loads of material taken out of the De Beers' group of mines alone, from 1889, the date of the consolidation of these mines, to June 30, 1910, would constitute a mass 500 ft. square and 5736 ft. high; and yet if the diamonds derived from this blue-ground could be packed into a case 5 ft. square and 4 ft. high, the case would weigh 26,967 troy pounds, and be worth $405,000,000.

If we add the product of the great Premier mine, from 1901 to October 31, 1910, 36,839,926 loads, the volume will be 500×500×8096 ft.—over a mile and a half. The case of diamonds would gain a little less than a foot in height to accommodate the 10,581,041 carats of diamonds from the Premier mine; it would then hold diamonds weighing 59,611-581 1/4 carats, weighing 32,773 troy pounds and worth $450,000,000.

How much more precious diamonds are than gold is shown by the fact that 32,773 lb. of that metal would only be worth $8,100,000, the ratio being about 1 to 55 1/2. Of course, this refers only to rough diamonds, many of them very small, for a brilliant-cut diamond weighing but one carat is worth nearly 1000 times its weight in gold, and the disparity increases at an exceedingly rapid rate in the case of larger diamonds.

Vaal River.—Bloemhof, the Vaal River, is some 200 miles southwest of Johannesburg. The field here known as Klipfontein, has an extent of 12 square miles, and was officially opened up as an alluvial digging in 1907. More recently deposits of greater premium were discovered at Mooifontein, 17 miles from Bloemhof, and here an area of about 700 acres was opened for diggings September 13, 1911, and another area of 900 acres was soon to be proclaimed here also. The diamond gravel in this region is overlain and concealed by from 6 in. to 1 ft. of soil. It lies on a bed of andesitic lava (amygdoloidal diabase) whose irregular surface affords depressions in which diamantiferous gravel has gathered. During the 6 months from April to September, 1911, diamonds weighing 10,575 carats were found at Bloemhof valued at $268,000. In general, the diamonds extracted 20 carats in weight, and one of 80 carats is said to have been found at Vaalbosch Rush; this was estimated to be worth $5,000. The smaller stones have sold for as much as $22 or $23 per carat (88s. to 92s.). The chief center of activity, however, now, is Mooifontein where there are said to be gathered as many as 10,000 diggers, and about the same number of Kaffirs are employed. While the total output is not inconsiderable averaging about $160,000 monthly, the declared output of both Boemhof and Mooifontein for November was only $146,916. It will readily be seen that this total

gives but very meagre returns for each individual digger, although of course some are more fortunate than others.[1]

The diggers along the Vaal have been at the mercy of traveling buyers who pay what they please. A meeting was held recently at Austin's Rush, to plan for a better system. It was proposed to get the Transvaal Government to aid the diggers by a loan, to be repaid from the sale of diamonds, all under various precautions. Then a delegation or committee would come to America, where rough diamonds can be imported free of duty, and arrange for the Vaal River product to be sent here for sale and cutting.

The old Monastery Mine,[2] on which the New Star Diamond Mines, Ltd., hold on option, is being worked anew. It is stated to be the only "diamond-pipe" which appears in hard sandstone, and this rock seems to have favored the formation of large masses of the various minerals formed within it. An exceptionally large and fine zircon discovered here was at first believed to be a diamond, and Mr. Barney Barnato is said to have offered £1000 for the crystal. As a rule, the diamonds found in this mine are covered with an aluminous coating, not removable by acids, and rendering the stones unusually dull in appearance. Many of the diamonds are distinctly inferior, while others are very fine; one gem weighing 5 1/4 carats was valued at $2625. This mine appears to have been worked in ancient times, as human skeletons have been found in it at a depth of 40 ft. These are now in the Bloemfontein Museum.

German Southwest Africa.—Although large quantities of diamonds have been obtained here, it now seems as though the production would fall off. The predictions of Dr. Rohrbach and other investigators cited in our last report, give the impression of enormous resources for many years to come; but the practical results do not yet warrant any fear of overproduction.

The yield for 1911 was 798,866 carats which sold for $5,325,000,[3] and which was larger than in the year before; but the advance has not continued.

The returns of the German diamond companies show decreased dividends. The Kolmanskop Society, after paying 55 per cent. in its first year, reduced the dividend to 45 per cent. in the second year, while the dividend for the past year will be 30 per cent. Still more unsatisfactory is the showing of the German Diamond Company, which has cut its dividend in half, from 10 per cent. in the last year, and that of the Luderitz-

[1] Report of Consul Edwin N. Gunsaulus of Johannesburg, Dec. 22, 1912; *Daily Circular and Trade Reports*, Feb. 3, 1912, pp. 535-538.
[2] *Diamond Fields Advertiser*, July 24, 1911.
[3] *N. Y. World.*

bucht United Diamond Mines, which paid none. Increased cost of production and the tax of 33 1/3 per cent. on gross value levied by the Government upon exported diamonds, are factors in this poor showing. Indeed, many of the smaller companies have suspended operations until the desired change is made from a tax on gross value to a tax on proceeds.

The diamond deposits are now known to extend along the coast from about 24° to 25° south latitude, from Conception Bay nearly down to the mouth of the Orange River, but probably not very many miles inland. The country is a nearly level stretch of barren granitic rock, thinly covered with sand, which is swept to and fro by the wind, and piled up in shifting dunes. In these sands are the diamonds, small but brilliant octahedral stones, rarely over half a carat in weight, but cutting into fine gems. Stones vary in size and quality according to several sources. But those sources, which must have been extremely rich, are as yet a mystery. Some have thought that they have been brought by rivers formerly flowing from the interior highlands; others believe that their source is in diamond-bearing pipes or fissures now under the sea, whence they have been washed and blown with the sand, or swept in by currents in a period of former depression. This last view has been strongly advocated by Mr. M. E. Francis, of Johannesburg, an authority on diamond geology. The "blue-ground" region reported as found, has been carefully investigated, and large garnets as well as the so-called "carbonado," or ilmenite, have been obtained there, but no diamonds. In these investigations, as well as in other matters pertaining to these diamond fields, the German Colonial Secretary, Dr. Solfs, has been indefatigable.

A recent lecture on the geology of this region by Prof. Dr. Scheibe, before the Royal Agriculture High School in Berlin, describes the coast plateau as composed of ancient crystalline rocks, covered with the present loose sands and gravels, remnants from sedimentaries now eroded.

This account, however, does not solve the problem of the source of the diamonds. It may be noted that with them are found strewn through the sand, pebbles of chalcedony and agate of various colors, which did not come from the underlying granites and schists, and point to some other source, apparently in igneous rock.

The diamantiferous sands vary greatly in their richness. Near Luderitz Bay, the average may be taken as from three to five small diamonds per cubic meter. But in places where wind-concentration has occurred, the lighter material being blown away, the average may be eight or ten times as many; and in the Pomona district, ten or more diamonds of larger size may be seen on a square meter of surface, imply-

ng perhaps several hundred to the cubic meter. Again, there are other arge areas where the yield is very scanty.

In many places the initial stages of washing on sieves are still entirely performed by hand, but these primitive methods have been largely superseded by the use of machinery.

By the introduction of the Schiechel sifting machines 90 per cent. of the diamonds are now secured, while only 60 per cent. were obtained under the older method of hand-working; it is also found that the introduction of these improved mechanical processes prevents theft. Since the German Government has taken entire control smuggling is almost impossible.

The output of the German diamond fields goes mainly to Antwerp or cutting, 90 per cent. or more being the proportion, and the small emainder goes to Berlin for cutting. An Antwerp syndicate takes harge of the main output, under an arrangement with the German ompanies somewhat like that of the London syndicate with the De Beers mines. The German Colonial Secretary, with the director of the German Diamond Company, has been visiting Hamburg, with the object of the possible opening of diamond-cutting industries there. The contract with the Belgian syndicate expires ere long; and it is probable that a German syndicate will be organized to take up the product of the mines and have it cut in German cities.

Australia.—The deposits at Copeton, New South Wales, are alluvial, he fields lying on the eastern watershed of the Gwyder River, ome 18 miles south of Inverell. The diamantiferous gravel for the nost part rests upon the granitic rock, and above it are overlying strata nd basalt. This flow covers a wide extent of country, and erosion has eft it in scattered hills and ridges. Below it are beds of river drift and ravel, sometimes many feet in thickness, the material being much olled and waterworn. All are regarded as of tertiary age, the basalt-low being subsequent to the river-drift. The conditions in this respect ecall those in California, where diamonds occur in the old river-beds, verlain by a Tertiary lava-flow. The diamond-bearing gravel consists nainly of quartz, with much tourmaline, and some garnet, topaz, sapphire, a little gold, and an important quantity of tin ore. This latter elps to make mining operations profitable even if the diamonds would ot suffice for this. As an instance when both diamonds and tin were ound in considerable quantity may be cited the "Streak of Luck" mine, here a shaft was sunk 140 ft. into a depression in the granite bottom, n rich diamantiferous gravel, from a few inches to 9 ft. in depth, and ielding on the average for each load from 8 to 10 carats of diamonds nd about 15 lb. of stream tin. These diamonds are said to have been

sold for 25s. per carat. This "pocket" was unusually rich. At the Ryder Brothers' mine, 5 miles south of Copeton, four men are stated to have found 2300 carats of diamonds in one year's work, the stones selling for 22s. and 23s. per carat. While the tin-bearing granites are known to exist in the vicinity, the source of the diamonds is another question and one of especial importance. A few small diamonds are said to have been found in a dolerite rock in a pipe or dyke form at the "Oakey Creek" mine, at the southern end of the diamantiferous belt. But as to this there is some question; and at best, a much richer matrix rock must be found to yield the quantity of diamonds found in these widespread gravels. Most of the diamonds from this region are small, from 1/8 carat to 3 carats; but they are of fine cutting quality, as indicated by the prices. The largest stone from the Copeton district is stated to have been taken from the "Star" mine; it weighed 7 carats. Besides the mines already mentioned, the "Round Mount," "Deep Shaft," and "Old Farm" mines, are among the most productive. All yield tin ore; and some, as the "Lone Star," "Soldier's Hill," and "Molacca" mines, are operated mainly for tin. Others are the "Banker," and "Kirk's Hill" mines, the latter adjacent to "Oakey Creek," where the occurrence of diamonds in basalt is reported. At "Kirk's Hill" mine, four loads of the wash furnished 1010 carats of diamonds and 2 cwt. of tin ore.

Frequently a land of ferruginous conglomerate is found, at various levels in the drift, evidently formed by deposit from iron solutions, and often forming a hard cement layer, from an inch to a foot in thickness. It may be quite productive, and strongly recalls the "cascallho" of Brazil. This material is thrown into a fire, when it crumbles, and is then easily crushed and washed, and the diamonds recovered. Frequently the deposits are reached by drifting into or under the basalt-capped hills.

The diamonds[1] of this region are very hard, requiring a long time to polish. They are mostly white, also yellow, brown, and occasionally of the rarer colored shades. Mr. Oscar G. Roberts, of Sydney, in a communication to the writer, states that for years, the diamond-seekers were unaware of the value of bort, and discarded it when met with. But of late, a demand for it has arisen for the diamond drills much used to develop the water-supply of Australia. The specimens found are true bort, not carbons; they resemble the white (not the yellow) diamonds occurring with them, in their hardness; the round pieces are set, together with Brazilian carbons. Good specimens on the field, are worth from 20s. to 80s. per carat, depending on size and quality. They are also sold for local use in china-drilling, etc.

[1] On the "Copeton Diamond Fields (N. S. W.)," *Australian Mining Standard*, April 20, 1911. p. 397, 8

A well-known mining expert, Mr. David Grove, is investigating the character of the diamantiferous area at Inverell, New South Wales, where he has acquired a half-interest in the pipe mining. He states that Mr. Levy, a well-known Amsterdam diamond-cutter, has pronounced the Inverell stones to be equal in brilliancy when cut to the best Brazilian, and also asserts that the statement made in South Africa, that Australian diamonds cannot be properly cut, is not founded in fact. Mr. Levy offers to purchase the output of the Inverell mines. Mr. Grove says he will make a comprehensive test to determine the value of the mine. If it proves a true pipe it will be found to contain large stones. If satisfied, cutters will come out to cut and polish the stones, thus establishing a new Australian industry.[1]

British Guiana.—The production of diamonds fell off from 1910, but showed an improvement in quality. The output is stated to have been 26,467 stones, weighing 3035 carats; while that of the year before was 85,537 stones, weighing 7180 carats. The number of diamonds produced was therefore less than one-third, but the number of carats fell only to three-sevenths. The stones of 1911 averaged nearly one-seventh of a carat, while the average of the year previous was only about one-twelfth of a carat. The obstacle to the development of diamond-production is the lack of communication and transportation. The only access is by boats and canoes, requiring a journey of 2 or 3 weeks.

Brazil.—It has long been known that diamonds occur in Goyaz, but there has been no important mining there. The newly discovered field reported recently in Brazilian papers lies in and along a little-known stream called the Garcas, a small affluent of the Aragura River, which flows northward to join the Amazon. The region is wild, and occupied only by a few Indians. A "rush" of diamond-seekers has already set in, and the neighboring states are stimulated to organize surveys and explorations in their stream-beds, in the hope of similar discoveries.

Canada.—An interesting discovery of diamond-occurrence, from a scientific point of view, though not from a commercial one, is reported from British Columbia, by Mr. Charles Camsell.[2]

Early in 1911, some specimens of chromite from the Tulameen District, B. C., obtained by Mr. Camsell, were analyzed by Mr. R. A. A. Johnston, and found to contain minute diamonds. The crystals break up on being freed from their matrix, but the fragments are clear and brilliant. The chromite occurs distributed in little veins and masses in peridotite rock. This rock appears as a stock, intruding sediments

[1] *The Mining Journal*, Vol. XCVII, No. 3999, p. 347; London, April 13, 1912.
[2] Private communication to the writer.

and flows of Triassic age, and has been much altered to serpentine, resembling thus the diamond-bearing peridotite of Arkansas, and perhaps the African kimberlite.

Analysis developed two parts or varieties in the chromite, one magnetic and the other non-magnetic. Diamond was present in the former to an extent of 9 per cent. in the latter only 3 per cent. Some gold and platinum were also found.

The diamonds were obtained from the chromite by fusion with sodium carbonate; and they have also been identified in their sections. They present two varieties, one a yellowish aggregate of very minute particles, the other well-formed octahedral crystals of great purity. It appears that the carbon was an original constituent in the peridotite magma, and crystallized out in solidifying, apparently before the chromite.

China.—Small diamonds are long known to have been picked up in the rice-fields after rains; but there is no definite knowledge as to their number or quality, as they are valued and retained among the natives. It is stated[1] that "black diamonds" of two or three carats are found in "two provinces in North China," through which provinces, is not stated. They are fairly brilliant, and harder than ordinary diamonds, and are prized by the natives as ornaments. This is believed to be the first announcement of the occurrence of black diamonds in China.

Arkansas.—In the year 1911 little was done at the diamond-field, owing to the lack of capital for active work. Two "pipes" are now determined, and probably a third is located. The Arkansas Diamond Co., which owns most of one of the recognized pipes, is endeavoring to raise the required capital. The washings have shown a yield of 28 carats of diamonds per hundred loads (of 16 cu. ft.); this, in the form adopted in the African reports, would be 0.28 carat per load, about the same as that of the Kimberley mine. But the diamonds are mostly small. One was obtained, however, from another part of the same pipe, known as the Mauney tract, that weighed 8 1/8 carats—a fine stone, the largest yet found in Arkansas; one or two others are also reported, of 2 or 3 carats' weight.

Illinois.—The occasional occurrence of diamonds in the glacial drift, along the line of the terminal moraine, from Wisconsin to Ohio, and again reported in Illinois, has led to the belief that there must be some source or sources for them in the Canadian wilderness Ungava, somewhere near James' Bay. A good deal of prospecting has been done at various points, but thus far with no definite result. Statements have been made in some newspapers, of diamonds being found in Ungava,

[1] Mr. Edward de Nilla.

and taken to New York and shown to the writer; but the reports have either lacked foundation, or the stones, though real, have had some uncertainty as to their actual source. An expedition has been organized and has lately started from Ottowa, with Indian guides, and ample supplies, to go up toward Hudson Bay, and explore the region during the present summer. The Illinois discovery above referred to, was in Jefferson Co., near Ashley, where several diamonds are said to have been found. The largest was a stone of 7 carats, and was sold to a firm in Alton, Ill., for $175.

Diamond-cutting Industry

The Industry has been on the whole quite a prosperous one in 1911. As before, the chief centers have been Amsterdam and Antwerp, of which the latter has increased. This is partly due to the fact that a Belgian syndicate had a contract for the output of the diamond field in German Southwest Africa. So far as the exports to America are concerned, many Amsterdam diamond dealers come weekly to Antwerp to sell their product for shipment from that port.

The fear felt of an overproduction has not been realized. The De Beers mine output is carefully regulated, so as not to exceed an annual amount of about 2,500,000 carats.

The regularly organized diamond workers in Amsterdam number about 10,000, representing some 70 establishments, beside a considerable body of unorganized workmen. In 1912, it is expected that 1000 apprentices will be added—most of them sons of the present operatives. Pupils pay large prices for instruction, even as much as $2000, for cleavers, payable in instalments; and the school for turners and polishers charged from $120 to $150. As soon as a pupil can do good work he begins to receive pay.

Small private cutting establishments are developing—with a consequent tendency to a lowering of prices.

Antwerp.—The reports received from the American consul-general at Antwerp, Mr. H. W. Diederich, give some interesting accounts of changes in the conditions of diamond-cutting in Belgium. The business has long been dominated by a syndicate, which, in particular, refused to permit apprentices. The coming-in of the large output from German Southwest Africa compelled an important increase in the number of cutters; but the syndicate would not modify its regulations. This led to a revolt in 1909. The movement has gone on, and spreading over Belgium, as Mr. Diederich says, independent diamond-cutting in small establishments is fast becoming a national industry. The independent workers are free to initiate their sons into the business.

From 1891 to 1911, the number of cutting establishments has advanced from 5 to 300, that of employees from 300 to 16,000, and that of diamond-brokers from 2 to 200. The wages paid to cutters in 1911, amounted to over $7,000,000, and the total volume of business to $48,000,000. This great increase is largely due to the imports of German-African diamonds, of which more than nine-tenths are taken up by a Belgian syndicate. How far this condition may be permanent, however, is not certain.

In exports of diamonds to America, Antwerp is ahead of Amsterdam, the value for 1911 having been $11,357,475 or about 13 per cent. more. This was $500,000 less than in 1910. Of this total, nearly two-thirds ($1,348,151) represented uncut stones, while in 1910 the uncut stones were less than half. This indicates the growth of cutting in the United States. There has been much talk as to a possible combination between the London syndicate, controlling the De Beers and Premier mines, and the German syndicate. But this does not seem very practicable. The character of the products is different. The Cape mines furnish all the larger-sized diamonds and control the market for them; but for the smaller stones (melées, as they are called in the trade) there is little demand in comparison. The German-African stones are all melées, and there are no larger diamonds to compete with the Cape output. At the same time, the German stones have gained much favor because of their brilliancy and their somewhat lower hardness, causing them to cut and polish more easily. The two syndicates are, therefore, not in competition.

The products of the several African mines, exhibited from time to time in London, are taken up almost as fast as they are displayed. A large share of the growing demand comes from the United States, but also a good deal from England itself, from France, and from some countries of Asia. Inferior grades of diamonds are rising in value with the prices of the finer stones.

It is of interest to note the statement that American-cut diamonds are in demand in all countries, being regarded as possessing the finest cutting and finishing.

At the present time it is estimated that the number of persons employed in this industry in various countries is about as follows: Belgium (as above) 16,000; Amsterdam, 8000–10,000; Germany, 1000; Switzerland, 400; London, 100; Paris, 100; New York, 300; Total, 25,900–27,900.

Although the diamond-cutting industry has made immense strides in the past few years, the year 1911 did not mark a continuance of the upward movement, there being, indeed, a very slight retrogression as appears in the following table:[1]

[1] From Consul General Henry W. Diedrich's report: *Daily Consular and Trade Reports*, p. 774: Feb. 21, 1912.

Quarters.	Cut Diamonds.		Rough Diamonds.		Total.	
	1910.	1911.	1910.	1911.	1910.	1911.
First	$ 2,907,064	$ 3,331,201	$ 134,727	$ 222,296	$ 3,041,791	$ 3,553,497
Second	2,373,125	2,701,961	222,561	317,713	2,595,686	3,019,674
Third	2,645,738	2,491,503	169,428	540,734	2,815,166	3,032,237
Fourth	2,135,916	1,484,659	268,479	267,408	2,404,395	1,752,067
	$10,061,843	$10,009,324	$795,195	$1,348,151	$10,857,038	$11,357,475

It will be noted, however, that the falling-off affected only the last quarter, the returns for the first three quarters showing an increase of $598,738 in the value of cut diamonds exported from Antwerp to New York; this was a little more than offset by a decrease of 651,257 in the last quarter. For both cut and rough, exports increased 518,437 over 1910. The exports from Amsterdam to New York in 1911 were but slightly less than those from Antwerp, being 10,080,851.

Agate.—The formation of agate chalcedony is exemplified in a specimen belonging to Herr Renck of Offenbach of enhydros still in the matrix. This enhydros has lost in volume through contraction (loss of water) and has thus become a so-called "rattling-stone." The enhydros is partially surrounded by a white, mealy mass, which should be amorphous silica. It is still fully embedded in a black, compact, fresh melaphyr; only one side being freed. A very narrow gap separates the enhydros from melaphyr. Traces of silica represent the beginning of a refilling of the gap.[1]

The writer believes it probable that in these formations the silica of the chalcedony shell was originally combined with the enclosed water and became freed therefrom. The separation of the silica from the main mass of water must have at first taken place on the surface, that is to say, a membrane must have been formed which soon became impervious to water. The enhydros crust did not arise by slow diffusion, but by a rather rapid precipitation.

Even in agate formation the outer crust formed is usually so firm that the mass retains its external form in spite of the loss of water. That is to say, the gap between melaphyr and agate is very slight. In agate very considerable internal contractions are to be noted. Indeed, in the enhydros also the filling of the cavity is scarcely ever complete, for a later cavity, the result of consolidation, allows the contents to be movable.

After an examination of thousands of agates and enhydros, the writer, however, believes that the enhydros were formed by moving heated waters that entered and passed from the gas pores in the igneous

[1] Raphale Ed. Liesegang and J. Rerick "Enhydros-Bildung" in *Centralbl. für Min. Geol. u. Palaontol.* 1912, No. 7, pp. 193–198.

rock that they formed in, and depositing layer after layer on the sides and at the minute entrance of the cavity eventually filled the opening and sealed the contents of water for endless time.

Emerald.—The occurrence of true emeralds in the northwestern part of South Carolina, has been known for some years;[1] but no actual locality has been known. There are translucent deep-green emeralds, cut and set, in the hands of individuals in South Carolina; but no mining has been done. A company has been formed to develop a property of 165 acres near Earle station, just south of the North Carolina line, which is believed to contain emerald deposits.

The Turner emerald mine in Cleveland Co., N. C., from which a number of fine gems have been obtained within the past 3 years, has been described by Mr. Douglas B. Sterrett.[2] He remarks (p. 26) that rock associations very similar in character to those containing emerald at the Turner mine, occur at a number of points in the region south of Shelby, N. C. This is quite near the border, it is probable that an emerald-bearing zone will be found to extend through this section of the two Carolinas.

The special conditions under which emeralds are formed, as brought out in this report, appear to be as follows. The region is one of Archean rocks—the Carolina gneiss, with igneous and granitic intrusions, not positively determined as to age, but the granite latest. These veins and lenses of pegmatite contain beryl and in some cases the true emerald. These are found only when the pegmatite traverses certain basic rocks—such as gabbro—which are found on analysis to contain a constant amount of oxide of chromium—0.16 to 0.19 per cent. This oxide, the coloring-matter of the emeralds, was evidently taken up into solution from these rocks by the pegmatite. If it contained the beryllium oxide necessary to form beryls, emerald then resulted. But only a fraction of the pegmatites are beryl-bearing; and hence the combination of two very special conditions was requisite to develop emerald.

Jade.—The natives of New Zealand have long used and greatly prized the green nephrite, to which they gave the name of *pounamu.* They obtained it from boulders along the west shore of South Island and in streams flowing down to that coast from the unexplored interior. Within recent years there has been careful search by prospectors and official geologists, for the source whence these boulders came; and some excellent localities have been found in the region of Milford Sound and up the Arahura and Greenstone Rivers. A considerable amount of true "greenstone" or pounamu has been taken out, and some of it exported

[1] 16th Ann. Rep. U. S. G. S., *Min. Res. of the U. S.*, Pt. IV, p. 600.
[2] *Min. Res. U. S.*, 1910, pp. 23–25, and 1911, pp. 19–26 (preliminary).

to be cut in Germany. The stone has hitherto been prized by the Maoris, but of late it is coming into favor among the more advanced races.

The Maoris had special names for different varieties of pounamu. One known by them as "tangiwai," the queen of greenstones, is an entirely distinct mineral. It was found to be bowenite, a variety of serpentine, very pale greenish in color, to almost white, and much more translucent than the nephrite which is a variety of amphibole. A very interesting discovery has been made within the past year, of what is probably the long-lost source of tangiwai. The new discovery corresponds closely with the place indicated by the native traditions. Samples of the stone taken to Auckland have been recognized by Maoris as the genuine *tangiwai*. Prof. Marshall, F. G. S., of Otago University, has examined the material and pronounced it to be bowenite "of unequaled quality" for all ornamental purposes.

The practical importance of this discovery lies in the fact that a ledge will yield not only pieces for all kinds of small objects, but masses of larger size. Characteristic cloudy waves, which add to its beauty, and are not easily imitated. Glass imitations of jade are found in the market in Europe.

Bowenite occurs in the United States, at Smithfield, Rhode Island, and is familiar to mineralogists, and it is known also in India, in the Punjab, where it is worked into small ornaments, as in New Zealand; but heretofore it has been obtained only in very limited amounts, and not very translucent. This newly-discovered locality promises an ample supply of this beautiful stone for export to America and Europe, and especially perhaps to China and Japan, where jade of all kinds has long been a material peculiarly esteemed.

Opal.—The Japanese opal locality, discovered in 1905, is situated at Takarasaka, in the prefecture of Fukishima. It is now being worked by the Tokio Hosaka Kaisha, Ltd. The matrix rock is reported to be a perlite; several varieties of the mineral are found—milk-opal, opal-agate, precious opal, glass-opal (hyalite?) and smoky or granitic opals. The precious opal is stated to resemble the Hungarian mineral; but a marked variety is one that is entirely transparent and colorless, but presents orange or emerald-green colors when seen at certain angles of light. The stone here has a specific gravity of 2.22 (typical) and a water-content of 8.49, with no extraneous matter.

The great extent of volcanic rocks throughout the far northwest furnishes many localities where this beautiful gem-stone may be found. In the author's reports to the Interior Department from 1890 to 1905, some sixteen distinct occurrences were noted.

There have at various times come to me specimens of opal from near Lovelock, in the southern part, and wood opal from the northeastern part of Humbolt Co., Nevada; also, opal in concretionary masses from Austin, Nevada. In Idaho, opal comes from Caldwell, and from Rockville, and Squaw creek, Owhyee County, and from Clover creek, Lincoln County in the Snake River region in the southwestern part of the state; and from near the Salmon River, from Baker and Durkee in Oregon, and from Walla Walla, Douglas County, and Whelan, Washington, (near Mexico, Idaho).

For the past 25 years small specimens of opal as float have been found near the junction of southwest Idaho, southeast Oregon, and northern Nevada. Opals from Drewsey, Oregon, have been described by the writer and also specimens from northern Washington. In 1889, there was found in John Day River a specimen of good fire-opal 1 in. long, 1/2 in. wide and 1/4 in. thick, a drift pebble, either out of some gravels or a river bed. The color was excellent and quite equal to the pale yellow fire-opal from Queretaro, Mexico.

About 3 years ago some specimens of fire-opal were found in northern Nevada, at Virgen valley, west of the Santa Rosa Mountains. This resembled certain types of the Mexican opal from Queretaro, though not precious opal such as was found in Washington State. Of these Nevada opals some are absolutely transparent and pellucid, either with large flames of color, or with a smaller harlequin flaking. These change gradually into pale yellow, yellow-brown and brown, and sometimes are only sub-translucent but with a great play of color, finally becoming almost black. These dark varieties include some strongly resembling the hue of certain crude petroleums, or that of the darker types of Burman or Roumanian amber.

More recently another locality has been found 8 miles from the above. This has furnished a number of specimens remarkable for their large flames of red, strongly resembling lumachelle marble. A number of these are lusterless and many of them are more or less cracked, partly due to the fact that they have a large water content, and partly because they have been mostly found very near the surface. In one respect, this occurrence resembles the deposits at White Cliffs, New South Wales, in that, as there, some of the opals are pseudomorphous after wood or other objects,. There was a newer find in 1911, a limb of a tree measuring 50 cm., or nearly 2 ft. in length and 8 cm. across, entirely replaced by opal; the outer parts were very brilliant, while the center was of the dull, common opal variety.

In nearly every instance these masses are found in decomposed volcanic rock, or in ash that has hardened. Apparently there must have

been a later flow of siliceous waters to change them to this form. The deposit is west of the Santa Rosa Mountains and near the Trout Forest Range and the Pine Forest Range. Some of the stones cut several years ago still hold their color, but it is possible that a number of them may not be of the more durable type. This is the most interesting occurrence of opal that has yet been noted in the United States. The deposit found in 1909 was traced to a depth of 16 ft., whereas the opalized tree and later deposits above mentioned were found at a depth of only 2 ft.

These opals were described by the writer before the N. Y. Academy of Sciences, on Nov. 6, 1911, and specimens exhibited. More material has since been obtained, of the same kind but even finer. Nowhere have more elegant opals been found, or better examples of the opaline replacement of wood. No trace of vegetable structure remains visible. The outer portion often breaks off with a beautiful pisolitic or small mammillary surface. The variety of colors and types of opal is very remarkable here; the dark browns, like fluorescent petroleum; dark with broad flames of color; or shot through with brilliant little lines and streaks of fire in a field of pale purple or blue; another type is a hazy or misty white with brilliant play of colored fire; and again with the transparency of water.

Gems of over 20 carats have been obtained here, and a great many of like dimensions. The question of durability is an important one with opals. A good proportion of these appear to hold their color well and remain unchanged; though some show a tendency to fracture after a time. These are of the fire-opal variety, an opal with a high water content.

Rhodonite.—A fine quality has been found at a locality in Josephine Co., Oregon, some 30 miles west of Grants' Pass. Specimens are of rich cherry red, and of darker shade, and local collectors have obtained handsome pieces. It is said all to come from a large vein.

A remarkable crystal of rhodonite (Fowlerite) from Franklin, N. J., has been acquired by the Brush mineralogical cabinet at New Haven, Conn. It is described by W. E. Ford and R. B. Crawford.[1] The crystal is unusually rich in faces, and one new face was determined, v, $(0\bar{4}1)$, and another, k $(2\bar{2}1)$, which had been reported before, but doubtfully.

Sapphire.—These have been in favor of late in jewelry, and the production from the Yogo Gulch, Montana, mines on Judith River, has gone on quite successfully. Montana stones are often not so dark or turbid as some of the oriental stones; and the desirable stones retain their blue color by artificial light. The season in 1911 was more favorable than in

[1] *Amer. Journal Science* (4), XXXII, p. 289 (Nov., 1911).

1910; water was abundant without damaging cloudbursts. The output therefore showed considerable increase, amounting, for the two companies engaged there, to some 500,000 carats. Of these, about 18 per cent. were suitable for cutting as gems, the rest being available, as usual, only for watch jewels. Some exceptionally fine sapphires were obtained, in particular two stones of rich cornflower blue, weighing 2 1/2 and 3 1/2 carats respectively, and bringing record prices.

Turquoise.—An important discovery has lately been made, of definitely crystallized turquoise, occurring at Lynch station, in Campbell Co., Va. A full account, with analysis and crystallographic details, is given by Mr. Waldemar T. Schaller.[1]

The mineral occurs in a matrix of glassy quartz, broken up into irregular fragments, united by the turquoise, which fills the interstices, and in some cases shows a botryoidal surface with minute drusy crystals. The turquoise is of a fine blue color, and the mass polishes well into a handsome ornamental "matrix" stone.

The crystals are clear and brilliant; few exceed 1/3 in. in diameter; and being closely aggregated, measurement is difficult. Some were determined, however, as triclinic, practically imomorphous with chalcosiderite.

Analysis gave the composition of true turquoise, corresponding to the formula CuO, $3Al_2O_3$, $2P_2O_5$, $9H_2O$.

This, Mr. Schaller feels satisfied to be the correct expression for the composition of this species. The following are the actual analyses, with those given by Prof. Penfield for a turquoise from Nevada.[2] The density is approximately 2.8.

	Virginia.	Nevada.
P_2O_5	34.13	34.18
Al_2O_3	36.50	35.03
Fe_2O_3	.21	1.44
CuO	9.00	8.57
H_2O	20.12	19.38
Insoluble	.....	.93
	99.96	99.53

The face of the mineral now being found, for the time, definitely crystallized, disposes of the question regarding the character of turquoise as a true mineral species. Mr. Schaller alludes to the other theory of its being a mixture of a copper and an aluminum phosphate, saying that these specimens give no support to it. The close agreement of the analysis, moreover, with that given by Prof. Penfield, which was itself

[1] *Amer. Jour. Sci.*, Jan., 1912.
[2] *Amer. J. Sci.*, Nov., 1900.

coördinate with a number of others from various localities, confirms and doubtless establishes this view.

Mr. Sidney Paige, in *Economic Geology*, for June, 1912 (Vol. VII, No. 4, p. 382), discusses the occurrence of turquoise in the Burro Mts. of New Mexico. The turquoise is confined to a very moderate depth, within the zone of alteration; while the apatite appears lower down. The phosphoric acid has no doubt come from the latter, the copper from the sulphide ores, and the alumina from the sericitic alteration of feldspars in the granite rocks, all as a result of erosion and weathering.

Zircons.—Specimens of Zircon have been brought from Campbell Island, some 300 miles south of New Zealand, about in W. long. 70° and between 52° and 53° S. lat. The specimens are small rounded nodules, some of them showing obscure crystalline form, but remarkable for their brilliant polish or luster; they are clear and of a rich hyacinth-red color, and would cut into small but very handsome gems.

Amber.—An exhaustive search for deposits of amber in Russia has been carried on recently by three Belgian mining experts. The best results were secured in the Volhynia and Minsk Governments, where amber was found in considerable quantities near Dombrovitza, on the Rovna district, at Goryni and near the village of Barashi in the Zhitomir district. It is believed that on further investigation these deposits will be found worthy of exploration, especially in view of the progressive exhaustion in Western Europe.[1]

Prof. Dr. Richard Klebs, connected with the Royal Geological Survey, and scientific adviser to the Royal Amber Works,[2] gathered and owned the great collection which was exhibited under the auspices of the Imperial German Government at the St. Louis Exposition in 1904. This great collection consists of 10,000 inclusions in amber, comprising beetles, flies, spiders, wood, leaves, and many other interesting objects. It has been held at $40,000 and will not be divided.

The last paper he wrote[3] appeared in the "*Schriften der physik.-ækonom. Gesellschaft zu Königsberg*, i Pr." Jahr. LI, pp. 217–242, III, 1910, and describes these fossils with illustrations.

Pearls.—It is evident from the recent reports of the Ceylon Company of Pearl Fishers that this venture has been very unsuccessful. A total deficit of £118,517 on operations up to the present time is reported. Naturally, the company is seeking for a modification of the terms of its lease with the Government of Ceylon, and the latter is said to be inclined to accord this, should the annual inspection reveal the presence of a

[1] Résumè from the *Mining Journal*, Vol. XCVI, No. 3994, p. 228; London, March 9, 1912; citing (merely by name) the Russian Universal Technical Review.
[2] Died in Königsberg, Prussia, June 20, 1911, in his sixty-seventh year.
[3] "Ueber Bernsteineinschlüsse im Allgemeinen und die Coleopteren meiner Bernsteinsammlung."

spat-fall, and should the application of wire netting to defend the young oysters from the attacks of predatory fish be efficacious.

The periodical disappearance of the oysters from the Ceylon pearl-banks, often for a number of years at a time, has been an element of uncertainty. Mr. Hugh M. Smith, U. S. Deputy Commissioner of Fisheries, in an extended article on the Ceylon pearl-fishery,[1] stated that in the last century pearl-fishing was possible in only 36 years, the barren intervals varying from a few years to as many as 16. The curious fact appears, from recent studies, that the destructive fishes themselves have a part in the actual production of the pearls. The latter are formed in the tissues of the young shell-fish by the secretion of nacre as an envelope around the bodies of small encysted parasitic worms (custoids), which enter the shells and penetrate the tissues and thereby cause irritation to the mollusc. They do not develop further in the pearl-oyster, and those that die form the nuclei of pearls. But if the molluscs are eaten by fishes, the encysted larvæ reach maturity to repeat the same cycle of life-history. Even were it possible in any way to keep the fish from access to the pearl-banks by wire netting, as proposed, this attempt might only defeat its own object.

The Ceylon pearls are extremely numerous and many of them very small as explained by the manner of their origin—the "seed-pearls" of commerce.

A serious problem has arisen in the pearl industry of Australia, in consequence of the decision of the Government that after January 1, 1913, only white men may be employed as divers and tenders. This work has hitherto been done by Malays and Japanese. U. S. Consul J. F. Jewell, of Melbourne, states that there is much concern and doubt among the pearlers, as to how far this change can be accomplished. Many of the pearl-banks lie beyond the 3-mile territorial limit; and if the new method is not successful, it is feared that the industry may pass into foreign hands. The Pearlers' Association of Broome, Western Australia, has laid before the Premier of that State a project for training white men for the work of pearl-diving. A committee in London is also going to endeavor to get men from northern Europe to meet the Australian need.

[1] *National Geographic Magazine*, Feb., 1912.

QUARTZ

Quartz is the most abundant of minerals, being one-eighth of the total at the earth's surface, and is mostly used in the form of sand. (See under Glass Sand, page 280.) Quartz or rock crystal has been used from ancient times for ornaments, works of art, lenses, etc. Its toughness, low dispersion and good heat conductivity are properties that are utilized in scientific instruments.[1] The sources of rock crystal to-day are chiefly pebbles found in the streams in the States of Goyaz, Minas Geraes and Rio Grande de Sul, Brazil.

In massive form, quartz is found in veins and often produced as a by-product where feldspar is mined. Flint is a compact, tough silica occurring in nodules in limestone or chalk or in pebbles weathered from them. It is used largely for tube mills and also finely ground in pottery, either for the body to prevent shrinkage, or in glazes.

The methods of preparing quartz by grinding for pottery purposes, are either wet or dry. Sometimes the massive quartz is broken up by heating in kilns and adding cold water to the heated mass. The material is usually ground in pans provided with a pavement of flat quartz blocks, the muller being made of the same material. Some of these mills will grind a ton of material every 24 hours. The material is then settled and dried for shipment. In the dry method, the crushing precedes screening to obtain the various sizes. For this process, rolls and tube mills are used. The sizing is finally done in air currents. The principal uses of ground quartz, besides pottery, are for wood fillers, paints and soaps. The crushed mass in coarser sizes is also used for the manufacture of sandpaper and sand for sand-blast apparatus. Other uses are for filters and tooth powders. In the manufacture of Dina's brick, material containing about 97 1/2 per cent. silica, 1 1/2 per cent. alumina, and 1 1/2 per cent. ferric oxide is suitable. Lime is sometimes added as a binding material. The expansion of these brick in burning is said to be due to the conversion of quartz into tridymite.[2]

Reliable figures for the production of quartz in the United States are not obtainable. In 1909, there were 135,469 tons produced in the United States and in 1910, 63,577 tons, as stated by the United States Geological Survey. Of this only about 14,000 tons were ground and the average sale price is given as $8.20 per ton in 1910.

[1] A. E. H. Fulton, Oct. 6, 1911, *Journ. Roy. Soc. Arts.*
P. J. Holmquist, Tonind-Zeit, 1911, 35, 1323, 1327.

Flint pebbles are imported principally from French and English coasts and the price is $4 per long ton. In 1910, the imports were valued at $307,286 as against $301,547 in 1909 and $219,754 in 1908. Quartz to the value of $219,664 was exported from Havre to the United States in 1910. Flint pebbles in addition worth $75,000 and $42,478 were exported to the United States in 1910 and 1911 respectively. Ground flint is furnished for pottery industries in England at $10.95 per ton. Sweden produced 12,520 tons of quartz in 1910, valued at 58,359 crowns.

Quartz from the mine is now largely used in tube mills. It is an effective grinder, but does not make a good lining. In feeding a mixture of quartz and pebbles, the pebbles make up the lining, while the quartz stays in the charge, so that the advantages of both materials are gained.

In recent years sand or quartz is used in the electric furnace for the manufacture of ferro-silicon and other alloys. Coke is used in order to reduce the silica to silicon and iron ore or scrap iron to supply the metallic iron. The silicon in these alloys varies from 10 to 80 per cent. Ferro-silicon is used in the manufacture of steels as a deoxidizer. The ferro-silicon imported from Germany into the United States was 3214 tons in 1910 and 4052 tons in 1911. Some of this material, about half the quantity, is also exported from the United States to Germany, which country imported 38,317 tons in 1911 and 27,440 tons in 1910, chiefly from Norway and Switzerland. From Norway, 5121 tons were exported in 1910; 2645 tons in 1909; 2004 tons in 1908 and 3788 tons in 1907. Silicon is also produced in the electric furnaces and the product is useful in the reduction of other metals, such as chromium and tungsten. It can be cast into molds suitable for chemical ware and is used in the manufacture of rheostats and electrical heaters, its electrical resistance being five times that of carbon.

Fused Silica

A. E. Marshall.—The production[1] of fused silica apparatus in sizes suitable for manufacturing operations was made possible by the Thermal Syndicate's electric-furnace process, which was patented in 1904. Prior to that date small transparent articles had been made by various experimental workers from rock crystal.

The method generally adopted was to build up the article from small pieces of crystal in an oxyhydrogen blow-pipe flame. The size of these vessels was, of course, limited, but despite this and their necessarily high cost, they were used for purposes where the peculiar properties of the ware rendered it more serviceable than other materials.

[1] American Chemical Society, March 8, 1912.

The electric-furnace process of fusing pure silica is applicable to the manufacture of large and small apparatus, the maximum sizes being pipes of 12 in. diameter and 30 in. long, basins 20 in. diameter, and retorts of 75 liters capacity. The articles made by this process are not transparent like the rock crystal product, as the mass does not become perfectly fluid during fusion, but attains only a plastic consistency with the result that the small bubbles disseminated through it, are unable to rise to the surface.

The small ware intended for laboratory purposes is subjected to a special process which increases the transparency and gives the material a highly glazed surface. The glaze is of the same composition as the ware, that is to say, it is pure silica and is not an added alkaline glaze. The application of the glazing process is at present limited to ware not exceeding 6 in. in diameter.

Fused silica possesses properties which render it of great value in the chemical and allied industries, the principal property being its great resistance to heat and to acids. It is highly refractory, the melting-point (which is not well defined) being between 1700° C. and 1800° C. The material softens appreciably at lower temperatures (about 1500° C.), but does not readily lose its shape.

There is one point in connection with fused silica that is sometimes overlooked and that is its tendency to devitrify or to pass from the vitreous to the crystalline state, if it is continuously heated above 1200° C. This, of course, does not prevent the use of the material for intermittent work at temperatures considerably greater than 1200° C.

The heat resisting properties of the ware received an unsolicited testimonial when the British section of the Brussels Exhibition was destroyed by fire in August, 1910. The metal fittings of the stand were fused into a shapeless mass, but the various pieces of fused silica remained intact despite the enormous temperature attained.

In order to prevent alkalies from attacking it, a coating of varnish or metal is applied, the latter by a spray process as for galvanizing.

The ware is unaffected by acids, with the exception of hydrofluoric and phosphoric acids. In the latter case the action only becomes pronounced at 400° C., so that for practical purposes such as the concentration of phosphoric acid, it can be considered as being proof against this acid.

The electrical insulating value of fused silica is high, and the resistivity decreases very slowly with a rise in temperature. This suggests that the ware may be utilized for insulating purposes when the required shapes can be produced.

There are various types of electrical heaters or radiators in use in

England and in Continental Europe, consisting of a heating element of non-oxidizable wire enclosed in thin-walled tubes of fused silica. The tubes allow the glow of the spiral to be seen, and a very pleasing effect is produced.

In the chemical industry, fused silica has found extensive applications, particularly in the construction of sulphuric, nitric, and hydrochloric acid plants. Certain operations such as the concentration or distillation of sulphuric acid have always presented difficulties to the manufacturer by reason of the failure of the material from which the plant has been built. In some cases, costly vessels of platinum or platinum-iridium have been used to overcome the difficulties.

Fused silica has been adopted with great success for these two operations, some 40 basin-type concentrators being in daily use in England alone. The silica basins are invariably exposed direct to the fire gases so that low fuel consumption and high outputs are obtainable.

In the case of nitric and hydrochloric acids, fused silica pipes have been extensively applied in place of earthenware. The pipes are made in the usual shapes with either socket or tapered joints. Considerable care is necessary in making the joints, owing to ordinary jointing materials and cements having coefficients of expansion considerably greater than silica. The best form of joint is one relying on asbestos for the main portion of the joint, the outside only being finished with cement.

A large variety of special shapes of apparatus, such as coils, tanks, retorts, crucibles, etc., are made for various branches of the chemical industry where either resistance to acids or heat, or a combination of these properties, is desired.

Ware of an ornamental character, such as finger bowls, fruit plates, etc., having a highly lustrous surface can be made by the same process as the chemical ware. The luster is due to the reflection of light from the surfaces of highly distended air bubbles, and it is therefore untarnishable.

The principal firms now making this product are, the Thermal Syndicate, Ltd, England and the Heraeus Company, Germany.[1]

The Deutsche Quartz Gesellschaft, Germany, was dissolved in 1911 as the product was not equal to the English and could not compete with it. The Electro Thermische Werke, Seebach, Zurich, Switzerland (Dr. Franz Wolf Burckhardt) makes zircon siloxyd besides quartz.

H. A. Kent, of the Silica Syndicate, of London, England, patented (1,003,271, Sept. 12., 1911) a process for making fused silica, the first claim reading as follows: "The manufacture of silica glass by causing grains or particles of silica to fall or be projected separately through a heated zone and onto a movable heated body in a chamber, said particles

[1] Charles Colne, *Met. and Chem. Eng.*, April, 1911.

being subjected to a temperature sufficient to sinter them by this single operation into a homogeneous mass of silica glass." In one method of making a silica rod or tube, the electric furnace is provided with adjustable electrodes. The ground silica grains are fed from the container through a tube with a revolving worm and drop down between the electrodes onto a heated cylinder of silica-glass upon which they form a homogeneous mass of further silica glass.

Siloxyd.—According to resuslt obtained in the Institute for Metallurgie, at Aachen[1] zirconia or titanic oxide, or both, were added to silica and produced mixtures that are prominently resisting to fusion and, in fact, have many excellent properties.

Zirconia silica or Z-siloxyd has not the silky appearance of the English Vitreosil especially when made in tubes. Tensile strength of small quartz tubes was found to be 860 kg. per square centimeter, but larger tubes show less strength and probably contain strains For thermal use or to withstand heat, 0.5 per cent. zirconia in siloxyd is recommended, whereas for mechanical strength which is superior to that of fused silica, 1 per cent. zirconia gives the best results. An intermediate product would be suitable for both uses.

The devitrification can be estimated by the ringing sound which becomes dull, but a piece unaffected has a clear ring like porcelain. The zircon siloxyd is more resistant to attack by bases such as alkalies and magnesia. A tube of silica at 1500° C. in contact with magnesia fused almost immediately whereas the siloxyd took over 1/2 hour to be attacked but was not rendered useless.

The siloxyd made by the Zirconglas Gesellschaft, M.B.H. at Frankfurt a. M. is formed and begins to melt at 1700° C. but the deformation is very much slower than the fused silica made by the same company. In grinding this material on emery wheels it will stand much rougher handling than the silica and there is not so much danger of flying to pieces.

The titanium siloxyd or T-siloxyd or "titanium glass" was tested, specimens being obtained from each of the works mentioned. Those containing from 0.1 per cent. to 2 per cent. TiO_2 were found to be tougher than quartz although the compressive strength of fused silica was somewhat greater. The titanium glass also was more refractory even than the zircon siloxyd. The combination of zirconium and titanium oxides with silica was found to have intermediate properties. The titanium siloxyd appears to be quite opaque and not homogeneous when more than 1 per cent. of TiO_2 is added and it has a blue, or dark blue color, depending on the reducing atmosphere in manufacture.

[1] Siloxyd, *Chemiker Ztg.*, Jan. 9, 1912. Dr. Felix Thomas.

QUICKSILVER

In the United States conditions for selling changed in 1911. The previous calendar year was not satisfactory and the same applies to 1909. As is well known the quicksilver production of Spain, Austria and Italy, is controlled by the Rothschilds and one selling agency in the United States also acted in harmony with this firm in London. Independent sales, however, were made in 1911 in this country and after the withdrawal of a number of producers' accounts from the syndicate the market is quite free and on a commission basis. The statistics are given in the following table:

STATISTICS OF QUICKSILVER IN THE UNITED STATES.

Year.	Production.					Exports.			Imports.	
	Calif. (a)	Texas.	Others.	Total.	Value. (f)	Flasks.	Metric Tons.	Value.	Pounds.	Value.
	Flasks.	Flasks.	Flasks.	Metric Tons.						
1897	26,648	.....	.....	919	$ 993,445	13,173	475	$394,549	45,539	$20,147
1898	31,092	(b)	153	1,077	1,194,746	12,830	445	440,587	81	51
1899	29,454	261	.....	1,025	1,416,790	16,518	573	609,586	131	83
1900	26,317	1,700	233	974	1,279,436	10,702	353	425,812	2,616	1,051
1901	26,720	2,932	75	1,031	1,382,305	11,219	389	475,609	1,441	789
1902	29,552	5,252	.....	1,208	1,515,714	13,247	459	575,099	*Nil.*	.....
1903	32,094	5,029	.....	1,288	1,564,734	17,575	610	719,119	*Nil.*	.....
1904	28,876	5,336	700	(c)1,204	1,348,185	21,064	731	841,108	212	160
1905	24,655	5,000	1,050	1,045	1,217,652	13,460	458	497,470	2,690	1,710
1906	19,516	4,517	1,276	861	1,035,138	6,455	220	244,299	84	50
1907	(d)17,532	3,000	400	712	868,678	5,132	175	192,094	16,566	6,719
1908	(d)16,969	2,832	346	685	903,391	2,995	110	124,960	15,113	8,215
1909	16,217	4,188	810	704	941,233	6,803	231	266,243	15,968	8,203
1910	(d)18,536	3,382	500	763	1,054,991	1,923	65	91,077	667	381
1911	(e)17,000	3,400	1,100	732	1,466,000	291	10	13,995	471,944	251,386

(a) Reported by the California State Mining Bureau, except 1907–08–10–11. (b) Included in "Other States." (c) Estimated; the weight of the flask was changed from 76.5 lb. to 75 lb. within this year. (d) Figures collected by the MINERAL INDUSTRY. (e) Estimated. (f) Computed at average price at New York.

The decrease in stocks caused a record importation. This came to New York mostly from England, but also 1203 flasks from Mexico and 600 flasks from Italy. The exports fell off in 1911. Usually the largest consumers are Canada and Mexico, although Costa Rica is third in order since 1910.

The United States Geological Survey[1] reports more ore mined in 1911 than in 1910. The amounts for the whole country were 143,582

[1] H.D. McCaskey.

and 131,813 short tons respectively. The average recovery was 11.5 lb. of metal per ton in 1911 and 12.5 lb. in 1910. Nevada and Texas together mined 12,246 tons of ore of which 7599 tons were treated yielding 2396 flasks in 1911. In 1910 the mines of Nevada alone produced 320 tons of ore, of which 35 yielded 70 flasks in 1910.

Prices.—For American quicksilver San Francisco is the primary market. The relation between New York and London prices is close; in 1911 New York quotations have been as much as $3 above those of London. The following table gives American prices:

QUOTATIONS FOR QUICKSILVER IN LARGE LOTS.
(Per flask of 75 lb.)

Month.	1908.			1909.			1910.			1911. (a)		
	New York.	San Francisco.		New York.	San Francisco.		New York.	San Francisco.		New York.	San Francisco.	
		Domestic.	Export.		Domestic.	Export.		Domestic.	Export.		Domestic.	Export.
Jan.	$45.00	$45.00	$43.50	$45.50	$45.30	$43.30	$52.20	$50.90	$48.90	$42.00	$41.63	$39.63
Feb.	45.00	45.00	43.50	45.50	45.50	43.50	50.00	49.50	47.50	48.69	48.00	46.00
Mar.	45.00	45.00	43.50	45.50	44.75	42.75	49.25	48.75	46.75	52.00	51.00	49.00
Apr.	45.00	45.00	43.50	45.00	44.25	42.25	48.00	47.56	45.56	49.40	48.90	46.40
May.	45.00	44.50	43.50	44.50	44.00	42.00	47.60	47.15	45.15	45.00	44.50	42.00
June.	44.25	44.00	42.50	44.50	44.00	42.00	47.00	46.25	44.25	43.75	43.44	40.94
July.	44.00	43.50	42.00	43.75	43.44	41.44	47.00	46.25	44.25	48.00	47.60	45.10
Aug.	43.30	42.70	41.30	43.75	42.95	40.95	46.00	45.88	43.88	48.25	47.63	45.13
Sept.	42.87	42.25	40.50	45.00	43.50	41.50	46.00	45.50	43.50	46.88	46.50	44.00
Oct.	46.25	43.50	41.62	47.00	45.90	43.90	45.80	45.40	43.40	45.20	44.70	42.20
Nov.	46.60	44.50	42.50	52.50	50.75	48.75	43.88	43.50	41.50	45.00	44.50	42.00
Dec.	45.75	45.12	43.12	52.50	51.00	49.00	42.00	41.50	39.50	44.25	43.75	41.25
Year.	$44.84	$44.17	$42.54	$46.30	$45.45	$43.45	$47.06	$46.51	$44.51	$46.54	$46.01	$43.62

(a) From *Eng. and Min. Journ.*

California. (By Lewis H. Eddy.)—In San Benito and San Bernardino Counties floods washed out roads and destroyed plants at the mines. The New Idria resumed operation after being out of commission for about a month and having reduced the output by 800 or more flasks.

There was nothing to stimulate production. The price in 1910 fell from $50 per flask in January to $41.50 in December. Much of the ore extracted or developed in the smaller mines was of low grade. The impression obtained in the early part of 1911 that the dissolution in April of the Eureka Company, of California, which had for many years been the selling agency for practically all the product of the state, might stimulate production in the smaller mines; but if it had any effect it was apparently to discourage production.

Of the total production in the state 10,000 flasks came from the New Idria and 6500, approximately, from the New Guadalupe and the Almaden. In the former mine there has been steady advance in develop-

ment and, with the exception of February and March, a steady production. In Santa Clara County there have been some new ore bodies opened and progressive development and increased output; this is true particularly of the New Guadalupe. Besides this there were encouraging developments in the El Dinero prospect in the immediate neighborhood of the New Guadalupe. The Helen and the Great Western in Lake County and the Oceanic in San Luis Obispo County have produced the greater part of the output of the smaller mines. The Altoona in Trinity County has been a small producer under a leasing system.

There are indications of reopening of former producers and some new prospects and locations have been recorded. The Klau mine in San Luis Obispo County has produced no quicksilver in the last year, but the expectation is that the furnaces will be completed and other repairs necessitated by the fire which destroyed the plant will have been made in time to resume operations. In Sonoma County the Cloverdale mine has been reopened and the furnace started. A company was formed in November for the taking over and reopening of the Mercy mine in Fresno County. The mine formerly operated by Alvinza Hayward in Stanislaus County has been taken over by a newly organized company.

Texas (By William B. Phillips[1].)—In 1911, the quicksilver industry in Texas ran on an even keel. There was little or no exploitation of new ground and none of the companies that were idle in 1910 resumed operations.

The deep shaft of the Chisos Mining Co. should now be approaching the Buda limestone, and it will be important to observe whether there is any change in the character of the ore after it has left the overlying bituminous shales. No other shaft in that part of the district has reached so great a depth, 600 ft., nor have any workings in the entire region known as the Terlingua district yielded any better material. The furnace material yields about 1 1/2 per cent. of metal and it is likely that ore of this quality can be depended on for some years.

The mining of the rich pockets in the hard limestones west of Vogel's Draw (western part of the district) has been discontinued and interest now centers around the eastern part of the district, between Vogel's Draw and Rough Run creek.

Certain large holdings of land in the heart of the producing area will be prospected vigorously. These lands can be bought or leased or secured under royalty without the necessity of considering the vexatious and uncertain clauses of the mining law. Some of these lands have already yielded excellent ore.

The yearly productions of quicksilver in Texas are given in an ac-

[1] *E. and M. J.*, Jan. 6, 1912.

ompanying table, and are the statistics of the U. S. Geological Survey, vith some additions that do not materially affect the totals.

TEXAS QUICKSILVER PRODUCTION.

Year.	Flasks.	Value.	Year.	Flasks.	Value.
900	1,800	$ 75,600	1906	4,761	$ 178,829
901	2,932	132,438	1907	3,686	148,387
902	5,319	239,350	1908	2,384	122,260
903	5,029	211,218	1909	4,188	180 084
904	5,336	232,116	1910	3,382	158,958
905	4,723	172,362	1911	2,300	108,000
			Total	46,840	$2,901,602

Texas supplies the St. Louis market, principally, and this market did ıot require more than the reduced shipments in 1911.

Navada and Oregon.—A small production of 100 flasks was derived rom these states. Nevada will soon become a more important producer, .s the Mercury Mining Co. is building a 25 ton Scott furnace at Ione and hould ship early in 1912.

Idaho.—Both the Sturkey and the Willis placers, in Custer County, ind a small quantity of high-grade cinnabar in dredging operations. 'he lode is supposed to occur in porphyry half a mile up the Stanley reek and its discovery is a possibility.

Washington.—The Globe Mining and Milling Company, of Wenatchee, ıas decided to put in a 25-ton furnace for the treatment of quicksilver ›res. The company claims a foot of ore that goes from 8 to 10 per cent. n quicksilver.

Foreign Countries

Algeria.—Quicksilver is mined at Taghit near Batna, and also at the Ras-el-Mâ mine; the output of the latter increasing, the former decreasing. 'he two mines ship together 20 to 30 metric tons of mercury *via* the ›ort of Phillipeville.

Australasia.—According to a correspondent of the *London Mining ournal*, an official report states that native mercury has been found at Yyponga, 8 miles from Willunga, South Australia.

In New Zealand the Ascot cinnabar mine is about 1 mile north of he Talisman mine at Karangahake.[1] The seams of cinnabar are also ıorizontal in flinty rock. It is estimated that the development work is ar enough advanced to guarantee a few hundred tons of ore in sight ıveraging 10 per cent. mercury. The Navok furnace has been installed o treat the ore. The amount of quicksilver ore produced in New Zealand luring 1911 was 2 tons valued at £400.

[1] Mining and Engineering Rev., January 5, 1911.

Austria.—At Idria, Carniola, Austria, 100,899 metric tons of ore valued at 2,373,634 K. were produced in 1910; metallic mercury 602.7 metric tons came from the state owned work at Idria. Several other properties are being developed. The Imperial quicksilver works are described by M. Von Lipold.[1]

Canada.—Important discoveries of cinnabar in northern Ontario are announced. It is stated to have been found in quantity at Groundhog on the Groundhog River, east of Cochrane, where about 20 locations have been staked.

China.—Vermilion made from quicksilver has a steady demand in China as a pigment. Another use of quicksilver in China is for gilding, by the Chinese method. The deposits of commercial importance are those in the province of Kuei-chou, while other occurrences of importance are at Yang-li and Yunnan.

Turkey.—The peninsula of Karaburun (Aidin) has cinnabar deposits being worked by the Karaburun Mercury Syndicate, Ltd. This company has three furnaces. The Sisma mine (Vilayet of Koniah). has 2 per cent. ore and is worked by the Koniah Mercury Syndicate, Ltd.

It is estimated that in 1911 Turkey produced 450 flasks.

United Kingdom.—The imports and exports of mercury, as stated by A. S. Pickering, together with prices are given in the table:

	Imports.	Exports.	Price.	
	Flasks.	Flasks.	Highest.	Lowest.
			£ s d	£ s d
1911	46,547	31,438	10 0 0	7 10 0
1910	44,595	24,748	9 15 0	7 9 0
1909	43,163	22,050	9 17 6	7 19 0
1907	39,448	29,465	8 5 0	6 15 0
1905	34,034	21,330	7 15 0	7 1 0
1903	34,886	18,846	8 15 0	8 5 0

QUICKSILVER PRODUCTION OF THE WORLD.
(Metric tons.)

Year.	Austria.	Hungary.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1897	532	1	192	294	616	1,728	965	4,328
1898	491	7	173	353	362	1,691	1,058	4,135
1899	536	27	205	324	360	1,357	993	3,802
1900	510	32	260	124	304	1,095	983	3,308
1901	525	33	278	128	368	754	1,031	3,117
1902	511	45	259	191	416	1,425	1,208	4,055
1903	523	44	314	188	362	968	1,288	3,687
1904	536	45	357	(e)190	332	1,130	1,192	3,782
1905	519	36	370	(e)190	318	853	1,045	3,331
1906	526	50	418	(e)200	210	1,568	963	3,935
1907	527	40	434	(e)200	130	1,212	712	3,255
1908	594	78	684	(e)200	147	1,068	685	3,358
1909	637	80	771	(e)200	7	1,393	713	3,801
1910	603	90	894	(e)250	(e)20	1,114	763	3,734

(*b*) Not reported. (*e*) Estimated.

[1] Abstract, *M. and Minerals*, Feb. 12, 1912.

RADIUM

The properties of radio-active substances have been much studied of late and it is well known how these studies have led to the better knowledge of the "transformation" and possibly of the disintegration of the elements. The amount of radium in any substance is estimated by the rays from products that are being given off and their effects on photographic plates or by the electroscope. Of these products there are now 28 known. A new method of determining the radium of the contents of minerals is that given by J. Joly,[1] and consists of fusing the mineral to be tested with alkaline carbonate. The usual method of dissolving the mineral and treating it chemically does not show the full amount of radium because it does not give the full strength. The amount of heat given out by pitchblende has been measured as .000065 calories per hour.[2] During the year many other interesting properties have been announced by the scientists.

Radium tests by electrometer give fairly good and rapid results. The instrument must be free from any radioactive substance and only small quantities should be used. Radium emanations will discharge the instrument even if not due to radium itself so that a larger amount of substance may have such an indirect effect.

By photographic plates these are enclosed in light tight black paper; a flat metallic object is placed on this and on top of this a layer of the weighed material is spread, usually in a box. The same exposure time is given for different plates and samples and the development gives a rough idea of the radium present.

In South Australia the State Geologist and Dr. Mawson describe a ledge containing at intervals uranium minerals. It is in the Yudanamutana mining district and extends for 3 miles. A prospecting company has been formed in Adelaide and a good deal of development has already been accomplished. Works will be started at Port Pirie for dealing with ores containing uranium and radium. The value of the radium associated with this ore carrying 1 per cent. uranium, it is estimated would be $291 per ton. The State Geologist expects many deposits to be discovered by further testing all within a radius of a few miles. Besides the uranium ores which are the chief source of radium there are also

[1] *Philosophical Magazine*, 127.
[2] H. H. Poole, *Philosophical Magazine*, p. 121.

reported ores carrying niobium and monazite. The harder minerals are also found in some placer deposits.

Since the discovery of radium and the powerful action of its rays, governments and private enterprises tried to discover and acquire suitable sources of this valuable product. While Governments are prompted exclusively by scientific motives, private individuals are attracted by the high price which is now stated as $80,000 per gram. According to Consul General Gaffney, at Dresden, the cost of producing a gram of radium is about $2000. A radium bank was established in Paris a short time ago. In 1910 it disposed of 1.92 grm. of the highest activity at $80,000 per gram. Of this quantity $15,000 worth was for industrial purposes and $139,000 worth for use in therapeutics. As the radium does not appear to deteriorate, the quantity on hand in the world is continually increasing with production. The consequence of this is that a hiring system has been developed which will somewhat affect the price as the radium production is associated with that of uranium. The competition for uranium in the uranium color manufacture prevents any immediate increase in radium from these ores. The new American production of inferior uranium also interfered with the market and a regular increase in this output would cause further depression of the radium market. The amount of radium distributed in the earth is larger than commonly supposed. It is estimated that the seas contain 20,000 tons, and the sea bottoms 1,000,000 tons of radium.

In Austria the Government mines, being the only two radium producers, are at St. Joachimsthal. They have been recently bought from private individuals at a price, it is stated, of $609,000. The names of these mines are the Sachsische Edelleutestollen and the Hergottszeche. These are deposits of pitchblende and it is estimated that the annual yield will be about 3 grm. of radium, together with 3000 kg. of uranium salts, the latter being worth about $10 per kilogram. These are said to be produced from an output of 16 to 20 tons of uranium pitch ore containing 55 per cent. of uranium oxide. The residue obtained in manufacturing radium-free uranium color is said to be 6 tons per annum. The radium output of the world is practically controlled by the Austrian Government and it is claimed that the annual production from the factory at Joachimsthal will be 5 grm. per year valued at $400,000, utilizing all the material from an old silver mine which has also been a source of supply. In the development of Joachimsthal as a resort for the treatment of disease by radium, the Government has planned, in connection with Saxony, for the extension of the railroad to connect with the Saxon lines including all the tunnels and 25 miles of track. The cost is estimated at $6,000,000. The opening of a large hotel with 300 rooms is

scheduled for May, 1912. The yield in Austria before 1911 was 3 1/2 grm. all told, mostly derived from old stocks probably exhausted now.

Saxony.—The production of radium and researches relating thereto, have been prosecuted vigorously. Uranium ore is known to exist in many mines. Only 1.378 metric tons of suitable pitchblende were produced in 1910. In one of the drives of the Schneeberg district water with radio-active properties was encountered. The State has subsidized some exploration work and the research work is most thoroughly done at the Freiberg Mining Academy.

Scandinavia.—The Swedish radium factory at Islinge has produced 5 centigrm. of radium bromide which was sent to the radium bank in Paris. This was the first shipment and a contract was made by which all the Swedish product of the company will go to Paris. Platinum iridium tubes are used in shipping it as gold has not been found suitable. The Swedish "Kolm" is better than the American or Australian raw mineral because all these have been tried in Sweden.[1] Radium is also found at Evje, Christiansund, Norway.

United States.—Extensive experiments with the treatment of uranium ores to produce radium and radio-active compounds, were carried on during 1911 by a private American syndicate at Pittsburgh. Satisfactory results are reported, but no definite plans are announced.

Attempts have been made in this connection to recover radium from Colorado ores (Placerville). It would appear however that the separation of the uranium from the vanadium in these ores still gives some difficulty. The radium content is said to be 1 grm. in 100 tons. O. B. Willmarth of the Colorado Carnotite Co. of Telluride, states that the Company has a process of chemical concentration for the ores which is very efficient in preserving the radio-activity in the resulting uranium product. The concentrates ordinarily obtained in this region are said to have no increased radio-activity.

Radium occurs in the Colorado carnotite and pitchblende ores. The pitchblende mines near Central City, Colo., are now being financed by an European syndicate. The Calhoun property near Central City has appreciable quantities of ore in places above water level and on the dumps.

Metallic Radium.—According to the process of E. Ebler,[2] radium-barium bromide containing 9 per cent. of radium bromide was dissolved in water and the carbonates precipitated by means of ammonium carbonate. The precipitate was filtered, washed, and while still damp dissolved in the smallest sufficient quantity of an aqueous solution

[1] *Zeitschrift für praktische Geologie*, 8, 1910.
[2] *Berichte*, 1910, 43, 2613, and Curie and Debierne, *Jl. Soc. Chem. Industry*, 1910, 1111.

of pure azoimide. The solution was evaporated in a vacuum over sulphuric acid and caustic potash. There was thus obtained 0.6 milligrm. of a white crystalline substance which remained constant in weight and radio-activity when kept for 8 days in a vacuum over sulphuric acid. The substance was transferred as completely as possible into a narrow glass capillary tube, and again dried.

According to Sir W. Ramsay the Trenwith Mine in Cornwall, has already yielded some 5000 milligrm. of radium product, containing 10 per cent. of pure radium bromide, being the first extracted in Great Britain from British ore. Polonium and actinium also exist in the concentrates of the pitchblende from the Cornish mines.

Radium is also found at Luntwengule Morogoro district, East Africa. Some tests on radio-active minerals found in the southeastern part of Finnland have been tabulated by G. H. Aertovaara.[1] The finely powdered mineral was placed in a thin layer on a pan 53 mm. in diameter and having a central aperture of 18 mm. The layers of mineral were 3 to 3 1/2 mm. thick, and were compared with uranium oxide prepared from uranium nitrate, and also with residues obtained in copper refining at Fergan, Siberia, which are uranium products determined by Mme. Curie to contain two-millionths of 1 per cent. of radium.

URANIUM-RADIUM MINERALS.

Name.	Locality.
Pitchblende (Uraninite)	St. Joachimstahl, Bohemia; Evje, Norway; Gilpin Co., Colo.; also Utah.
Carnotite	Colorado and Utah; also Australia.
Autunite	Madagascar; Tonking, China.
Uranophane	Quebec, Canada.
Samarskite	Caucasus Mts., Russia; Norway.
Mackintoshite	Llano County, Texas.
Thorianite	Ceylon.
Pilbarite	Australia.

The curative power of radium compounds in the form of pastes, gelatines, etc., has been demonstrated in a large number of cases, especially, cancer. While the medical profession knows that these cures have been made and are preferable to surgery, it recognizes that the method is not infallible and has been unsuccessful for some unknown reason in a considerable number of cases.

Radium is now applied to diseased tissues, not by solid pieces of bromide, but by thin coatings of a solution made on celluloid of the proper shape; also in gelatine which on becoming warm and liquefying is charged with emanations, giving both primary and secondary rays. The curative power of springs, as at Carlsbad, is not due to direct solution of

[1] Annals of the Finish Academy of Science, Ser. A, Vol. III.

radium but to secondary or decomposition products that have a definite period before losing the rays.[1]

Thorium is a radio-active element and minor quantities of radium have been found in the monazite sands in North Carolina.

The value of radium compounds imported into the United States in 1911 for the year ending June 30, was $25,631.

The chemistry of the radio elements is discussed by Frederick Soddy.[2]

[1] H. Lieber, *Jl. Franklin Inst.*, Dec., 1911.
[2] The Chemistry of the Radio Elements by F. Soddy, Longmans, Green and Co.

SALT

The production of salt in the U. S. in 1911 is estimated at 31,183,968 bbl. valued at $8,345,692 as compared with that of 30,305,656 bbl. valued at $7,900,344 in 1910.[1] The principal producers are Michigan, New York, Ohio, Kansas and Louisiana, in the order named, the first two states having been the largest by far in recent years and their production being nearly equal in value and weight. In 1910, the relative proportion of grades was the same as before excepting an increase in the proportion of the fine salt and also of rock salt. The increase in these has been gradual but is specially noticeable in the rock salt production in the last 5 years. The grades produced in 1910, statistics from the U. S. G. S., were as follows:

	1909.	1910.
Table and dairy	3,042,824	3,514,748 bbl.
Common fine	7,745,204	6,153,296 bbl.
Common coarse	2,843,393	2,602,737 bbl.
Packers	385,802	327,304 bbl.
Solar	1,283,548	1,223,371 bbl.
Rock	5,938,721	6,965,938 bbl.
Other grades	97,347	129,036 bbl.
Brine	8,770,807	9,389,226 bbl.
Totals	30,107,646	30,305,656 bbl.

In this classification "common fine" includes salt not artificially dried, such as C fine and No. 1 fine "anthracite," etc. Common fine includes all grades coarser than the last and made by artificial heat, such as steam coarse, No. 1 coarse, pan solar, G. A., Liverpool ground, C. C., etc. Packers salt is that sold for packing fish, etc. Mill salt is used in silver or gold mills. Brine includes all salt liquor for soda ash, bicarbonate, caustic or sold without evaporating to dryness.

California supplies salt, some of which is exported, to the extent of nearly 1,000,000 bbl. per year. Most of this is obtained by solar evaporation, although some 80,000 bbl. are mined by one company. The industry is chiefly in Los Angeles County. The price of salt on the coast is high, compared with other parts of the United States. In 1909, the average was 63 cents and in 1910, 55 cents per barrel of 280 lb.

The salt production in Idaho comes entirely from the southern border, in Bannock and Bear Lake Counties. The operators have

[1] U. S. Geological Survey.

shipping head quarters at Afton and Auburn, Wyo. The geology is described by C. L. Breger, *Bull.* 430, U. S. G. S., p. 556. The production is not 1000 bbl. per year.

PRODUCTION OF SALT IN THE UNITED STATES. (*a*)
(In barrels of 280 lb.)

Year.	California.	Illinois.	Kansas.	Louisiana.	Michigan. (*c*)	Nevada.	New York. (*c*)	Ohio, W. Virginia and Pa. (*b*)	Utah.	Other States.	Total Barrels.
1901	601,659	99,700	2,087,791	451,430	7,729,641	13,781	7,286,320	1,385,257	334,484	569,092	20,869,342
1902	682,680	90,009	2,158,486	399,163	8,131,781	14,829	8,523,389	2,318,579	417,501	1,112,824	20,566,661
1903	629,701	(*d*)	1,555,934	568,936	4,297,542	(*d*)	8,170,648	3,043,135	212,995	489,238	23,849,221
1904	821,557	(*d*)	2,161,819	1,095,850	5,425,904	(*d*)	8,600,656	3,030,829	253,829	639,558	18,968,089
1905	664,099	(*d*)	2,098,585	1,055,186	9,492,173	(*d*)	8,359,121	2,728,709	177,342	1,390,907	22,030,002
1906	806,788	(*d*)	2,198,837	1,179,528	9,936,802	11,249	8,978,630	3,436,840	262,212	1,361,494	25,966,122
1907	626,693	(*d*)	2,667,459	1,157,621	10,786,630	6,459	9,657,543	4,007,390	345,557	464,143	28,207,743
1908	899,028	(*d*)	2,588,814	947,129	10,194,279	9,714	9,005,311	3,572,635	242,678	1,291,042	29,719,493
1909	886,564	(*d*)	2,769,849	(*d*)	9,966,744	16,107	9,880,618	3,835,267	246,935	2,505,562	28,754,630
1910	937,514	(*d*)	2,811,448	1,372,248	9,452,022	17,535	10,270,272	3,829,475	249,850	1,365,292	30,305,656
1911	.....	....	.....	.....	.....	.....	10,082,656	.....	.....	.....	31,183,968

(*a*) Statistics of the U. S. Geological Survey except for New York since 1905, which are from reports of the State Geologist. (*b*) The production of Pennsylvania since 1905 is included in "Other States." (*c*) Includes brine used in manufacture of alkali. (*d*) Included in "Other States."

CONSUMPTION OF SALT IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
1898.....	2,465,769	$6,212,554	185,530	$588,653	8,640	$ 63,624	2,642,659	$6,737,583
1899.....	2,759,206	6,867,467	189,051	579,682	12,600	86,465	2,935,657	7,360,684
1900.....	2,921,708	6,944,603	199,909	634,307	7,511	65,410	3,114,106	7,513,500
1901.....	2,879,332	6,617,449	201,733	676,324	9,433	86,414	3,071,632	7,207,359
1902.....	3,338,892	5,668,636	184,764	647,554	5,094	55,432	3,518,562	6,260,758
1903.....	2,655,532	5,286,988	165,981	495,948	12,750	95,570	2,808,763	5,687,366
1904.....	3,084,200	6,021,222	166,140	467,754	13,964	113,625	3,236,376	6,375,351
1905.....	3,635,257	6,095,922	161,159	492,189	34,238	239,223	3,762,178	6,348,888
1906.....	3,944,133	6,658,350	170,505	502,583	33,988	274,627	4,080,650	6,886,306
1907.....	4,160,729	7,439,551	153,435	452,227	30,802	232,895	4,283,362	7,658,883
1908.....	4,024,345	7,486,894	156,609	440,484	26,627	202,338	4,154,327	7,725,040
1909.....	4,215,070	8,343,831	158,487	447,983	40,158	269,273	4,333,397	8,522,541
1910.....	4,242,792	7,900,344	142,549	388,015	(*b*)41,908	286,363	.....	.....
1911.....	4,365,756	8,345,692	137,759	378,083	(*c*)25,911	(*c*)77,083	.....	.....

(*a*) Estimated. (*b*) For year ended June 30, 1910. (*c*) Half year ending Dec. 31, 1911.

Kansas.—The production is increasing and amounted to 2,811,448 bbl. in 1910 as compared with 2,769,849 bbl. in 1909; about one-half the production is mined; in 1910, this amounted to more than 1,300,000 bbl. There are three mines producing, and two companies operate them. Much money has been spent on failures in sinking difficult shafts since the beds were first discovered by drilling for oil and gas in 1888. The deposits underlie the central part of the state and extend into Oklahoma. They are made up of layers of shale, limestone, gypsum and salt, the latter being from 1 to 20 ft. thick. Another difficulty encountered by these enterprises and the salt industry in Kansas is the

high freight rates to such points as Fort Worth and St. Louis; this, however, is only a temporary matter. No bromine is present in the rock salt of Kansas. The milling of the salt consists in crushing in corrugated or toothed rolls and screening on movable bar screens when lump salt is desired. Impact screens are coming into use for the fine screening. The drying of fine salt is done in a revolving cylinder. The grades and uses are as follows: No. 1 is 1/8-in. grains, used for salting hides, for soap, soda ash, caustic; No. 2 is 1/8- to 1/4-in. grains, for salting hides, freezing mixtures, etc.; No. 3, larger than 1/4-in. grains used for refrigerators; No. 4, grains about 1/16 in. like coarse evaporated salt, used for soap, glass and stock; C fine, all sizes from 1/16 in. down, uses similar to No. 1; lump, used for stock. A description of the mining, etc., is given by C. M. Young.[1] The geology and mining have also been given by S. Ainsworth.[2] The technology of the salt industry was described by M. Z. Kirk,[3] and by C. M. Young.[4]

New York (By D. H. Newland).—In 1910, New York outranked Michigan as the leading salt producing state, with a total of 10,270,273 bbl. which was a new record. Statistics received for the industry for 1911 show a slight falling off, the output of 10,082,656 bbl., however, being larger than that of any previous year with the one exception. Conditions in the trade have not been very favorable of late notwithstanding the large volume of production reflected by the above figures. The selling prices of the various grades of evaporated salt have been on the decline for a number of years and at the present level the manufacturer's profit must be very small. Competition with imported salt brought in under the recent reduced tariff regulations is felt most keenly by the local industry since the seaboard markets are its natural field.

There were 30 mines and works active last year. Two of these were rock-salt mines operated by the Sterling Salt Co. and the Retsof Mining Co., both in Livingston County. The manufacture of solar salt which is still carried on extensively in Syracuse and vicinity was represented by 19 plants, all of which sold their product through the Onondaga Coarse Salt Association. A very considerable part of the total output was contributed by the Solvay Process Co. who used the brine directly, without evaporation, for alkali manufacture. The eight artificial evaporating works which contributed to the output were distributed among the five counties: Seneca, Livingston, Schuyler, Tompkins and Wyoming.

Louisiana.[5]—This state has been a producer of rock salt since 1790. The mining methods of to-day are modern and the production annually

[1] MIN. INDUSTRY, XIX, p. 602.
[2] *Engineering and Mining Journal*, Sept. 4, 1909.
[3] *Min. Resources*, Kansas, 1898, pp. 68–123.
[4] *E. and M. Journ.*, Sept. 18, 1909, p. 558.
[5] Paul Wooten, *Mining World*, Feb. 17, 1912.

is 300,000 tons. The salt bed lies along the coast in Iberia Parish and is worked at two points, Week's and Avery's islands, to which the work is confined. On the former the Myles Salt Co.'s mine has a capacity of 1500 tons per day. The island is 9 miles in circumference and is entirely underlain by salt as is shown by systematic diamond drilling. The theory is that the crystallization of the salt caused an expansion and elevation of the island. On this basis a 5 per cent. expansion being an elevation of 200 ft., the deposit could be 4000 ft. thick; the drilling shows salt to the deepest hole being 1200 ft. The overburden of soil is 90 ft. The quality of the salt is 99.84 per cent. pure sodium chloride. One level extending from the 650-ft. shaft is expected to supply the output for years. Rooms 85 ft. high leaving pillars. The salt is crushed underground and screened at the surface ready for shipping. Lump salt is sold for stock, grades A and No. 1 are for ice cream making and car refrigeration. Grade No. 2 is used for curing hides and brine for pickles. A special grade of packing salt is also made. The table salt and bath salt are simply obtained by fine grinding as the quality is very uniform.

Electrical haulage is to replace the mule haulage shortly. The salt is shipped by rail as the Vermilion Bay waters are too shallow for vessels. A projected intercoastal canal would cheapen the shipment.

Utah.—The solar salt industry is described by Percy E. Barbour.[1] There are two companies making salt at the present time. The crude salt is shipped by one company. The Inland Crystal Salt Company conducts large operations. Seven thousand gallons per minute of 17 per cent. brine are pumped from the lake to the ponds. Of the total, 35 ponds of 9 acres each, one-half is harvested. The annual crop is from 2 to 6 in. thick depending on season. In 1910, this was 5 in., 50,000 tons being harvested from 100 acres. The French Atlantic fields yield 11 tons, and the Corsican 68 tons per acre. The composition of the Utah crude salt[2] is given as follows:

$NaCl$	97.85
$MgCl_2$	trace
$CaSO_4$	0.49
$MgSO_4$	0.02
Na_2SO_4	0.06
Al_2O_3	0.01
SiO_2	0.07
Moisture	1.20
Crystallization water	0.30
Total	100

The new mill where this salt is refined is probably the most modern in the United States.

[1] *E. and M. J.*, July 8, 1911.
[2] Analysis by H. Harms.

England.—The salt industry has been dwindling for some years. The amount exported to India and the East averages now only 180,000 tons, when it was nearly double 25 years ago. Much salt has since been supplied from the Mediterranean with cheap solar evaporation. The British Salt Union could not compete with outside sources on account of the railroad transport, but improved appliances have allowed the industry to survive. Recently, a new plant has been established near Liverpool on the Manchester Ship Canal. In Cheshire the center of the great salt beds is near Droitwich and Northwich. Millions of tons of brine have been pumped for years but the saline contents is undiminished. The general method of recovering salt has been by evaporating in open pan plant using Neond gas or coal. The new plant, however, uses a triple effect vacuum system. The capacity of these three evaporators is very large being from 150,000 to 200,000 tons per annum and each one 66 ft. high and 26 1/2 ft. diameter, probably the largest ever built. The Mirrlees Watson Company (Mr. Malcolm, engineer of the salt mines, designer) is the builder. Weston receives the brine through a 12 1/2-mile pipe. The brine is heated in preliminary heaters to 140° F., 100 tons per hour. The heated brine is pumped into the evaporators by electric pump handling 3900 gal. per minute. A 28-in valve admits exhaust steam to the first vessel, which can also be started with live steam. The steam emitted in the third pan is condensed by a jet condenser 62 ft. high and 10 ft. in diameter; this can handle 60,000 lb. per hour requiring 330,000 gal. per hour. Agitators are provided for stirring. The salt is dredged up continuously from the bottom of each evaporator which is provided with a boot, the buckets working through uptakes, made airtight by a seal of brine. The capacity of these elevators is 8 tons per hour. They are driven by 8-h. p. motors and discharge into wooden cars of 5 tons capacity each. The draining brine falls to the feed tanks below. The greater part of the equipment is controlled electrically. The salt here produced contains MgCl and $CaSO_4$ which makes it unsuitable for curing and dairy work. A new process is used by the Salt Union for making pure salt, requiring caustic soda, also made at Weston Point; the pure salt is manufactured at Winsford. It is expected to compete with salt produced by solar evaporation.

SELENIUM AND TELLURIUM

The sources of selenium and tellurium are the residues of the electrolytic copper refineries in the United States and to some extent flue dusts of acid chambers. The ores from which these elements get into the by-products are difficult to trace but these elements are quite universal. Any pyrite is likely to contain selenium and tellurium, although a few are prized as being free from these elements.

All of the copper-refining muds carry much selenium, either as selenium metal or as a selenide. They also carry a high amount of tellurium and when, therefore, they are added to the lead bath selenium and tellurium compounds are likely to occur in the by-products from this process, especially the fumes. Smoke from cupellation will naturally carry these compounds. The mud also contains sulphur, arsenic, sulphates, etc., and in burning the SO_2 reacts with the SeO_2 to form SO_3 and Se. A certain temperature in the flues will give a position where the highest-grade selenium material is found in the settled flue dust. Similarly tellurium occurs at another place close by. The amount of selenium in the dust has been found in one smeltery to be as high as 30 per cent. and the tellurium in the same product about 6 per cent.

The analyses of copper-refining muds vary considerably. One given from Montana[1] is as follows: Silver 40 per cent.; gold, 2 per cent.; copper, 25 per cent.; selenium and tellurium, 5 per cent.; arsenic and antimony 10 per cent.; lead, iron and bismuth sulphates, 18 per cent. The corresponding cathodes produced contain .002 per cent. arsenic; .002 per cent. antimony; .0004 per cent. bismuth; .0009 per cent. selenium, trace tellurium; .0035 per cent. silver; copper 99.96 per cent.; and the balance oxygen. The mud analyses at the American Smelting & Refining plants vary at the plants and also from time to time, but probably the average selenium contents is between 3 and 4 per cent. and tellurium generally does not exceed 10 per cent. of the selenium. Anodes might be .02 per cent. of selenium and tellurium; but some bullion is remarkably free from selenium, so that no average can be estimated.

As far as the possible production of selenium and tellurium is concerned this is known to be very great. More especially in the eastern smelters where copper bullion is refined would it be possible to make a large output, particularly if the purity of the product is not required to

[1] *M. and M.*, Vol. XXXII, p. 622.

be very high. The tellurium is generally associated to the extent of 4 per cent., while perhaps 5 per cent. of the product is oxygen and other impurities, such as arsenic. The cost of producing selenium is estimated at about $1 per pound, although this depends entirely upon the proportion of charges made against the various processes in the course of which it is obtained. It is also estimated that at one works producing about 150 tons of metallic copper per day, an amount of about 1/100 of 1 per cent. of this output could actually be turned out as selenium. The possible production can, therefore, be estimated by including the works of the A. S. & .R at Baltimore and at Perth Amboy, N. J.; the Delamar Works at Carteret, N. J., and the Raritan Works at Perth Amboy, N. J. It is believed that all these works and also the General Chemical Company's works are in a position to produce selenium. But considering the electrolytic copper production of 1,000,000,000 lb. annually, there is a possible production of over 50 tons per year in the United States. The actual production is not known. A number of processes have been used experimentally and produced the crude substances, and stocks were thus obtained and the annual production reduced. In Germany a production of selenium of 750 kg. is reported for the year 1910 valued at 32,500 M. or $3.50 per pound. The price of tellurium is high, about $1 per ounce and the processes for its recovery from copper slime are unsatisfactory.

It has been difficult to obtain pure selenium in the market and it has sometimes been sold as chemically pure when in reality tellurium is present to 5 per cent.

The method of separating by distilling the metals directly cannot be very much recommended nor is it possible to make a clean separation in this way alone. In the laboratory for analytical purposes, distillation in strong sulphuric acid with continuous addition of strong HCl drop by drop in a Knorr still will make a good separation. The tellurium remains behind.

(By Donald M. Liddell).—In medicine, experiments made by Ehrlich on mouse cancer have been successful. The cure of this disease in mice by selenium-eosin and nigrosin derived from coal-tar dyes appears to be accomplished, but it is still a far cry from mouse to human cancer. Tellurium also has a very limited use in medicine. The use of tellurium in resistance wires has been patented.

Apart from the above no new uses have been developed for selenium or tellurium in the last year, and the potentialities of production are far ahead of any demand. So far as can be learned selenium is used as the coloring matter in the best red glasses (such as those used for signal lights); in a limited way in wireless telephony, in phototelegraphy, in

automatic gas lighters, and for other electrical uses depending on its unique property of conductivity varying with the amount of light falling on it; and it is used in some of the yellowish-brown glazes which mark the best art potteries.

There were exports of selenium through the port of Baltimore in 1911, and there was also a good-sized output from the northern copper refineries.

The flue dusts nearest the doré furnaces will be high in tellurium and low in selenium, and those further away, high in selenium and low in tellurium. This property may be utilized to effect a preliminary separation of the metalloids.

In general, the greatest difficulty in the purification of the metalloids is their similarity. It is no great trick to obtain either metal comparatively free from any impurity except its companion. The final separation, so far as my own experience goes, depends chiefly on the methods worked out by Keller, Mager, and Johns, which take advantage of the facts that sulphur dioxide reduces selenium from selenious form in strong hydrochloric-acid solution, while it does not reduce tellurium, and that selenium will dissolve in potassium-cyanide solution, while tellurium will not; and on differences in volatility of the oxides of the metalloids.

Another interesting property which is turned to metallurgical use was pointed out to me by Dr. Edward Keller (according to his knowledge it was also a unique property among the elements); when a solution of an alkaline selenocyanide is electrolyzed, the element is deposited in *metallic* form at *both* anode and cathode.

One of the specific forms of selenium poisoning is likewise a unique feature of this element. The skin lying under the finger nails is peculiarly absorptive of selenious-acid solutions, even as the selenium-eosin compounds take to cancerous tissue. After absorption there the acid is reduced to metal, which appears as red spots under the finger nails. During this reduction period there is a most intolerable burning.

SILICON CARBIDE

By F. J. Tone

The production of silicon carbide in the United States during 1911 was 10,376,620 lb. which is a slight decrease from that of 1910. The progress of the industry may be seen from the following table of production from 1891 to 1911.

PRODUCTION OF CARBORUNDUM IN THE UNITED STATES.

Year.	Pounds.	Metric Tons.	Value.
1891	50	.....	.....
1892	2,145	1	.....
1893	15,200	7	.....
1894	52,190	24	.....
1895	225,930	102	.....
1896	1,190,600	540	$365,612
1897	1,242,929	564	153,812
1898	1,594,152	724	151,444
1899	1,741,245	791	156,712
1900	2,401,000	1,089	168,070
1901	3,838,175	1,742	268,672
1902	3,741,500	1,698	261,905
1903	4,760,000	2,160	333,200
1904	7,060,380	3,203	494,227
1905	5,596,280	2,539	391,740
1906	6,225,280	2,824	435,770
1907	7,532,670	3,418	451,960
1908	4,907,170	2,226	294,430
1909	6,478,290	2,938	388,697
1910	10,707,110	4,857	642,427
1911	10,376,620	4,707	622,597

In the United States the Carborundum Company of Niagara Falls is the only producer. Manufacture on a considerable scale is carried on in most foreign countries including France, Germany, Austria, Belgium, Switzerland and Canada. Two new foreign works are projected for immediate construction. At Arendal, Norway, the Arendals Smelteverk has organized with a capital of 900,000 kr. and has contracted for 3000 horse-power. An English company is projected which will use water-power to be developed on the west coast of Scotland. These additions to producing capacity would point to a considerable over-production unless new outlets are found, which can only occur with greatly decreased costs of production.

While there has been no expansion in the consumption of silicon carbide in the abrasive industries, its use as a refractory has, with decreasing cost of production, become very promising, and the developments in this field in the near future will be of much interest. The selection of

refractory materials has become a most specialized problem. Where formerly the metallurgist had only two or three materials to chose from, he now has at least ten, all of them available to the trade in standard and special shapes and often in many different grades. This calls for a close study of the composition and properties of refractories in selecting a material for a given location.

Silicon carbide has properties which make it unique among refractories. It possesses the highest melting-point or point of decomposition, the greatest hardness, the highest thermal conductivity and a very low coefficient of expansion. While its property of heat conductivity is a disadvantage in many classes of furnace linings where insulation is important, it is a positive advantage in other refractory structures such as muffle furnaces, saggers for porcelain kilns, and zinc retorts. Zinc smelters were among the first to recognize silicon carbide as a refractory and have applied it very extensively in the manufacture of their retorts, where the product finely ground is combined with fire clay, molded and burned in the usual way. Its advantages are its low absorption of zinc, the small loss by filtration of the zinc vapors and, most important, a high fuel efficiency on account of its good conductivity. In the more recent metallurgy of zinc, silicon carbide is also being used to great advantage. Low-grade ores and residues which it has been impossible to treat by the former process are now first subjected to a metallurgical concentration and then to complete reduction in a second furnace. Both electric and fuel-fired furnaces are in use. The units are of large capacity and in some processes work continuously. The refractory requirements in cases have been most severe, but the silicon carbide lining resists the abrasion of the moving charge, and constitutes a dense, impervious and at the same time highly conductive partition between the fuel and the charge, thus solving the refractory problem.

Electric furnace metallurgy has called for refractory material possessing a melting or decomposition point far above ordinary refractories and for this reason silicon carbide is finding wide application in this field It is used in the roofs of arc furnaces for steel making, as a protective coating for electrodes, as a lining in granular resistance furnaces of the type where crucibles are embedded in the resistor or where a heating chamber is formed in the same manner. It is the only material which can be used in proximity to a carbon heating element.

In resistance to abrasion silicon carbide bricks surpass all other refractories. Their mechanical strength, especially at high temperatures, is distinctly superior to that of fire brick. For this reason they are being used in the floors of by-product coke ovens and other locations where the wearing qualities are of importance.

The first determinations of the thermal conductivity of silicon carbide were made by S. Wologdine.[1] His determinations on several refractory materials for a temperature of 1000° C. and expressed in gram calories per second per cubic centimeter per degree C. are as follows: carborundum brick, 0.0231, magnesia brick 0.0071, chrome brick 0.0057, fire brick 0.0042, silica brick 0.0020. This gives a value for the resistivity of silicon carbide of 10.3 thermal ohms per cubic centimeter and of fire brick 57 ohms.

F. A. J. Fitzgerald has recently determined the relative heat conductivity of silicon carbide, alundum or fused alumina, and silica.[2] At a temperature of 435° C. the silicon carbide brick transmitted 79 per cent. more heat than the alundum brick, and 117 per cent. more than the silica. The curves indicate that at higher temperatures the difference would be still greater. Silicon carbide is indicated as the most efficient possible refractory where it is desired to transmit heat through it, as for example, in pyrometer tubes. A trial was cited in which with a clay tube the pyrometer came to constant reading in 38 minutes and with a graphite-silicon carbide tube in 15 minutes, the clay tube giving, moreover, a reading 10 degrees lower than the other. Determinations are given of the electrical resistivity of silicon carbide bricks made up with a small amount of clay binder.

Temperature, deg. C.	Resistivity, cm. cub.
20°	318×10^5
656°	63×10^5
809°	32×10^5
940°	16×10^5
1040°	4×10^5

When the refractory articles are made of silicon carbide without any binder as, for example, from compacted carborundum or silundum the electrical resistivity is much lower.

[1] *Electrochem. and Metallurgical Industry*, Vol. VI, 1909, 383.
[2] *Metallurgical and Chemical Engineering*, Vol. X, 1912, p. 129.

SODIUM AND SODIUM SALTS

As stated in the last volume, metallic sodium is used and made in the United States by two manufacturers, the Virginia electrolytic and the Niagara Electrochemical Co. There are no imports reported. The Niagara Company uses the Castner process, caustic being the electrolyte and hydrogen a waste product. The metal was made in about the same amount as in 1910 when it was estimated at 2000 tons. The furnace used by the Virginia company is described in Vol. XIX. The by-product chlorine is used in the manufacture of chlorine compounds.

The uses of sodium are increasing. Four years ago it was estimated that the world's production does not exceed 3500 tons and was divided about equally between the two chief uses, namely, for sodium cyanide, and for sodium peroxide. Both of these uses have largely increased, but actual production figures cannot be given.

In the manufacture of peroxide a secret process is used; a by-product of pure nitrogen is at present wasted.

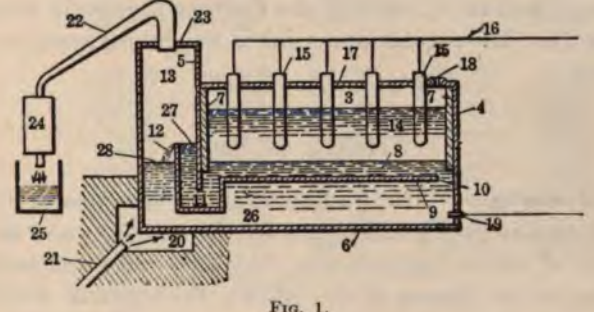

FIG. 1.

Metallic Sodium by Electrolysis of Fused Sodium Chloride.—In the electrolysis of fused sodium chloride with a cathode of molten lead the sodium set free at the cathode alloys with the lead and it is necessary to remove the sodium-lead alloy from contact with the molten electrolyte for two purposes: First to provide a fresh molten lead cathode surface ready to alloy with sodium, and second to recover the sodium from the sodium-lead alloy in the form in which it may be desired, for instance, as metallic sodium. For this removal of the lead-sodium alloy from the electrolytic cell, Mr. Robert J. McNitt, of Niagara Falls, N. Y., in a

patent assigned to the Roessler & Hasslacher Chemical Company, makes use of differences of specific gravity to provide the circulation of the molten electrode material. Pure molten lead has a higher specific gravity than a molten alloy of lead and sodium. If, therefore, in Fig. 1 the channel 26 is first filled with molten pure lead and the compartment 8 (just above the partition 9) and the seal 11 with a molten alloy of lead and sodium on which rests the molten electrolyte, 14, of sodium chloride, there will be a difference between the hydrostatic levels at the points 27 and 28. When a direct current is then passed from the anodes 15 through the sodium chloride 14 to the lead cathode 8 (electrical connection being made at 19 to the external circuit), the sodium chloride is decomposed, the sodium alloying with the molten lead 8. More sodium chloride is now fed into the cell, etc. Meanwhile, of course, a portion of the cathode alloy is forced down through seal 11 under wall 5, over wall 12 into the outside furnace chamber 13. In this chamber 13 the temperature is maintained high enough so that some or all of the sodium in the lead-sodium alloy is driven off by distillation and is condensed in 24. There remains back in 26 a weak alloy which returns through opening 10 into the electrolytic chamber 3. The difference of specific gravity is, therefore, utilized to maintain a difference of levels at 27 and 28 and therefore provides an overflow from 27 to 28. (995,476, June 20, 1911.)

In Norway, sodium is made by the Castner process in the plant built by M. Ashcroft. The regular operation is much hampered on account of lack of power.

Nitrate of Soda

(By W. J. Dwyer)[1].—The Southern States consume probably 85 per cent. of all fertilizers manufactured in the country, and as a result of the low price of cotton last autumn, they have not purchased fertilizers.

According to the figures of the Nitrate Propaganda Association, the 1911 production of nitrate in Chile amounted to 55,100,000 quintals, of 101.44 lb. each, comparing with a total production in 1910 of 53,685,000 quintals. Germany is the largest consumer of nitrate, with the United States second and France and Belgium third and fourth.

The only regular American importers of nitrate are W. R. Grace & Co., Wessel, Duval & Co., and the Du Pont powder works. The Armour Fertilizer Works and the International Agricultural Corporation also brought in direct cargoes during the year.

The Norwegian artificial-nitrate industry has not influenced the

[1] *E and M.J.*, Jan. 6, 1912.

European markets for the natural product, although the production of atmospheric nitrates has been a paying business in Norway for some time. There seems to be a sufficiently large market to take care of both products. A plant is being erected at Charlotte, N. C., and its nitroline product may become a competitor with the Chilean sodium salt (see Nitrogen, p. 554). The initial annual capacity will be 2000 tons. At least one company was formed in California during 1911 to exploit the nitrate beds there. The deposits in Texas also came in for some publicity. Operations on a commercial scale, however, have not apparently been undertaken in any portion of the United States.

IMPORTS OF SODIUM NITRATE INTO THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Quantity.	Value.	Value per ton.	Year.	Quantity.	Value.	Value per ton.
1902.....	205,245	$5,996,205	$29.21	1907.....	364,610	$14,844,675	$40.71
1903.....	272,947	8,700,806	31.88	1908.....	310,713	11,385,393	36.64
1904.....	228,012	9,333,613	40.93	1909.....	422,593	13,281,629	31.43
1905.....	321,231	11,206,548	34.89	1910.....	529,172	16,601,328	31.37
1906.....	372,222	14,115,206	37.92	1911.....	544,878	16,814,256	30.86

(a) As reported by the Bureau of Statistics, Department of Commerce and Labor. The figures of value appear to be doubtful, especially with respect to the earlier years.

NITRATE OF SODA STATISTICS. (a)
(In tons of 2240 lb.)

Year.	Shipments from South America.	Consumed in Europe.	Consumed in United States.	Consumed in World.	Stocks in Europe.	Visible Supply at close of year.
1900.....	1,429,000	1,126,000	175,000	1,324,000	221,000	794,000
1901.....	1,238,000	1,154,000	192,000	1,364,000	243,000	617,000
1902.....	1,360,000	1,028,000	214,000	1,259,000	263,000	660,000
1903.....	1,435,000	1,127,000	265,000	1,412,000	155,000	654,000
1904.....	1,476,000	1,131,000	275,000	1,447,000	162,000	672,000
1905.....	1,623,000	1,190,000	308,000	1,547,000	183,000	674,000
1906.....	1,700,000	1,243,000	355,000	1,636,000	190,000	733,000
1907.....	1,626,000	1,252,000	350,000	1,658,000	202,000	695,000
1908.....	2,017,000	1,378,000	309,000	1,732,000	402,000	928,000
1909.....	2,100,000	1,465,000	407,000	1,938,000	337,000	999,000
1910.....	2,300,000	1,651,000	501,000	2,241,000	310,000	969,000
1911.....	2,406,000	1,696,000	550,000	2,349,000	479,000	1,047,000

(a) Statistics of W. Montgomery & Co., London.

Sodium Nitrate in Chile.—The deposits are the chief government revenue source. The production based on figures issued by the Nitrate Propoganda are as follows:

1908..... 41,751,284 quintals of 101.6 lb.
1909..... 44,794,465 quintals of 101.6 lb.
1910..... 53,685,000 quintals of 101.6 lb.
1911..... 55,100,000 quintals of 101.6 lb.

The government revenues now average $29,000,000 per annum. Since 1880 the export tax collected was $12.34 per metric ton.

Nitrate export taxes for 1911 aggregated $30,154,110, against $18,823,-

780 received by the Chilean Government for import duties during during the same time.

The supply is estimated as follows early in 1911.

Tarapaca	725,300,000 quintals.
Toco	614,925,000 quintals.
Antofogasta (central)	714,184,000 quintals.
Aguas Blancas	1,083,032,000 quintals.
Taltal	2,087,063,000 quintals.
Chañoral and Copiapo	183,700,000 quintals.
Total	5,408,204,000 quintals.

Only one-eighth of the area is stated as being worked. There are no surface indications of these deposits and one can walk over them for miles and not recognize the presence of anything valuable. Prof. J. F. Newsom states that no such deposits have been found in the United States but that no one can tell from the surface.

In Chile where the seasons are reversed and the autumn months here are most active there, the year opened with a price of 7s. 3d. per quintal for prompt shipment; in March, 1911, it fell to 6s. 10d., April 7s., October 8s. Then a severe setback in American demand ensued, prices closing as in Europe at 7s. 4 1/2d. per quintal for spot, and less for future delivery.

Owing to delays in completing contemplated oficinas, the closing of others for shortage of caliche or raw material, the increase in production was only slight. It has been predicted that a shortage of labor will prevent an increase in the industry in any event; but on the contrary, the high wages are an inducement to keep all labor required. If a "combination" or convenio is again necessary it will be the result of a decline in prices, and production will be curtailed. The present industry is divided in the order of importance between Chilean, German and English companies; the latter as has been stated erroneously do not control the industry. Hence, without any such influences it may be expected in 1912 that 2,000,000 quintals will be added to the production by the new plants, the Caya, Savona, Bellavista and Annibal Pinto, and by the modernizing of a number of others. The stocks on the West Coast were in excess of what they were at the end of 1910. Probably this was not intentional on the part of producers; of course, it does not stand them as much as if they had paid for sacking, railroad freight and export duty. The stocks amounted to 450,700 long tons in 1910 and 475,300 in 1911, on December 31. But these notable increases do not bear out the contention that the supply is scarce and they can only be detrimental to the producers.

The agricultural industry had an influence on the consumption in Europe. The summer drought was felt, although the previous year's heavy crop of beet-sugar opened the year favorably. In Germany where the requirements of Bohemia and Moravia are supplied through Ham-

burg, the consumption fell off, as it did also in Spain; but it did not increase in the United Kingdom, excepting Scotland (7510 tons increase), where it is used in making nitric acid in the manufacture of explosives. Holland received a larger tonnage than before, mostly at Delfzyl where it is distributed for agriculture, but also some at Rotterdam for the Rhenish Chemical Works in Germany.

In Vol. XIX of MINERAL INDUSTRY, mention was made of the so-called "propoganda" intended to increase the income of the producers by expending $500,000 during 1911; the increase in consumption was 108,000 tons for the world; unless the effect of this expenditure is to bear fruit in years to come the industry will hardly be compensated by what has resulted, unless perhaps as must be, the consumer pays for the losses.

The cost of production in Chile, f. o. b., duty paid, is estimated at no more than 5s. 6d. per quintal.

Use of Nitrate of Soda.—Nitrogen in some form is indispensible to plant life, very much like phosphorus and potash. As a fertilizer containing nitrate the Chile niter has the advantage of being clean; it is odorless and easily distributed. It is very soluble, so much so that rains in 1910 made its use inefficient since much was washed away; on the other hand, drought in 1911 caused the action to be too slow as repeated moisture addition causes the proper action on the soil. One of the large competitors of nitrate is ammonium sulphate, so largely produced as a coke by-product. The supply of local manures is limited and the farmer will if he uses more nitrate use less sulphate and other nitrogen-containing materials and *vice versa*.

Nitrate of soda is cheaper than ammonium sulphate when the unit of nitrogen is considered. The efficiency is an unsettled question and varies with kind of crops.

With the exception of Dunkirk, France, all the European ports, Hamburg and Antwerp were well supplied with nitrate.

SODA.

East Africa.—The deposit at Lake Magadi is not far from the Uganda Railway. A company has been formed in London for the purpose of working it and it is said that over 6,000,000[1] have been raised to build Railway and plant. The deposit consists of trona or urao, the sesquicarbonate of soda. This salt is composed of two molecules of bicarbonate, one of monocarbonate and three of water. The deposit is very extensive and there is every reason to believe this to be a commercial undertaking. The treatment will be to heat the salt to a dull red, expelling water and carbonic acid, forming soda ash. Oil fuel will be used.

[1] *M. Mag.*, April, 1911.

The Magad or soda lake in German East Africa is fed by the Guasso Njiro and is n south latitude 2 to 3° (36° east of Greenwich); it lies near the English border. The system of depressions or ancient lakes extends from the Red Sea to south latitude 6°. Soda or magadi was used and produced by the natives to add to snuff.[1] The lake is 614 meters above sea level and is 55x24 km. In the dry season the lake becomes swampy or dusty. The incrustations contain 45 per cent. Na_2CO_3 and 38 per cent. $NaHCO_3$; there are no sulphates or nitrates. On March 20, 1911, at a meeting of the goverment commission to continue building the Usambara railroad, the tonnage was confirmed as being 70 million tons available. Recently 50 tons of salt were shipped for testing to Germany.[2]

The Government is trying to inform itself on the soda industry, now British largely, so that a German company will obtain the rights to work this deposit. The British soda deposits of Magadi are about 30 kilometers north.

The salt occurs in bars said to have 68 1/2 per cent. Na_2CO_3; many springs increase the supply continuously. The deposit is 16 meters × 5 kilometers and is 620 meters above sea, surrounded by mountains 2000 meters high.

The deposit is estimated at 30-million tons; the branch R. R. to be built (to the Uganda R. R.) will make it possible to refine the product on the ground, where a plant having a monthly capacity of 10,000 tons of product will be erected. The harbor of Monibassa Kilindini is however quite shallow.

Russian Soda Developments.[3]—In 1911 German soda factories decided to reduce the price of caustic soda by 1 mark per 100 kilograms, or about 2d. per pood (36 lb.), says a Russian correspondent in his comments on the high price of soda in Russia. For 10 years the price of both calcined and caustic soda has remained at about a uniform level. The price of calcined soda was 75–90 copecks in Hamburg and 1 rouble 30 copecks in St. Petersburg, and the respective quotations for caustic soda are 1r. 50–60c. and 2r. 65c., or a difference of 75 per cent., and of course in more distant parts of the country the cost is relatively higher. The Russian soda industry besides being able to supply the entire internal needs of the country uses raw material almost all found in the country. The duty on calcined soda is 90 3/4c., and on caustic soda 1r. 48 1/2k. per pood (36 lb.).

The California Trona Co. worked the surface deposits on Searles or Borax Lake for soda. Before that the San Bernadino Borax Co. made

[1] Dr. Stuhlman, *Handwerk und Industrie, in East Africa*, 1910, p. 48.
[2] *Deutsche Kolonial Zeitung*, 1911, p. 222.
[3] *Chemical Trade Journal*, Feb. 4, 1911.

borax, but a haul of 80 miles to Mojave was unprofitable. The surface of this mud lake is covered with a crust composed of salt, soda and borax. Thicker deposits of common salt occur near the center of the lake and are 25 ft. thick. A crust of solid soda is said to extend over 3 miles, is 500 ft. wide and 2 ft. thick. The hanksite (sodium carbonate, sulphate and trona) beds are encountered at 40 ft. depth.

IMPORTS OF SODIUM SALTS. (*a*)
(In tons of 2000 lb.)

	1907.		1908.		1909.		1910.		1911.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Arseniate	84.2	$9,306	159.1	$13,922	125.4	$7,108	242.3	$14,368	84.2	$5,199
Ash	3,373	73,052	1,979	41,607	2,220	48,674	2,495.0	55,262	1939.1	43,365
Bicarbonate	68	4,274	42.6	2,555	60.9	4,821	73.2	4,472	42.6	2,634
Bichromate and Chromate	3.5	425	54.0	4,304	1,206	97,661	1,332.1	101,687	.....	1
Caustic	642	39,396	566	31,716	391	24,742	519.4	33,882	1,492.1	73,968
Carbonate (Crystal)	63	2,026	30	1,937	84	3,464	69.9	2,569	152.6	4,235
Chlorate	642	39,396	.....	.....	.....	12	.....	.....	.....	2
Chloride (Salt)	155,318	474,833	168,686	471,476	145,678	437,892	138,376.2	374,420	141,227.8	390,043
Hyposulphite	35	1,093	6	448	6	312	22.3	626	14.7	612
Nitrate	342,080	14,041,346	329,605	12,547,611	353,558	12,583,417	542,334.0	16,548,036	546,394.6	17,101,155
Nitrite	389	37,623	100	10,015	76	7,258	566.3	52,237	238.8	21,205
Phosphate	14	855	13	619	25	707	31.0	1,045	22	750
Prussiate	1,074	175,674	857	129,082	981	123,880	649.9	76,648	598	66,971
Sal Soda	380	4,771	249	3,419	252	3,231	241.1	4,630	107.2	1,512
Salt Cake	3,044	37,044	208	2,169	18	767	3,928.0	40,640	11,398.1	121,218
Silicate	618	11,461	744	13,376	509	8,578	599.5	12,994	617.3	11,713
Sulphide	723	20,988	300	9,539	267	7,764	298.3	9,086	228.0	4,751
Sulphite	12	803	58.5	3,259	235.4	10,150	125.5	4,082	493.7	14,810

(*a*) For fiscal years ending June 30.

SULPHUR, PYRITE AND SULPHURIC ACID

By Edwin C. Holden

Sulphur

In recent years Louisiana and Sicily have supplied over seven-eights of the world's sulphur. During 1911 the shipments from Sicily were 456,200 metric tons, an increase of 60,110 tons over 1910, while the production of the United States was 209,561 metric tons or a decrease of 50,138 tons from that of 1910.

The imports of crude sulphur or brimstone into the United States in 1911 were 24,250 long tons valued at $436,725, and the exports 28,103 tons worth $545,420 as against 28,656 tons of imports and 30,742 of exports in 1910.

Despite the rising quotations for pyrite the prices of the various grades of sulphur have remained constant during the year; Louisiana prime at New York at $22.50 up; roll sulphur per 100 lb. $1.05 to $2.15; flour per 100 lb. $2 to $2.40; flowers, sublimed, per 100 lb. $2.20 to $2.60; powdered commercial bags $1.50; Sicilian, extra quality unmixed second crude brimstone to New York at $22.50 per long ton.

The long anticipated decision regarding the classification of Japanese sulphur was rendered in July, 1911, when the U. S. Treasury Department ordered the levying of duty on sulphur of a fineness approximating 100 per cent., on the ground that such sulphur was really refined sulphur and as such was subject to a duty of $4 per ton, as against the unrefined sulphur, which is admitted free under the tariff act. After further hearings, however, the department decided that the Hokkaido sulphur which is roasted in furnaces and contains 90 to 97 per cent. pure sulphur is not refined within the meaning of the act, and in November reversed the ruling so that this sulphur which is exported from Hakodate, Japan, is now admitted free. The revocation, however, does not apply to the so-called Bungo sulphur, which is dutiable at $4 per ton.

Germany and France are large users of sulphur. The former country imported 46,054 tons in 1911 while the exports were 1757 tons.

The imports of sulphur into France were 201,500 metric tons in 1910 and 268,000 in 1911.

Sweden imported 26,836 tons in 1910.

IMPORTS OF SULPHUR INTO THE UNITED STATES.
(In tons of 2240 lb.)

Kind.	1907.		1908.		1909.		1910.		1911.	
	Amt.	Value.	Amt.	Value.	Amt.	Value.	Amt.	Value.	Amt.	Value.
Crude	20,399	$355,944	20,118	$318,577	26,914	$458,954	28,647	$495,988	24,250	436,725
Flowers	1,458	41,216	793	22,562	770	23,084	912	30,180	3,891	83,491
Refined	606	14,589	692	17,227	966	26,021	985	25,869	986	24,906
Precipitated	60	8,426	30	4,012	58	7,565	.....	.....	.....	.....
Total	22,523	$420,175	21,633	$362,378	28,708	$515,624	30,544	$552,037	29,127	545,122

SULPHUR MINING IN THE UNITED STATES

The Union Sulphur Company with its properties in Caleasieu parish, Louisiana, continues to be the controlling American producer. Its output for 1911 was below that for 1910, but was in excess of 200,000 tons.

Wyoming reports the estimated value of her sulphur produced during 1909, 1910, and 1911 respectively at $9000, $14,000, and $38,000. The nearest railroad point to the property of the Sulphur M. & .M Co., is at Belfry, Montana. This property was said to contain 5,000,000 tons

CONSUMPTION OF SULPHUR IN THE UNITED STATES.
(In tons of 2240 lb.)

Source.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Sulphur—Domestic production.	35,098	193,492	215,000	294,000	307,806	307,761	303,000	259,699	206,200
Imports	190,931	130,421	84,579	64,646	20,318	20,118	26,914	28,647	24,250
Total	226,029	3 3,913	299,579	358,646	328,124	327,879	329,914	288,346	230,450
Exports	967	2,493	1,713	14,419	35,925	27,894	37,152	30,742	28,103
Consumption.	225,062	321,420	297,866	344,227	292,199	299,985	292,762	257,604	202,347
(a) Sulphur contents	220,560	314,992	291,909	337,342	286,355	293,985	286,906	252,452	198,300
Pyrites—Domestic production.	199,387	173,221	224,980	225,045	261,871	206,471	210,000	223,700	261,087
Imports	427,319	413,585	515,722	533,346	627,985	668,115	692,385	806,590	1,001,453
Total	626,706	586,806	740,702	758,391	889,856	874,587	902,385	1,030,290	1,262,540
Exports	1,330	.....	.....	.....	.....	.....	.....	.....	.....
Consumption.	625,376	586,806	740,702	758,391	889,856	874,587	902,385	1,030,290	1,262,540
(b) Sulphur in domestic	87,730	76,217	98,991	99,020	115,223	90,847	92,400	98,428	114,878
(c) Sulphur in foreign	200,215	194,385	242,389	250,673	295,153	314,015	325,420	379,097	470,683
Total sulphur content	287,945	270,602	341,380	349,693	410,376	404,862	417,820	477,525	585,561
Grand total sulphur consumption. (e)	508,505	585,594	633,289	687,035	(d)696,731	(d)698,847	704,726	729,977	783,861

(a) Includes crude and refined sulphur. Sulphur content of crude is computed at 98 per cent. (b) Computed at 44 per cent. (c) Computed at 47 per cent. (d) This figure is in excess of the true consumption as a large percentage of the domestic output of sulphur was stored. (e) The above totals are of course exclusive of by-product sulphur.

of 50 per cent. rock. The Wyoming Sulphur Company has thoroughly prospected a portion of its sulphur-bearing area, which is 1/2 mile wide and 1 1/2 miles long. A plant with 20 tons daily capacity has been erected. The deposits lie about 3 miles north of Thermopolis and are bedded. The company was a producer in 1911.

The Western Sulphur Company of Duluth mined some 2000 cubic yards of 11 per cent. sulphur rock in Idaho, erecting a modern plant near the mine for refining the product. The operation is reported to have been unsuccessful commercially.

Utah produced a small quantity at Black Rock, Millard County.

Nevada, although claiming large sulphur deposits in both the northern and southern portions of the state, reported no production during 1911.

The output of Arizona was negligible.

Foreign Sulphur Mining

India.—The sulphur in the Mekran series in the State of Las Belas, is deposited from springs and is of no commercial importance. The deposits in the Siwalik series, however, are of a different nature and contain, besides, alunite. The mines near Sanni had been worked for years but were stopped on British occupation. Some of the claybands are so full of sulphur that they burn with ease. Small amounts are now extracted for making gun-powder locally.

Japan.—Before the exploitation of the Louisiana deposits in 1903, Japan ranked second only to Italy as a producer of sulphur. Since that date it has with a gradually increasing production held third place. The Japanese product supplies more than half of the sulphur imports of the United States. The deposits are of solfataric type. In 1910, 43,155 metric tons were produced as against 36,317 in 1909. Of the sulphur produced in 1910 about 17,400 tons were exported to the United States; in 1911, the corresponding quantity was 16,185 tons valued at $280,000.

Formosa.—In Formosa sulphur is produced from volcanic craters. The output for the past 5 years has been:

1906..... 801 metric tons valued at 15,876 yen.
1907.....1316 metric tons valued at 28,494 yen.
1908.....1969 metric tons valued at 44,593 yen.
1909.....2162 metric tons valued at 53,599 yen.
1910.....2245 metric tons valued at 52,725 yen.

Mexico.—According to consular reports the great bulk of Mexican sulphur is obtained from the mines near Cerritos in San Luis Potosi, the production being about 800 metric tons of refined sulphur per month. The crude ore carries from 37 per cent. to 90 per cent. sulphur and the

deposit is of the chimney or crater type. The lowest present workings are but 190 ft. deep and the deposit is claimed to be one of the largest in the world. The mines employ some 700 men.

Of the product, about one-third is marketed in Mexico, the present price being around $32 per metric ton. The property is owned by the Virginia-Carolina Chemical Co., but is under lease to German interests and the excess over domestic demands is shipped to Germany. For imports, see p. 941.

Sulphur occurs elsewhere in the state and at Venado, about 50 miles north of San Luis Potosi a deposit of 30 per cent. sulphur has been developed and is equipped with Italian furnaces to produce 3 to 4 tons per day.

The deposits in Lower California near the border were described in Vol. XIX of the MINERAL INDUSTRY. They are of prospective importance but were idle during the past year.

Peru.—In northern Peru in the department of Piura the Sociedad Azufrera de Sechura is reported to have produced 150 tons of sulphur daily. This is probably an overestimate.

Sicily.—The last annual report of the Consorzio shows a continued decline in production. Five years ago there were 720 mines working, whereas there are but 476 now. Several factors have contributed to this result. Government control, restriction of credit for sulphur mining, and scarcity of labor due to heavy emigration are the important causes.

Official statistics for the past 5 years are as follows:

(In Tons of 2240 lb.)

	Production.	Sales.	Shipments.	Stock at End of Year.
1906–7	427,719	161,318	322,000	529,896
1907–8	407,277	441,055	384,000	546,399
1908–9	396,295	300,552	349,000	593,461
1909–10	396,737	409,645	397,000	586,627
1910–11	391,978	816,818	445,500	523,141

The heavy sales during the last year are for future deliveries. A decline of over 25,000 tons in exports from the United States to Europe in 1911 probably aided in making the above satisfactory reduction in the Sicilian stock.

The Consorzio's policy to sell crude sulphur in competition with foreign pyrite has resulted in one company starting to use sulphur for making sulphuric acid. Gill furnaces now return 60 per cent. of the total refined.

Parsons & Petit, the United States agents of the Consorzio, report

that shipments to the United States and Canada in 1911 were slightly curtailed. The usual average is about 15,000 tons. A table shown herewith gives the shipments of Silician sulphur to various countries.

TOTAL EXPORTS OF SULPHUR FROM SICILY, 1903–1910.
(In metric tons.)

Country.	1903.(a)	1904.(a)	1905.(a)	1906.(a)	1907.(a)	1908. (d)	1909. (d)	1910.(d)	1911. (d)
Austria	17,926	23,374	25,111	22,756	24,597	32,501	24,820	29,800	35,700
Belgium	15,233	13,627	14,442	13,940	8,853	11,410	16,536	14,315	10,300
France	74,372	103,040	96,170	67,536	59,725	93,829	87,831	93,230	115,300
Germany	32,553	31,613	28,319	34,967	37,100	30,399	28,788	30,230	28,660
Greece and Turkey	22,133	25,376	25,069	26,560	27,608	27,810	21,131	21,090	25,300
Holland	5,157	8,122	4,425	5,539	11,379	8,775	7,184	9,730	12,650
Italy	45,572	79,619	99,633	79,519	58,926	60,551	50,602	63,200	75,400
Portugal	14,064	8,373	13,196	12,302	12,778	12,294	14,797	13,500	22,550
Spain	4,099	4,064	2,478	3,120		6,125	5,928	5,260	22,550
Scandinavia (c)	28,292	20,120	18,288	21,608	25,155	27,209	18,703	20,210	30,200
Russia	15,068	15,141	16,673	16,181	15,210	19,960	19,366	25,870	23,485
United Kingdom	19,210	18,108	18,847	20,883	16,561	20,119	19,374	19,085	19,800
United States	155,996	100,000	70,332	41,283	9,476	12,006(e)	19,491(e)	12,060	8,560
Other countries(b)	25,833	25,167	23,277	21,238	26,646	13,078	24,368	38,540	58,325
Totals	475,508	475,745	456,260	387,432	334,014	376,066	358,919	396,120	456,230
Stock in Sicily, Dec. 31	361,220	396,541	462,437	525,115	576,377	614,866	640,409	640,711	551,420

(a) Reported by Emil Fog & Sons, Messina. (b) Mainly South Africa, Northern Africa, Asia, Australia, and the East Indies. (c) Including Norway, Sweden and Denmark. (d) Reported by Parsons & Petit, New York. (e) Includes Canada.

South Africa.—It is reported that deposits running 65 per cent. sulphur have been discovered and that a syndicate has been formed in Cape Town to exploit them.[1]

WORLD'S PRODUCTION OF SULPHUR. (a)
(In metric tons.)

Year.	Austria. (b) (e)	Chile.	France. (b)	Germany.	Greece.	Italy. (b)	Japan.	Spain.	United States.	Total.
1895	932	.....	4,213	2,061	1,480	370,766	15,557	2,231	1,676	398,916
1896	781	940	9,720	2,263	1,540	426,353	12,540	1,800	3,861	459,798
1897	642	664	10,723	2,317	358	496,658	12,013	(b)3,500	1,717	528,592
1898	589	1,256	9,818	1,954	135	502,351	10,339	3,100	2,770	532,312
1899	671	989	11,744	1,663	1,150	563,697	10,241	1,100	1,590	592,290
1900	985	2,472	11,551	1,445	891	544,119	14,439	750	4,630	581,282
1901	5,048	2,516	6,836	963	2,336	563,096	16,548	610	6,977	604,930
1902	3,826	2,636	8,021	487	1,391	510,333	18,287	450	7,565	552,996
1903	4,610	3,560	7,375	219	1,266	553,751	22,914	1,680	35,660	631,035
1904	6,431	3,594	5,447	209	1,225	527,563	25,587	605	196,588	767,249
1905	8,542	3,470	4,637	205	1,126	568,927	24,652	610	218,440	830,609
1906	15,258	4,598	2,713	178	(d)1,000	499,814	27,589	700	298,704	845,956
1907	24,199	2,905	2,000	176	(d)1,000	426,972	33,329	3,612	312,731	801,911
1908	17,429	2,705	2,189	811	(d)1,000	445,312	33,419	13,872	312,700	829,437
1909	12,856	4,508	2,900	1,185	(d)1,000	435,060	36,317	21,750	303,000	817,608
1910	15,976	(c)	2,641	1,272	(d)1,000	430,360	43,155	30,113	259,699	.....
1911	(c)	(c)	(c)	1,251	.....	414,671	(c)	(c)	246,300	.....

(a) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to between 20,000 and 30,000 long tons annually, is not included. (b) Crude mineral; limestone impregnated with sulphur. (c) Not yet reported. (d) Estimated. (e) Includes such production from Hungary.

[1] *Chemical Trade Jour.*, Feb. 3, 1912.

Technology

On receiving the Perkin medal Mr. Herman Frasch, inventor of the successful sulphur-well practice in Louisiana, made an address giving a full history of the development of the process and its subsequent improvements. Notable among the latter are the use of compressed air in the ratio of two parts to one of sulphur to make it possible to pump the molten sulphur, the use of 12-in. casing with telescopic joints and stuffing boxes to prevent breakage by excessive grip of the sand and earth, the artificial sealing of the rock around the melting zone to prevent cold water leakage freezing the well and thus shortening its life, and finally, filling in broken wells by blasting them and with a 4000 cu. yd. dredge refilling the cavity left by the abstracted sulphur.

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Pyrite

Stimulated by an active demand, the production of pyrite in the United States during 1911 was 261,087 long tons valued at $1,062,914 as against 223,700 tons valued at $830,150 in 1910. Virginia is still the largest producer, New York, Illinois and Wisconsin being next in order. The imports of sulphur ore containing over 25 per cent. of sulphur were 1,001,453 tons valued at $3,788,632 in 1911, as against 806,590 tons worth $2,773,627 in 1910. The total consumption of pyrite in the United States, exclusive of by-product pyrite, was thus 1,262,540 tons, or an increase of 22 per cent. over 1910.

The imports for the fiscal year ending June 30, 1911, showed a total of 894,281 tons received, of which 742,758 were from Spain, 117,996 from Portugal, and the remainder from Canada and Newfoundland. Spain and Portugal thus supplied almost 70 per cent. of the total pyrite

primarily so used in the United States. By-product sources of sulphur, such as pyritous copper, zinc, or gold ores, are of constantly increasing importance, and in order to credit the mines with this output which does not appear in the pyrite returns, this product is treated under the subject sulphuric acid.

Markets.—The rise in price of pyrite during 1910 was continued through 1911. Domestic lump ore in January, 1911, was quoted 11 1/2 to 12 cents per unit of sulphur per long ton, domestic fines from 3/4 to 1 cent lower; foreign lump ore 12 to 12 1/2 cents, fines 1/2 cent lower, and arsenical fines 1 1/2 cents lower. At the close of December, 1911, domestic lump had risen to 12–12 1/2 cents, fines 10–12 cents, and foreign lump to 12–13 1/2 cents, fines 12 1/2 cents, and arsenical fines 12–12 1/2 cents.

Production, imports and consumption of pyrite in the United States in long tons are shown in the following table:

PRODUCTION, IMPORTS AND CONSUMPTION OF PYRITES IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production.		Imports. (b)		Consumption.	
1897	133,368	$404,699	259,546	$847,419	392,914	$1,252,118
1898	191,160	589,329	171,879	544,165	363,039	1,133,494
1899	178,408	583,323	310,008	1,074,855	488,416	1,658,178
1900	201,317	684,478	322,484	1,055,121	523,801	1,739,599
1901	234,825	1,024,449	403,706	1,415,149	638,531	2,439,598
1902	228,198	971,796	440,363	1,650,852	668,561	1,622,648
1903	199,387	787,579	425,989	1,628,600	625,376	2,416,179
1904	173,221	669,124	413,585	1,533,564	586,806	2,202,688
1905	224,980	752,936	515,722	1,780,800	740,702	2,533,736
1906	225,045	767,866	597,347	2,138,746	822,392	2,906,612
1907	261,871	851,346	656,477	2,637,485	918,348	3,488,831
1908	206,471	744,463	668,115	2,624,339	874,586	3,368,802
1909	210,000	756,814	692,385	2,428,638	902,385	3,185,452
1910	223,700	830,150	806,590	2,773,627	1,030,290	3,603,777
1911	261,087	1,062,914	1,001,453	3,788,632	1,262,540	4,851,546

(a) These statistics do not include the auriferous pyrite used for the manufacture of sulphuric acid in Colorado. (b) Net imports, less re-exports of 3060 tons in 1902 and 1330 tons in 1903.

Pyrite Mining in the United States

Virgina produced 60 per cent. of the domestic pyrite during 1911 and New York 20 per cent. Illinois, Wisconsin, Georgia, Ohio, California, and Massachusetts followed in the order named. The prospective pyrite developments of importance in the United States are not numerous, and, despite the increase in domestic output during 1911 it did not keep pace with the increase in consumption. Only 20 per cent. of the pyrite used was from domestic ores. The great future source of cheap sulphur in the United States lies in the further extension of the by-product processes of making acid, as practised in Tennessee, Illinois, Indiana, California, and elsewhere.

California.—In 1911, 6341 tons of pyrite valued at $31,352 were mined and used as such. In addition to this an indefinite tonnage, which makes the large percentage of the totals given in the regularly published reports, is responsible for the acid produced as a by-product at two of the large copper smelters.

Georgia.—There were but two producers in Georgia during 1911. Of prospective importance may be mentioned the Chestatee Pyrites Company, which owns a tract of 744 acres containing a bedded vein of pyrite occurring on the contact between a quartzose mica schist and a hornblende schist. The vein outcrops for 3600 ft. and is traceable for a total of 5800 ft. It has been developed by a 225-ft. inclined shaft, 300 ft. of drifting, and 1250 ft. of diamond drilling, and is proved to vary from 4 1/2 to 40 ft. in width, averaging not less than 30 ft. The ore is a clean pyrite free from arsenic and pyrrhotite, with a light micaceous gangue. A test shipment of lump ore carried 43.52 per cent. sulphur and 3.09 per cent. copper. It burned satisfactorily without clinkering. Upon securing railway connections with the Southern Railway at Gainesville this property should become an important producer.

Indiana.—A number of coal mines produce pyrite as a by-product, but the quantity is small and it does not reach the market, some of the companies merely waisting the product.

New York.—The state produced about 55,000 tons of pyrite averaging 45 per cent. sulphur during 1911. The St. Lawrence Pyrites Company continues to be the only important producer and mined and milled about 125,000 tons of crude ore, most of which came from the Anna mine. The Stella mine was operated only 3 months during the year. The finished product carrying about 45 per cent. sulphur realized about 11 3/4 cents per unit of sulphur.

Virginia.—This has long been the largest pyrite producing state, yielding over half of the total output of the United States exclusive of by-product pyrite. The ore is mined principally in Louisa and Prince William Counties from lenticular deposits which are sometimes several hundred feet long and 60 to 80 ft. thick. At some mines there is sufficient chalcopyrite to justify precipitating the copper from the mine water, and at the Cabin Branch mine near Dumfries these is a small copper smelter.

Pyrrhotite occurs also in Floyd, Carroll, and Grayson Counties and is mined by the Pulaski Mining Company near Monaret, Carroll County, and the ore treated at Pulaski for sulphur.

The total production of pyrite, including pyrrhotite, in Virginia for the past 6 years is as follows:

Year.	Long Tons.	Value.	Average Price per Ton.
1906	128,794	$431,378	$3.35
1907	124,740	372,586	2.99
1908	116,340	435,522	3.74
1909	114,176	423,283	3.71
1910	140,106	525,437	3.75
1911	150,000 (e)	570,000 (e)	3.80 (e)

Wisconsin.—The zinc ores of southwestern Wisconsin have associated with them large quantities of pyrite and marcasite as gangue minerals. With the growing demand for sulphur for chemical plants in Indiana and Illinois and the recent improvements in methods of separating pyrite from sphalerite, the shipments of pyrite from the state have grown from 6627 long tons in 1909 to 12,803 long tons in 1911, with indications of a decided increase in 1912 if the present demand is maintained.

WORLD'S PRODUCTION OF PYRITES.
(In metric tons.)

Year.	Belgium.	Bosnia.	Canada.	England.	France.	Germany.	Hungary.	Italy. (a)
1897	1,828	.....	35,291	10,752	303,488	133,302	44,454	58,320
1898	147	3,670	29,223	12,302	310,972	136,849	58,079	67,191
1899	283	.....	25,112	12,426	318,832	144,623	79,519	76,538
1900	400	1,700	36,308	12,484	305,073	169,447	87,000	71,616
1901	560	4,570	31,982	10,405	307,447	157,433	93,907	89,376
1902	710	5,170	32,304	9,315	318,235	165,225	106,490	93,177
1903	720	6,589	30,822	9,794	322,118	170,867	96,619	101,455
1904	1,075	10,421	29,980	10,452	271,544	174,782	97,148	112,004
1905	976	19,045	29,713	12,381	267,114	185,368	106,848	117,667
1906	908	13,474	35,927	11,318	265,261	196,971	112,623	122,364
1907	397	3,671	35,494	10,357	283,000	196,320	99,503	126,925
1908	357	5,000	42,934	9,599	284,717	219,455	95,824	131,721
1909	214	(b)	58,645	8,564	273,221	198,688	98,971	149,084
1910	213	571	48,871	10,393	250,432	215,708	92,464	165,688
1911	.....	3,118	74,994	10,000	.....	217,459	.....	165,273

Year.	Japan.	Newfound-land.	Norway. (c)	Portugal. (c)	Russia.	Spain.	Sweden.	United States.	Total.
1897 ..	7,626	32,790	94,484	276,738	19,380	100,000	517	133,502	1,252,472
1898 ..	8,726	32,335	89,763	302,686	24,570	70,265	386	194,219	1,341,383
1899 ..	8,376	26,154	95,636	347,234	23,250	107,386	150	181,263	1,446,782
1900 ..	16,166	*Nil*	98,945	402,870	23,154	34,638	179	204,538	1,464,512
1901 ..	17,589	7,532	101,894	443,397	30,732	33,953	*Nil*	238,582	1,568,999
1902 ..	18,580	26,000	121,247	413,714	26,465	145,173	*Nil*	231,849	1,713,654
1903 ..	16,149	42,674	129,939	376,177	22,780	155,739	7,793	202,577	1,692,812
1904 ..	24,886	61,166	133,603	383,581	31,667	161,841	15,957	175,992	1,696,099
1905 ..	25,569	51,534	162,012	352,479	30,689	179,079	20,762	228,580	1,789,816
1906 ..	36,038	28,583	197,886	350,746	20,660	189,243	21,827	228,646	1,832,475
1907 ..	56,166	28,000	236,038	241,771	18,316	225,830	27,000	266,061	1,854,849
1908 ..	33,867	(e)35,000	269,129	81,417	56,345	263,457	29,569	209,774	1,768,365
1909 ..	27,066	*Nil*	282,606	12,337	(b)	236,000	16,104	213,371	(e)1,730,000
1910 ..	78,418	*Nil*	322,000	17,630	(e)50,000	294,184	25,445	227,280	1,826,854
1911 ..	.....	2,500	(e)350,000	.....	.....	.....	.....	265,272	.....

(a) Cupriferous in part. (b) Reports not yet available. (c) Both iron and copper pyrites. (d) Not including Newfoundland. (e) estimated.

The basis of settlement for sulphur in the district during 1911 was from 11 cents per long ton unit for 35 per cent. ore to 12 cents for 45 per cent. ore less freight to acid works. Impurities are restricted to 3 per cent. lime, 1 per cent. lead, and 6 per cent. to 7 per cent. zinc. The net shipping returns to the mines average from $3 to $4 per ton.

Pyrite Mining in Foreign Countries

Australia.—In South Australia, the pyrite produced in 1910 was valued at £3270 and in 1911, £2580. The sulphuric acid produced in 1910 was worth £3370 and in 1911, £6840.

Canada.—The production of pyrites in Canada during 1911 was 82,666 short tons, valued at $365,820. This included 39,122 tons of copper pyrites shipped from mines in the province of Quebec, valued at $247, 555, and 43,544 tons of iron pyrites from mines in Ontario, valued at $118,265. Of the iron pyrites about 31,000 tons would average from 42 to 43 per cent. of sulphur, the balance running somewhat lower than this.

Shipments were made from several properties in Hastings County, from the Helen mine, Michipicoten, and from the Vermilion mine at Dinorwick on the Grand Trunk Pacific.

The exports of pyrites during the year, according to the record of the Customs Department, were 32,102 tons, valued at $120,585.

Norway.—The production of pyrite in Norway is rapidly becoming important. There are a dozen producing companies and the mines are scattered from south of Bergen to the extreme northern end of the peninsula. The deposits are usually replacements in chlorite schist at or near contacts with gabbro and mica schist. The ore shoots are usually presistant and of moderate thickness, although several of the larger properties have developed some very large masses. The ore is usually copper-bearing and free from arsenic. The practice is to separate the higher-grade copper ore either by sorting or wet concentration and it is smelted locally to blister copper. The bulk of the ore is classed as export ore and carries from 1 to 3 per cent. of copper and 42 to 49 per cent. sulphur. It is shipped to southern Norway, Sweden, England, Germany, and Russia.

The production of export ore in metric tons for the last 10 years is as follows:

1901	101,900	1906	194,500
1902	121,200	1907	234,500
1903	129,900	1908	251,500
1904	133,500	1909	282,606
1905	161,000	1910	322,000 ca.

The production has thus trebled within 10 years. An increase to

about 400,000 tons was anticipated for 1911 but a miners' strike closed most of the mines from June until September so that a decrease will probably be shown. The principal producers and their outputs during 1910 were:

Sulitelma	120,000 tons.	Bossmo	27,500 tons.
Foldal	62,000 tons.	Killingdal	25,000 tons.
Orkla	50,000 tons.	Roestvangen	14,239 tons.
		Roeros	13,375 tons.

Sulitelma and Roeros also shipped a total of 1750 tons of blister copper. Several of the above companies are planning extensive enlargement of their milling capacity and as a number of promising prospects are being developed a considerable increase in production during the next few years may reasonably be expected.

Russia.—The richest pyrites deposits in Russia occur in the Urals in a band along the eastern slope parallel to the mountain range. In the Goroblagodat district the Spaso sulphur pyrites mine worked by P. K Ushkoff & Co. produces about 16,000 tons a year. The Kalatinsk deposit on the Perm railway produces from 10 to 13,000 tons of ore and is developed to yield three times this output if needed. The ore carries 41 per cent to 48 per cent. sulphur and contains 2 per cent. to 3 per cent. copper, but some arsenic. The mine near the station of Rudyanka contains no copper but is high (51 per cent.) in sulphur. It ships about 10,000 tons a year and could easily increase its production if required. Also on the Perm railway near the Polieff factory is the Systersk mine, which is rich in copper at a depth of 70 ft. In the district of the Kussktim Mining Works the deposit of Tysoff similar to the latter contains up to 7 per cent. copper at depth. The Annensk, Konynchovsk deposit 4 miles further north is not quite so rich in copper. Most of the product is used in local chemical factories and the output could be increased. The ore is also used in the chemical districts of the Kama and the Volga and the district of Moscow.

In the central and southern mining districts of Russia pyrites occurs with coal from which it is sorted to a 30 per cent. sulphur product. About 2500 tons are produced in this way per annum. Much carbon exists in the product.

About 5000 tons of pyrites per year are produced from copper-bearing sulphides in the Caucasus.

Spain and Portugal.—The importance of these countries as pyrite producers is to some extent concealed by the government statistics of Spain which officially classify cupriferous pyrites as copper ore. As a large proportion of the tonnage of the Huelva district is leached for its copper contents it is classed as copper ore, and thus in recent years, while the Spanish returns have shown in the neighborhood of only 250,000 tons

annual pyrite output, the imports into the United States alone of sulphur-bearing ore from Spain have been from two to three times that quantity.

The following table of shipments from the port of Huelva of copper and iron pyrites during the year 1910 more nearly indicates the scale of the industry.

Shippers.	Cuprous Pyrites.	Iron Pyrites.	Total.
Rio Tinto Company, Ltd	596,324	694,816	1,291,140
Tharsis Sulphur & Copper Mines, Ltd	25,149	443,065	486,214
Societe Francaise des Pyrites de Huelva	2,469	214,641	217,140
Pena Copper Mines, Ltd	17,399	121,422	138,821
Esperanza Mining Co, Ltd	88,845	11,443	100,288
Huelva Copper Mines, Ltd	12,484	2,822	15,306
San Miguel Copper & Sulphur Mining, Ltd	8,158	52,984	61,142
Societe des Mines de San Platon	33,140	15,772	48,912
United Alkali Co., Ltd	73,489	56,187	129,676
La Hispalense Mine	7,805	342	8,147
Hijos de Vazquez Lopez	345	36,712	32,057
Societe des Mines de San Pedro	6,385	4,043	10,428
C. and J. Sundheim	.....	4,929	4,929
Other mines	2,976	700	3,676
Total	874,968	1,659,908	2,534,876

The above shipments were made to the following countries:

	Cuprous Pyrites.	Iron Pyrites.
Algiers and Natal	.....	14,263
Australia	.....	15,192
Austria-Hungary	.....	24,195
Belgium	22,845	48,314
Denmark	10,399	.....
France	16,243	239,842
Germany	125,885	256,409
Netherlands	236,070	171,797
Italy	2,884	65,691
Norway	1,878	.....
Portugal	.....	9,104
Russia	204	22,154
Spain	61	50,719
Sweden	5,856	17,858
United Kingdom	221,021	278,146
United States	231,622	446,224

Rio Tinto is the greatest pyrite mine in the world and is credited with a production of about one million long tons of copper during the past 20 years, and it it claimed to have some 60 years proven ore ahead. The average copper content of the ore mined has declined in recent years and it is understood that the deepest exploration has developed a huge body of pyrite practically free from copper.

The property of the Tharsis Company after 45 years successful exploitation is approaching exhaustion.

The Pena Company was engaged in litigation with the Rio Tinto

Company regarding selling contracts and transportation a year ago, and is now building its own railway to the port of Seville, which is being improved. The American Metal Company has recently become the American agent for the sale of Pena ore.

Imports of pyrite into Italy from Spain were 63,526 metric tons in 1911. A total of 103,905 tons imported were valued at 2,182,005 lire.

The exports of iron pyrite from Italy were: 1909, 1094 tons valued at 19,692, 1910, 807 tons valued at 14,526 and 1911, 1613 tons valued at 29,034 lire.

Sweden.—The production of the Stora Kopparbergs Company in 1911 was 21,592 metric tons of pyrite. The same company produced 6159 tons of sulphuric acid.

Technology

Utilization of Iron in Pyrite.—There are now at least five nodulizing plants with nine 100-ton capacity kilns in operation in the eastern states handling spent pyrites. After treatment in the acid plants, the pyrites still carry a small percentage of sulphur. This is reduced to a trace in the finished product from the nodulizing plants and the fines have become agglomerated, thus making an artificial ore of ideal character for the iron blast-furnace. The conversion of the iron from a waste product into an asset virtually reduces the freight on the sulphur content by about 40 per cent. and has thus stimulated imports from Europe.

Recent experiments in sulphite manufacture in New York for pulp, are reported as favorable to the use of cheap low-grade pyrite from 25 to 30 per cent. sulphur. This is a large industry for consuming sulphur now used at about $26 per ton. In one locality 900 tons of pulp are made a day and this consumes 35 tons of sulphur. It is questioned whether the plant and apparatus will be changed for using pyrite unless the experiments are conclusive. It is probable also that sulphur can be produced more cheaply than its present selling price indicates.

Estimating Sulphur in Pyrites.—In estimating sulphur in pyrites, to obviate the inconvenience attending the presence of ferric salts in the solution from which sulphuric acid is to be precipitated by barium chloride, the ferric salt may be reduced to ferrous, and traces of nitric acid at the same time be destroyed by boiling the acid solution with sufficient sodium hypophosphite. This method gives results about 1/2 per cent. higher than are obtained by precipitating in the presence of ferric chloride. It also requires less time and a small quantity of solution is used.

Sulphuric Acid

The present importance and enormous possibilities of the manufacture of sulphuric acid as a by-product in treating the ores of various metals, together with the fact that the sulphur from these sources is not at present credited to either the pyrite or sulphur production, are the reasons requiring the treatment of this chemical in the Mineral Industry.

Sulphuric acid is the back-bone of the chemical industries and there is justification for Humboldt's oft-quoted dictum that the best measure of the degree of civilization of any country is the extent of its use of sulphuric acid. It is principally used in the manufacture of the soluble phosphate fertilizers. It is also required for making high explosives, for refining oil, in making steel, and textile products, and in a large proportion of chemical manufacturing processes.

According to recently issued reports of the Thirteenth Census the total production of sulphuric acid in the United States in 1909 reduced to 50° Baumé was 2,797,723 tons, an increase of 50 per cent. over 1904. Of this total 1,242,074 tons or 44 per cent. was consumed in the establishments where manufactured; 933,848 tons were made as a chief product and 621,801 tons as a secondary product.

In 1911 the United States consumed 585,561 long tons of sulphur from pyrite mined or imported as such, and equivalent to about 2,850,000 tons of 50° B. acid. In addition there was produced in Tennessee 275,000 tons of 60° B. as a by-product from copper ores. The zinc smelters of the country produced 252,142 tons of 60° B. acid as a by-product, and the western production from gold and copper ores was probably half as much.

The total production of the United States in 1911 including both the acid marketed and that consumed where manufactured was thus approximately 3,500,000 tons reduced to 50° Baumé. This is exclusive of brimstone acid.

California.—Owing to the curtailment of the copper smelting industry in California due to smoke litigation, the 1911 production of acid was less than for previous years. Heretofore the regularly published reports of pyrite production for the state were at best approximations, owing to the inclusion of by-product pyrite. The smelters themselves kept no separate record of pyrite commercially used, and the pyrite equivalent of the ore directly used in by-product recovery was only roughly estimated. Thus for 1910 the U. S. G. S. Mineral Resources reported pyrite produced 23,700 tons valued at $110,134. While, for the same year the State Mining Bureau under its former management reported pyrite produced as 42,621 tons valued at $179,862.

Tennessee.—The Tennessee Copper Company produced about 150,000 tons of sulphuric acid during the year 1911. The acid as produced ran slightly over 60° B., probably averaging 61° B., but the above figures are on the basis of 60° acid.

There were no important additions to the acid plant during the year 1911, except the completion of two very large Gay Lussac towers which were, however, begun in 1910. These towers have effected a very large saving in the niter consumption. In June a serious fire occurred which damaged the Glover towers and shut the plant down for about 6 weeks, otherwise the production would have been much larger. The niter-charging house was also found to be in such bad condition that it was entirely rebuilt and in doing so the method of charging niter and sulphuric acid was radically improved so that much better results are now obtained than previous to the fire. Radical improvements have also been effected in the method of charging the blast-furnaces, the most notable being the installation of a bedding plant, and these improvements have resulted in a gas with a much more uniform percentage of SO_2 than had hitherto been obtained.

There are five other important acid producers in the state: The Ducktown Sulphur, Copper, and Iron Co., the Tennessee Chemical Co., the Federal Chemical Co., the Read Phosphate Co., and the Virginia-Carolina Chemical Co. The total production of acid within the state was about 275,000 tons at 60° Baumé.

Sulphuric Acid from Zinc Ores.—The total production of sulphuric acid of 60° B. as a by-product of zinc ores was 252,142 tons as against 202,000 tons in 1910.

Technology

Draft in Sulphur Burning.—Among the advantages claimed for fan draft in the chamber process are: greater production per cubic foot of chamber, even draft on the sulphur burners, reduced cost of niter per ton of acid, the concentration of a greater quantity of chamber stock to acid of 60° B, in the Glover tower, better absorption of niter in the Gay-Lussac towers because of the proper excess of oxygen in the chamber process, and finally, the successful use of intermediate towers which can rarely be done with the natural draft.

Cooling Burning Gases.—A recent French patent (431,067) provides for the cooling of pyrites-burner gases by the use of a sectional lead tower with an annular water jacket. The tower is sealed in a trough of sulphuric acid and the gases are drawn off from the bottom. It is claimed that the burner gases can be cooled from 500° C. to the ordinary temperature and all the sulphuric acid fog they contain separated as liquid.

Acid from Lead Ore.—At the smelter Binsfeldhammer, at Stollberg, Rheinland, lead ore, the greater part of which is Australian concentrates, after pre-roasting is treated by the Dwight-Lloyd process on an apparatus which is described in the German Patent No. 225,421, which separates the gas into two portions, one containing 3 to 5 per cent. SO_2, 54 grams H_2O and 1 to 2 grm. dust per cubic meter, and the other proportion containing from .6 to .9 per cent. SO_2. The rich portion is cleaned in a coke filter and dried in a sulphuric acid tower, and is then made into sulphuric acid by the Grielo-Schroeder contact process. In this manner about 50 per cent. of the sulphur contained in the ore is recovered as sulphuric acid. The present plant produces about 13–14 tons of 100 per cent. H_2SO_4 per day.

Sulphuric Acid from Gypsum.—A new process, involving the preparation of sulphuric acid from gypsum has recently been described by M. Jules Auzies in *Le Phosphate.*

If calcium sulphate or gypsum as nearly dehydrated as is possible ($CaSO_4$), be mixed in suitable proportions with iron oxide, as for example, the oxide Fe_2O_3, and the mixture be calcined, it is found that the calcium sulphate in contact with the iron oxide yields its sulphuric acid radicle to the latter, and gives rise to ferric sulphate $Fe_2(SO_4)_3$.

The iron sulphate may then be heated to suitable temperature, when it breaks down into $Fe_2O_3+SO_2+O$. This reaction permits the recovery of the initial iron oxide, which may be thus used over and over again. The temperature required for this dissociation is between 800 and 1500° C.

The two reactions described above may be simultaneously operated; they are respectively represented by the following equations:

$3CaSO_4+Fe_2O_3=Fe_2(SO_4)_3+3CaO$; $Fe_2(SO_4)_3=SO_2+3O+Fe_2O_3$.

On a commercial scale this process may be applied by using ordinary gypsum, as dehydrated as is possible, together with the ferric oxide, in the proportions of 408 parts by weight of gypsum to 148 parts of iron oxide. This represents the ideal mixture, but in practice only 1 part of oxide to 4 of gypsum is required. The mixture should be continually stirred during the process.

To produce the sulphuric acid anhydride, the mixture of heated SO_2 and O gases is passed over a catalyzing oxide. For this purpose there may be used either of the manganese oxides MnO_2 or Mn_2O_3, or the tungsten oxides WO_2 or W_2O_3, or the molybdic anhydride MoO_3, or even thorium oxide ThO_2. The catalytic agent should be heated originally to a temperature of 200° C. to facilitate the union of the oxygen and sulphurous oxide. Subsequently, the heat generated by this union will be sufficient for the continued facilitation of the process. The action

of ultra-violet rays also favors the formation of the sulphuric acid anhydride.

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TALC AND SOAPSTONE

By Richard H. Vail

The talc industry of the United States declined slightly in 1911, the total production amounting to about 143,500 tons. Nearly half of this production was from New York, with Vermont second and Virginia a close third with 26,759 tons. The output from Virginia, however, is almost entirely soapstone slabs, the value exceeding that of the New York production.

STATISTICS OF TALC AND SOAPSTONE IN THE UNITED STATES. (*a*)
(In tons of 2000 lb.)

Year	Production.						Imports.		
	Fibrous Talc.			Talc and Soapstone (*b*).					
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Value Per Ton.
1897...	52,836	$283,685	$5.37	27,068	$259,948	$9.60	779	$8,423	$10.54
1898...	54,807	285,759	5.21	27,974	237,280	8.48	445	5,526	10.70
1899...	57,120	272,595	4.77	26,682	241,267	9.04	254	3,534	13.91
1900...	45,000	236,250	5.25	26,726	249,777	9.35	79	1,070	13.50
1901...	69,200	483,600	6.99	28,643	424,888	14.83	2,386	27,015	11.32
1902...	71,100	615,350	8.65	26,854	525,157	19.36	2,859	35,336	12.36
1903...	60,230	421,600	7.00	26,671	418,460	15.69	1,791	19,677	10.99
1904...	65,000	455,000	7.00	27,184	433,331	15.94	3,268	36,370	11.13
1905...	67,000	519,250	7.75	40,134	637,062	15.87	4,000	48,225	12.06
1906...	64,200	541,600	8.43	58,972	874,356	14.82	5,643	67,818	12.02
1907...	59,000	501,500	8.50	72,010	905,047	12.57	10,060	126,391	12.56
1908...	70,739	697,390	9.86	46,615	703,832	15.01	7,429	97,096	13.07
1909...	48,536	359,957	7.42	81,802	862,002	10.53	4,417	56,287	12.74
1910...	71,710	728,180	10.15	79,006	864,213	10.94	8,378	106,460	12.71
1911...	62,500(*c*)	562,500	9.00	81,521	1,032,732	12.66	7,113	88,050	12.38

(*a*) Statistics for 1902 and subsequent years are as reported by the United States Geological Survey, except that fibrous talc is as reported by the New York State Geological Survey. (*b*) The value of these products has not much significance owing to the diverse conditions of the material reported. (*c*) Estimated.

About 7000 tons of high-grade ground talc were imported, mainly from Italy and France; small shipments were also received from other countries, particularly Canada and Austria; some crude talc was also imported. The bulk of the world's production is supplied by the United States, France, Italy and Austria.

Most of the Appalachian States produce both soapstone and the ordinary foliated talc, the deposits of which, as a rule, are not extensive and their economic importance depends primarily on their proximity to transportation and market.

The conditions surrounding the production of fibrous talc in New York are exceptional. The superior quality of this fibrous talc for

book and writing paper gives the Gouverneur district a practical monopoly in this field. Indeed some of this talc is exported to Europe, chiefly to Germany, and used in the manufacture of writing paper.

Deposits of talc are not rare, and if of only ordinary quality, the mineral is seldom profitably shipped as mined. Most of the talc is owned or controlled by manufacturers or users, and the mineral therefore is usually in the market in manufactured form. There is a good demand for the best grade of talc, as is indicated by the importation from Italy and France, notwithstanding the import duty of at least 20 per cent. *ad valorem* now prevailing on the manufactured or ground talc. The freight rate from Italy has also increased.

Talc is marketed under three general heads: (1) The ground fibrous talc used in paper-making, this being obtained almost exclusively from the deposits in St. Lawrence County, New York; (2) the massive variety known as steatite or soapstone, the supply of which comes mainly from Virginia and Vermont; (3) ordinary ground talc of various grades, produced in many of the Atlantic States, especially in Vermont, Massachusetts, Pennsylvania and North Carolina.

The prices for ground talc varied from $7 to $20 per ton for American talc, $18 to $30 for French talc and from $30 to $40 for Italian. The ground talc enters a much diversified market according as the various grades meet the qualifications required by the different industries. The price for ground fibrous talc for paper-making was about $9 per ton, f.o.b., Gouverneur, N. Y. The producers of soapstone and block products enjoyed a better market but the products are so diversified that it is impracticable to give the range of prices. The average value of the crude talc as sold in the United States was about $4 per ton; when sawed into slabs it was $20 per ton; manufactured articles varied from $24 to $40, averaging about $28 per ton.

Talc and Soapstone in the United States

Arkansas.—There is a large steatite deposit in this State, near Benton, in Saline County, but attempts to raise capital for its operation have been ineffectual. The soapstone of these deposits is reported to be suited to the manufacture of refractory products but a railroad connection is required; a branch road 9 1/2 miles long would connect with the Rock Island Railroad, or connection might be made with the St. Louis, Iron Mountain & Southern Ry. at Bryant.

California.—The only producer in 1911 was the California Talc & Soapstone Co. in Tulare County, about 8 miles east of Lindsay.

Georgia.—Talc deposits were operated in Murray County, the princi-

al producing companies being the Georgia Talc Co. and the Cohutta alc Co. with mills at Chatsworth. Talc of a grade suitable for crayons nd other block products is obtained, and lower grade material, amount- ıg to about half the production is ground. Talc properties not yet n a producing basis are owned by the Piedmont Talc Co. and the ort Mountain Talc Co. in the schist belt a few miles southeast of hatsworth.

Maryland.—A small production was made in this State by the)eland Mining & Milling Co. from the Bald Friar quarry in Cecil County.)ther deposits from which no production was reported are those of the Iaryland Soapstone Co. of Baltimore which succeeded to the Steatite orporation's mill and quarry near Marriottsville; a deposit near onowingo is owned by the Cecil Mineral Co. There are also deposits in arroll County.

Massachusetts.—Talc production increased slightly in 1911, the roducers being the Massachusetts Talc Co., the Foliated Talc Company nd the Northampton Talc Co. The former continues to be the ading producer and during the year rebuilt its burned mill at Zoar. he mines are 4 miles to the north in the town of Rowe, near the mine nd mill of the Foliated Talc Company.

New York (By D. H. Newland).—The talc mines in the Gouverneur listrict contributed nearly their usual output in 1911, or between 60,000 nd 65,000 tons, all of which was shipped in ground state, principally to he paper trade. The average price received for the product was about 8.50 per ton, the paper grades selling generally at $9, and the other rades of inferior quality at less. Production during the last three easons has been handicapped by the long drouths that curtailed the vater supply; the mills are run by water power supplied by the Oswe- atchie River. This condition will probably be remedied by the use of lectric power from one of the local commercial plants, as a supplement- ary supply.

The International Pulp Co. has recently converted its No. 4 mill at Hailesboro into a power station. This company has also taken steps oward concentrating its operations, now distributed among the various nines and mills acquired. Its supply of rock is now taken mainly from he two mines at Talcville and from a new mine that has been recently pened in the southwestern part of the district. The latter is situated n the vicinity of the old Wight mine formerly worked by the Union Talc o., and is reported to furnish a very good grade of talc. Of the milling plants owned by the company, No. 3 and No. 6 and the Columbian mills ıave been most active and are apparently to be the main dependence in he future operations.

The Uniform Fibrous Talc Co. began active production in 1911, having recently opened a mine and erected a mill near Talcville. The talc has been found to be of excellent grade, although the outcrop was not of very promising character. It shows a width of over 30 ft. in the stopes, and the talc is the usual mixture of fibrous and foliated varieties common to the district. The company has its own hydro-electric power plant situated on the west branch of the Oswegatchie about a mile from the mine. An interesting feature of the milling process is the adoption of the Hardinge conical mill for preparing the talc for its final treatment in the dump cylinders. This takes the place of the Sturtevant mill, buhrstones, or the large pebble mills employed in the other plants.

The Ontario Talc Co. continued operations in the Potter mine where it has developed a large body of high-grade rock, sufficient to maintain the mill output for years to come. The milling plant has also been improved, and the company expects to take a larger share in the industry than in the past.

An occurrence of talc at Natural Bridge, Lewis County, was mentioned in the review for the preceding volume of THE MINERAL INDUSTRY. A mill has been erected on the property by the St. Lawrence Talc and Asbestos Company. The talc appears to be mostly either a foliated or massive variety, and is associated with crystalline limestone and serpentine. The product is ground by buhrstones and finished in a large-size pebble mill.

North Carolina.—The output from this State decreased nearly 10 per cent. in 1911. About one-half of the production marketed as talc is pyrophyllite, a hydrous silicate of alumina closely resembling talc. Pyrophyllite is mined especially in Moore County, while true talc deposits, some of them of excellent quality, were operated in Swain, Alleghany, Graham and Jackson Counties. The product of the Hewitt mine in Swain County has an international reputation. Pyrophyllite properties are owned near Glendon in Moore County by the Glendon Mining & Manufacturing Co., the Croatan Mining Co. and the American Talc Company.

Pennsylvania and New Jersey.—Deposits of talc occur on both sides of the Delaware River in the vicinity of Phillipsburg, Warren County, N. J., and near Easton, Penn. John O. Wagener & Co. and C. K. Williams & Co. operate quarries and mills on the Pennsylvania side of the river, and the Lizzie Clay & Pulp Co. operated on the New Jersey side, grinding most of its product. At Easton the deposit is in part serpentine which is locally designated as verdolite and is ground into mineral pulp for various uses. Some of the rock takes a good polish and is used in the place of marble for wainscoting.

Vermont.—In 1911 the output reached a total of over 29,000 tons, an increase of about 14 per cent. Most of the production was ground; a small portion was manufactured into laundry tubs, etc., and some was sold as it came from the mine. The outcrops of talc are numerous and vary greatly in quality. Some of the deposits are exceedingly poor but the product from some properties is injured by the presence of iron and silica. Few of the quarries, however, furnish sound slabs of such size as are needed for laundry tubs, etc. Part of the waste from the soapstone factories is ground for sale as powdered talc. Among the oldest companies now operating are the Union Soapstone Co. at Chester Depot and the Vermont Soapstone Co. of Perkinsville. The American Soapstone Finish Co. is also at Chester Depot; among other producers are the American Mineral Co. of Johnson, the Eastern Talc Co. of East Granville, Rochester and Stockbridge. The mill at Windham is now operated by the Massachusetts Talc Co. of North Adams, Massachusetts.

Virginia.—The principal supply of soapstone in the United States comes from Albemarle and Nelson Counties. There was no production from Amelia or Campbell Counties. A new producer of soapstone, the Utica Mining & Milling Co. of Rhoadesville, Orange County, appears in the 1911 list. The Virginia Soapstone Co., with its affiliated marketing company the Albarene Stone Co., continues to be the dominant factor in the soapstone market. The Old Dominion, the Climax, the Phoenix and the Piedmont, all in the main soapstone belt, were producers in 1911. There was a small production of ground talc from Fairfax County in the northern part of the State.

Talc in Foreign Countries

The chief sources of talc in Europe are in the French Pyrenees and in the Italian Alps. Talc production has been increasing in Austria, though that country exported only a small amount to the United States in 1911. The best grades of talc come from Italy but some of the French and Austrian properties produce talc of almost equally high grade. Talc of excellent quality is also obtained from Syria, though the operations are not extensive. Canada is shipping talc to the United States, most of it in the crude form. Though not of international importance, limited talc operations are conducted in Brazil, China, Japan, India, Australia, New Zealand and indeed in almost every country. Talc is imported into the United States in the form of rough rock, powdered talc, soapstone slabs for use in plumbing work and blocks for making gas tips and other small articles of manufacture.

Austria.—The talc production in 1911 amounted to 13,800 metric

tons, largely from the province of Styria. Much of the Austrian production is used in Germany in paper making. Only limited shipments were made to the United States in 1911. The leading producers were: Federweiss-Interessentenschaft der Commune Mautern, Mautern, Styria, 4000 tons; Arthur Bittner, of Vienna, 2000; Brüder Wiedenhofer, Anger, Styria, 1200; Aflenzer Grafit-und Talksteingewerkschaft, Aflenz, Styria, 600; Graf Karl Lamberg, Pöllau, Styria, 1000; Eduard Elbogen, of Vienna, in the works at St. Kathrein and Floing, 300; Fritz & Co., Anger Styria, 1000; Bischetsrieder & Gielow, Salzburg, 1200; Carl Marx, Rohoncz, Hungaria, 1000; Kärntner Talksteinwerke A. Conte Veith, Grüsserhof, 1500 tons. Steps were taken toward an organization of the talc interests under the title of "Verband der Talkum-Interessenten in Oesterreich-Ungarn" of which it is expected that Graf Karl Lamberg, of Pöllau, Styria, will be the first president.

Brazil.—The inaccessibility of the Brazilian talc deposits has limited the production from the beds near the cities of São Paulo and Loreno in the State of São Paulo. The talc from these deposits is usually marketed in three grades, according to color.

Canada.—In the vicinity of Madoc, in the Province of Ontario, the Henderson talc mine has been operated for a number of years. A few years since, a mill was erected and a portion of the output is now ground. In 1911, another talc vein was opened on the farm of Charles Conley, near Madoc, on what was reported to be an extension of the Henderson vein. Talc deposits also occur in the Province of Quebec, particularly in the vicinity of the Black Lake asbestos district. The talc production of Canada in 1911 amounted to 7300 tons, valued at the mine at about $3 per ton.

France.—The talc production of France now amounts to about 40,000 metric tons annually, which is valued at the mine at slightly under $7 per ton. The principal deposits are in the Pyrenees in the department of Ariège. Talc is also mined in the departments of Aude, Haute Loire and Pyrenees Orientales. Some of the talc is of excellent quality, nearly equaling that of the best Italian grades, though much inferior talc is also shipped. The important deposits at Luzech are of excellent quality, being of a whitish or bluish-white color much appreciated by consumers in certain trades.

Italy.—In 1911, the talc production of Italy increased to 15,620 tons, the crude product at the mines being valued at $7.13 per metric ton. This was an increase of nearly 25 per cent. over the production of 1910. Talc is mined in the northern part of Italy, especially in the vicinity of Pinerolo in the Italian Alps, and is usually shipped from Genoa. It has long been mined in the valley of Chisone and also in the high valleys

of Susa and Lanzo. The premier grades of talc in the world are mined in this country, the Italian talcs bringing the highest prices and finding their main use in toilet and medicinal preparations on account of their superior color and absence of grit. Inferior talcs are of course produced, but the greater portion of the Italian talc imported into the United States is of the higher grade. The talc is all obtained by underground mining in which 332 workmen were employed in 1911; nearly the entire output was ground in 11 mills employing 87 workmen. The finished product is valued at the mills at about $12 per metric ton.

Uses of Talc

In the form of soapstone slabs, talc finds a wide use in hospitals, chemical laboratories, acid tanks and plumbing work, as it is unaffected by most acids. It resists heat well and is used for griddles, hearthstones and gas tips; also for marking pencils such as tailor's chalk. It has great dielectric strength, requiring 30,000 to 40,000 volts to pierce a 1/2-in. slab, and is used for the flooring of electric stations, switchboard panels and other electrical uses.

Powdered talc is used in the manufacture of paper, toilet powders, foundry facings, sizing for cotton cloths, insulated covering for electric wires, dressing for skins and leathers, facing for rubber molds, and to some extent in inferior grades of soap and paint, particularly enameled paints. The "French chalk" of commerce is mainly powdered talc. Aside from its legitimate uses, talc has been used as an adulterant in numerous trades and this fact in part accounts for the attitude of secrecy prevailing among many producers and also explains the reticence of jobbers of talc products in giving information or statistics concerning its uses.

The value of powdered talc depends on physical qualities, and is judged by its "feel" or slip. The whitest talcs, or those with a bluish-white shade, command the highest price. The poorer grades have a yellowish hue, a portion of the French output having this disadvantage. The Italian talc is the whitest and best for such purposes as the talcum toilet powders which have come into extensive use in recent years. Some of the best grades of domestic talc are also used in these powders. Talc is used in the sizing of cotton cloth, regardless of color, but freedom from grit, which dulls the cutting knives, is essential in this trade.

The fibrous talc mined in St. Lawrence County, New York, is especially valuable as a filler in the manufacture of book and writing paper, as it increases the strength of the paper and eliminates or reduces the brittleness characteristic of paper weighted with clay or inferior grades of

talc. Although apparently granular when ground to 100 mesh, the fibrous nature of the talc is still maintained and is readily observable under the microscope. The fibrous talc is superior to the ordinary foliated talc, in that a larger proportion is retained in the paper pulp. This is particularly true as compared with the china clay formerly used, of which only about 30 to 35 per cent. was retained in the pulp as against 75 to 90 per cent. when the fibrous talc is used. The talc is prepared by crushing in the usual rock breakers, the product from which is passed to rolls and then further reduced by buhrstones, tube mills or Hardinge conical mills. No artificial drying is necessary and most of the product in this district is not bolted, as such fineness is not required in the paper-making industry. In the mills in other States, the talc is usually bolted to insure the fineness desired by most industries.

TIN

The tin-mining industry of the United States continued during 1911 to be insignificant. Little or nothing was done in connection with the deposits in the Black Hills. The Pahasa Mining Co. made no developments on its property during the year, and at present the stockholders have not yet decided what the future course of the company will be. In Texas, at the mines on Mt. Franklin, near El Paso, the mill and smelting furnace were operated intermittently from January to May, and about four tons of pig tin, 99.8 per cent. pure, were produced and shipped to market. The intermittent character of the operations was due to bad weather and other causes. The mine is now temporarily closed pending new arrangements. The deposits and the smelter were described in the *Engineering and Mining Journal*, Jan. 21, 1911, and in Vol. XIX of THE MINERAL INDUSTRY. The engineers in charge of this property are optimistic as to the supply of tin existing there and their ability to treat the ore. The smelting furnace of the company was fired with crude oil and excellent results in smelting were obtained from the use of that fuel.

The experience in smelting with oil obtained at El Paso may be useful in the treatment of black tin in Alaska, from which a shipment of 92 tons of concentrate was made in October by the York Dredging Co. This company installed a dredge on Buck Creek, 14 miles north of York. Operations were begun on Sept. 10 and continued until Oct. 15. Its capacity is about 1000 cu. yd. per 24 hours. A recovery of 6 to 7 lb. of black tin per cubic yard is reported.

Imports of tin into the United States during 1911 amounted to 53,468 short tons as compared with 52,569 tons in 1910. The 1911 imports comprised 28,118 tons from Great Britain, 2448 tons from Germany, 18,590 tons from Straits Settlements, and 4313 tons from other countries.

The receipts from Great Britain were largely of Straits tin, which goes to that country in transit. Those from Germany were largely of Bolivian tin, which is smelted and refined in that country.

Exports during 1911 included 1132 short tons of tin metal (all of foreign origin); manufactures of tin, valued at $1,143,406; and tin plates, terneplates and taggers tin to the gross weight of 68,747 short tons, valued at $4,776,256.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1902..	85,043,353	$21,263,337	1906.	101,027,188	$37,446,508	1910.	105,137,740	$33,921,449
1903..	83,133,847	22,265,367	1907.	82,548,838	32,075,091	1911.	106,936,872	43,390,639
1904..	83,168,657	22,356,896	1908.	82,503,190	23,932,560	.....	.....	.....
1905..	89,227,698	26,316,023	1909.	95,350,020	27,559,937	.....	.....	.....

Alaska.—The tin placers of Buck Creek, 14 miles north of York, on Seward peninsula, have been worked in a small way for nearly 10 years. In 1911, these operations gave way to dredging. A gasolene-power dredge with 2 1/2-ft. buckets was installed by the York Dredging Co. of San Francisco, about the middle of September and operated for 34 days, or until the close of the season. The total recovery was 92 tons of cassiterite, averaging 64 per cent. tin, which was at the rate of 6 to 7 lb. cassiterite per cubic yard of gravel. Details of the construction and operation of this dredge are in the *Engineering and Mining Journal* of July 15 and Dec. 23, 1911. It is reported that new drifting on the Lost River and Cape Mountain cassiterite-bearing lodes met with encouraging results. The United States-Alaskan Tin Mining Co., of Milwaukee, has accomplished a total of 864 ft. of tunnelling and drifting on its claims on Cape Mountain.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (a)
(In tons of 2240 lb.)

	1902	1903	1904	1905	1906	1907	1908	1909	1910	1911
English production	4,392	4,282	4,132	4,468	4,522	4,407	5,052	5,198	(b) 5,810	4,300
Chinese production (Yunnan)	3,788	2,443	2,979	4,463	3,948	3,480	4,558	4,445	(b) 4,500	2,600
Straits to Europe and America	51,831	52,212	57,419	56,840	57,143	53,520	60,491	58,541	54,625	50,270
Straits to India and China	1,882	3,123	3,261	1,484	1,292	2,178	2,187	2,030	(b) 2,100	3,400
Australia to Europe and America	3,199	4,934	4,846	5,028	6,482	6,612	5,748	5,384	4,563	3,825
Banka sales in Holland	14,978	15,070	11,363	9,960	9,286	11,264	11,530	11,973	(b) 12,000	12,581
Billiton sales in Java and Holland	3,897	3,650	3,215	2,715	1,968	2,229	2,235	2,241	(b) 2,250	2,650
Bolivian arrivals in Europe	10,150	9,630	12,978	14,245	16,394	15,594	17,032	18,121	18,225	22,064
South Africa	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,200
Totals in long tons	94,117	95,344	100,193	99,203	101,035	98,284	108,833	107,913	104,073	104,090
Totals in metric tons	95,628	96,874	101,802	100,795	102,657	99,861	110,580	109,637	105,764	105,753

(a) Compiled from commercial reports. There is also a small production in Germany. (b) Estimated.

Tin in Foreign Countries

Africa.—Tin ore is produced on a growing scale in nearly every section of southern Africa; in fact, development of the tin industry in these localities has called forth a good deal of misplaced and exaggerated interest. Total imports of tin-ore concentrates into Great Britain from all the South African localities amounted to 4795 long tons in 1911, compared with 4308 tons in 1910 and 2858 tons in 1909. These figures

account for practically the entire output, except for some shipments to Straits Settlements by two tin-mining companies in the Transvaal.

Congo Free State.—Cassiterite is found in the Bussanga district of Katanga, in a belt extending for 100 miles and lying parallel and just north of the copper belt. At one locality, an area of 1,125,000 sq. yd. is said to have been proved.

Nigeria.—Exploitation of the tinfields in Bauchi has already attained considerable proportions, and about $12,500,000 has been invested in the various enterprises. The stanniferous areas are undoubtedly large and valuable, but in several cases company promoters and others have expressed hopes and forecasts which can never be fulfilled. As far as is known at present, tin is to be found in more or less profitable quantities over 9000 sq. miles of territory, extending from Ningo in the extreme east to Duchin Wei, situated about 40 miles east of Zaria, on the extreme west; and from Liruen-Kano in the north to the Ninkada and Mada districts in the south. Mining leases have been granted over about 12,000 acres and exclusive prospecting licenses over about 355 sq. miles. Prospecting rights have been granted to 149 prospectors. During the year 1910 11,800 tons of cassiterite was exported. Opinions vary as to the probable future prospects of the tinfield. It is safe to say, however, that the existence of the tin over a large area has been proved, but that up to the present time sufficient prospecting has not been effected to render it possible to draw any conclusion regarding the value of the field as a whole.

The question of transport has been the great trouble. From the Niger the Baro-Kano main line is being extended across northern Nigeria. This was planned before the discovery of tin ore and was part of the general colonial development of the territory. With the finding of tin and the necessity for quicker transport, a special branch line was started from Zaria on the main line to run for about 100 miles to the west and to terminate at the foot of the Bauchi plateau at Warno.

Rhodesia.—Excitement occurred in 1911 over several tin discoveries in widely separated districts of Rhodesia. The first discovery was made in 1909 on the Uniabi River, where tin oxide was found to occur in small quantities in chloritic schist near a granite contact. This was not payable. The next discovery was made northeast of Salisbury in the gold-mining district of Enterprise, where cassiterite occurs in pegmatite dikes more or less altered to greisen. Farther north, at Abercorn, tin occurs in greisen and also in pegmatite. These tin occurrences are similar to several in the United States, especially in South Dakota. So far, exploration has shown that the Enterprise tin discoveries are scarcely likely to be important, as the tin ore appears to be irregular in its distribution and denudation has not been sufficient to furnish alluvial deposits of any

importance. In the Abercorn district, prospects are more favorable for the opening of payable dikes and some alluvial tin. Recently promising discoveries have been made in the more eastern portions of Rhodesia in the Victoria district. Large masses of crystals of cassiterite up to 30 lb. in weight have been found as float, while tin ore in what appears to be payable proportions occurs in several of the outcrops, and the prospects of procuring a large tonnage from these dikes, which are 10 ft. or more in width, appear to be promising.

Swazieland.—The alluvial tin ore produced during 1911 in Swazieland amounted to 476 tons and was valued at £42,250. According to official returns, Swazieland is credited with exports of 1119 long tons of tin ore to Great Britain in 1911; some of it may have originated in other sections from which no record of production is kept.

Transvaal.—Tin production was of record dimensions during 1911 the total value being estimated at £410,000. Previous production has been by fiscal years, ending June 30, 1906-7, £27,180; 1907–8, £81,700; 1908–9, £148,340; 1909–10, £303,210. For the half year to Dec. 31, 1910, the value was £147,530.

Imports of Transvaal tin ore into Great Britain amounted to 2199 long tons in 1911, compared with 2936 tons in 1910, the decrease in imports in the face of increasing production being accounted for probably by exports to Straits Settlement by the Rooiberg Minerals Development Co. and the Zaaiplaats Tin Mining Co. In the case of both of these mines, cassiterite occurs in rock, and is extracted in stamp mills. In March, 1912, the Zaaiplaats Company mined and treated 2679 long tons of ore at a total cost of £2900, for a yield of 138 tons of concentrate carrying 68 per cent. tin. The Rooiberg mine, during the same month, treated 1573 tons, recovering 61 tons of concentrate assaying 68 per cent. tin, at a profit of £3015.

The establishment of a tin smelting works in the Transvaal has been discussed and an experimental furnace run was made, but the product was high in iron. The average grade of concentrates from this district is 70 per cent. tin; whence the saving on freight to Europe would not probably equal the excess cost of local smelting as compared with that conducted at the established English works. Transvaal tin is not quite up to "standard" grade, and is, therefore, more suitable for alloys than for tin plate.

Australasia.—In Queensland the production for 1911 was 3091 long tons of tin oxide valued at £307,847 which is 138 tons and £64,576 in excess of 1910. In the Herberton district exploitation of the various deposits has resulted in additions to the supplies of ores, and a corresponding stimulus to operations. The most important developments

occurred at the Vulcan mine at Irvinebank. The group of mines at Koorboora contributed an important output while lode-tin in quantity was also obtained from Herberton and Watsonville. The yield of alluvial tin was won principally from the Herberton, Cooktown and Stanthorpe districts. In the latter district, three dredges recovered tin ore valued at £9130 from a total of 279, 300 cu. yd. of gravel.

The output of New South Wales in 1911 included 958 long tons of tin metal, valued at £191,000, and 971 tons of tin ore worth £116,089. This was an increase of 111 tons in metal, but a decrease of 50 tons in ore shipped, as compared with 1910. Of the output of tin ore, 1742 tons, or 68 per cent., came from dredges, operating mainly at Tingha and Emmaville. Four bucket dredges and 28 pump dredges were at work on tin gravel. The bucket dredges handled 577,977 cu. yd. of material, recovering therefrom 147 long tons of tin ore valued at £17,565; this was at the rate of 0.57 lb. of cassiterite, worth 7.29d., per cubic yard. The pump dredges treated 2,533,782 cu. yd. of gravel, recovering 1418 long tons of tin ore valued at £167,284; or at the rate of 1.25 lb. cassiterite, worth 15.84d., per cubic yard.

Tasmania yielded 3953 long tons of tin ore, valued at £513,500, in 1911, compared with 3701 tons (£399,393) in 1910. At the end of 1911, operations were being conducted by 43 concerns, in addition to some of such small scale as not to be separately listed. The industry employs 1695 men. The largest producers are the Mt. Bischoff, the Pioneer and the Briseis companies. The Anchor mine in the Blue Tier Range, is often referred to as an example of the low cost of mining by the opencut system where the conditions are favorable. During the year 1910–11, 102,944 tons of ore were mined and milled at a cost of 2s. 7 1/2d. per ton. The recovery was 160 1/2 tons of tin oxide (cassiterite), or 3.46 lb. of tin oxide, equal to 2.38 lb. of metallic tin, per ton of ore treated.

Details of mining and milling operations in the Tasmania fields were published in the *Engineering and Mining Journal*, April 8, 1911.

In Western Australia, a small production of tin ore (estimated at 420 tons in 1911) is maintained by several widely separated fields: Kimberly, Pilbara, Thomas River, Gascoyne, Murchison, Coolgardie and Greenbushes. The latter field has yielded about 8000 tons to date. Here the granitic country is intersected by dykes of acid rocks with which cassiterite is associated as stringers. The present source of tin ore is in the adjoining alluvial deposits, some of which are 90 ft. deep. The Pilbara gold field has yielded about 5000 tons of tin ore to date, taken from alluvial deposits. The only district in which lode mining is profitable is Wodgina, on the Turner River, 74 miles from Port Hedland. This belt is 80 miles long and 30 miles wide, and its prospects are considered encouraging.

Bolivia.—The tin output of Bolivia is now second only to that of the Malay Peninsula. The output of 1911, based on the first half-year's returns, consisted of 38,100 metric tons of cassiterite, the metallic contents of which was roughly 22,860 tons of tin. Small additional shipments of locally smelted metal are made every year, going almost entirely to Great Britian. Exports of ore, which have to be shipped via Mollendo, Peru, or Arica and Antofagasta, Chile, are very evenly divided between Great Britain and Germany. In 1911, England imported 16,372 metric tons of Bolivian ore; estimating imports into Germany at 17,000 metric tons in 1911 (15,925 tons in 1910 and 13,099 tons in 1909), gives a total of 33,372 tons of ore going from Bolivia to those two countries in 1911.

EXPORTATION OF TIN FROM BOLIVIA.
(In metric tons.)

Year.	*Barrilla.* Tons	Tin Contents. (*a*)	Year.	*Barrilla.* Tons.	Tin Contents. (*a*)
1900	15,088	9,053	1906	29,370	17,624
1901	21,573	12,943	1907	27,678	16,607
1902	17,340	10,404	1908	29,938	17,963
1903	21,785	13,071	1909	35,566	21,340
1904	20,369	12,221	1910	38,548	23,129
1905	27,690	16,614	1911	38,100	22,860

(*a*) Tin content of the barrilla (black tin concentrate), computing the latter at 60 per cent. metallic tin.

According to Miltiades Armas (*Eng. and Min. Jour.*, Aug. 26, 1911) the average grade of all ores, as mined, in Bolivia is 3 per cent. metallic tin, and only in rare cases reaches 4 per cent. The principal mines have mills equipped with modern machines, oil or producer-gas being the usual motive power. Stamps are the usual crushing appliance, followed by jigs, Huntington mills, classifiers, shaking tables and buddles. It is usual to make concentrates carrying 65 per cent. tin, unless the ores contain bismuth, in which case the concentrates, carrying only 50 per cent. tin, are shipped to Havre for further extraction of bismuth. Operating expenses at two typical mines are as follows, in U. S. currency per ton of ore treated.

(1) Hydraulic motive power. Mining, and conveying to mill, $4.78; sorting and handling, $0.48; milling and assaying, $1.51; general expenses, $0.36; total production, $7.13. Freight per ton of concentrate from mine to Guaqui, $35.88; Guaqui to Mollendo, $10.14; Mollendo to Europe, $7.80; export duties, $15.60; commissions, insurance, etc. $10.92; total

shipping charges, $80.34 per ton of concentrate (ratio of concentration: 20 into 1).

(2) Fuel is kerosene. Mining, $2.73; milling, $1.89; general, $0.90; administration, $0.78; total production, $6.30 per ton of ore. Freight duties, insurance and commissions, from this mine to Europe, amount to $70.20 per ton of concentrate.

RECEIPTS OF BOLIVIAN TIN AT LIVERPOOL, HAVRE, AND HAMBURG. (a)
(In tons of 2240 lb.)

Year.	Liverpool.				Havre. Tons Fine.	Hamburg. Tons. Fine.
	Bars.	Ore.		Total Tin.		
		Crude Weight.	Metallic Content.			
1900	1,507	5,431	3,530	5,037	.....	.....
1901	1,730	9,086	5,905	7,635	.....	.....
1902	1,685	10,961	6,576	8,261	.....	.....
1903	1,614	10,401	6,240	7,854	.....	.....
1904	1,573	13,824	8,294	9,867	.....	.....
1905	1,386	17,504	10,504	11,888	.....	.....
1906	1,569	20,489	12,293	13,862	691	1,827
1907	1,143	18,532	11,119	12,262	924	2,390
1908	1,174	21,502	12,901	14,075	863	2,056
1909	834	22,859	13,716	14,550	1,216	2,275
1910	1,003	24,524	14,724	15,727	808	1,690

(a) As reported by H. A. Watson & Co., Liverpool.

China.—Practically all of the tin mines of China are in the province of Yunnan. The metal is chiefly shipped through the port of Mengtze, thence going via Haiphong to Hongkong. About 97 per cent. of the output goes through Mengtze; the remainder through Wuchow and Kowloon, with a trifle now and then from Hankow. The tin reaches Hongkong in rude slabs. At Hongkong are several refineries which handle about 470 tons per month, producing three grades of metal, viz., No. 1, with 98 1/2–99 per cent.; No. 2, with 97 1/2–98, and No. 3, with 95–96. The exports of tin from China in recent years have been as follows:

Year.	Pounds.	Year.	Pounds.	Year.	Pounds.
1906	9,053,177	1908	10,635,345	1910	13,000,000
1907	8,196,790	1909	9,931,775	1911	12,000,000

(a) Exports to Great Britain and Europe only; reported by Hong Kong Chamber of Commerce.

Large portions of the shipments to Europe are made on option for the United States and the latter country takes a much larger share of the output than the figures indicate. Declared exports to the United States direct from Hongkong amounted in value to $798,941, in 1911, as compared with about $750,000 the year before.

Besides the tin exported a considerable quantity is used in China,

especially at Ningpo and Swatow, where it is manufactured into tinfoil and also is coined for token money. German interests, which recently secured concessions in Yunnan, are installing a modern dressing and smelting plant at Kotieou, near Mengtze, and this is expected to increase the output of tin materially, besides furnishing a better quality of metal.

Dutch East Indies.—During 1911, 4,936,800 lb. of Billiton tin were sold at public sales (by tenders) in Batavia. These sales were held monthly, at each of which 411,400 lb. were sold excepting the December sale with 374,000 lb. The average price per pound paid for the tin was 39.32 cents, against 32.22 cents for 1910. The highest price in 1911 was reached in December, 41.98 cents, and the lowest in March, 36.96 cents. Exports of Banca tin (Government) for the first 11 months of 1911 were 29,803,992 lb. and of Billiton tin (private) 4,149,088 lb., making a total of 33,953,080 lb. (37,040,000 lb. estimated for the year) against 37,425,840 for 1910.

Federated Malay States.—The chief mineral mined is tin ore. At least nine-tenths of the ore is obtained from alluvial deposits. Lode mining on the east coast of the peninsula has been carried on almost exclusively by one company, with varying results. On the west coast, tin ore is found in the limestone, granitic and schistose formations. In the limestone rocks the ore occurs in "pipes," but such deposits are irregular and unreliable. In the granitic and schistose rocks small irregular veins are found which contain arsenical and sulphurous ores in addition to tin.

The alluvial deposits are worked by opencast mining, by underground methods, and by ground sluicing, hydraulicking, or by dredging. The most suitable lands for opencast mining are broad flats in which the beds of tin-bearing ground are regular. Underground methods are resorted to when the deposits occur at depths which make opencast mining impracticable. This method is profitable only with the richer deposits. Dredging and hydraulic mining make it possible to work ground that otherwise would be untouched. By these methods, in some instances, ground carrying one-fourth to one-half pound of tin per ton may be profitably exploited.

There are 500 licensed purchasers of tin ore in the Federated Malay States. Over two-thirds of the ore is shipped to the Strait Settlements to be smelted, although 25 years ago the miners smelted their own ore. Chinese smelters in the states use primitive blast-furnaces, with charcoal as fuel, slag being treated nine times over, until it becomes uneconomical to work further. Chinese competition with the Straits Settlement smelters occurs only when tin commands a high price.

Labor in the tin mines is almost entirely Chinese, only 5 to 7 per cent.

being Indians, Javanese and Malays. There are two methods of employing laborers: They may be hired at the owner's risk, in which case the latter takes all profits, or the land may be let to the laborers for a fixed tribute. There were 184,699 laborers employed in the four states in 1909 and 170,361 in 1910.

PRODUCTION OF TIN IN THE FEDERATED MALAY STATES. (a)
(In pikuls of 133⅓ lb.)

	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Perak	436,296	443,507	446,781	435,909	431,386	467,784	461,665	421,335	437,339
Selangor	284,592	300,413	289,867	268,624	273,900	282,540	266,007	240,192	231,175
Negri Sembilan	85,461	84,849	85,133	77,766	75,155	64,221	48,072	34,697	29,230
Pahang	25,317	27,469	34,879	34,488	33,195	39,520	43,144	40,674	43,954
Total	831,666	856,238	856,660	816,787	813,636	854,065	818,888	736,898	741,698
Metric tons	50,254	51,790	51,793	49,859	48,411	51,654	49,529	44,579	44,870

(a) Exports of tin and tin ore, the latter being calculated to its assumed metal contents, upon which duty is paid to the F. M. S. government.

Duty is levied on all tin and tin ore exported from the Federated Malay States, and is calculated on a sliding scale which varies with the price on the London market.

Agitation toward a reduction in this export tax met with refusal. The Government has, however, decided that, in cases where it is proved that ore is difficult to win or to treat, a special rebate may be allowed after investigation by a committee appointed for the purpose. The tin deposits of the Federated Malay States are regarded as part of the capital of the country which is being withdrawn from the country and not being replaced. It is obviously the correct policy of the Government to secure for the country generally a fair share of that capital, and this is obtained by the export duty, or rather royalty, which is imposed on the wealth which is being withdrawn from the country. It is not admitted by the Government that the present rates bear heavily on the mining industry, or that the normal development of the industry is retarded by those rates, and no attempt appears to have been made by those who advocate a general reduction of royalty to show that the cost of placing tin obtained in the Federated Malay States on the London market is high compared with the cost in other countries.

Germany.—The tin-mining industry has practically disappeared, but that of smelting tin ores, imported mainly from Bolivia, is in a flourishing condition. The production of tin metal in 1911 was 12,412 metric tons (valued at 39,496,000 marks) as compared with 11,394 tons (31,272,000 marks) in 1910. Imports of tin and tin ore were as follows:

IMPORTS OF TIN ORE INTO GERMANY.
(In metric tons.)

Country.	Tin Ore.				Country.	Tin Metal.			
	1908.	1909.	1910.	1911.		1908.	1909.	1910.	1911.
Bolivia	9,136	13,099	15,925	.....	Dutch Indies	7,640	7,596	8,135	.....
Chile	882	978	914	.....	Australia	2,211	1,826	1,129	.....
Spain	598	672	227	.....	Straits Settlement.	1,087	671	471	.....
Austria-Hungary	63	190	48	.....	Great Britain	971	1,390	1,302	.....
Great Britain	523	324	34	.....	British Indies	804	1,058	1,105	.....
Other countries	218	160	195	.....	Other countries	1,326	996	2,155	.....
Total	11,420	15,423	17,343	17,971	Total	14,039	13,537	14,297	14,500
Value (1000 M.)	14,845	20,050	26,709	.....	Value (1000 M.)	37,814	36,414	43,773	.....

Russia.—Geological investigations are being made in the Sossnowics and Dombroff basins of Poland, in search of tin ore. Tin ore has already been struck at Korguvka. It is intended to proceed to the exploitation of these deposits at an early date.

In Siberia, German capitalists of Berlin and Hamburg, with the co-operation of the St. Petersburg International Bank, have formed a company for the exploitation of tin mines in the province of Transbaikal (Nerchinsk district), 3 1/2 miles from the station of Olovyanka, on the Transbaikal Railway. Tin ore has been known for a long time to exist in this district. Recently investigations were made here, and it was seen that the veins are of considerable extent. In the neighborhood of the Pervonachalny placer are gravel deposits containing stream tin. In this district are also the known Kulundinsky tin placer, the Sharanaisky tin placer, and the Zavitinsky tin placer. The Transbaikal Tin Smelting Works will be the pioneers of the tin industry in Russia.

Siam.—The most exploited region is the island of Tonka, a little to the south of the isthmus of Kra which, in 1908, produced 3713 metric tons of cassiterite, yielding block tin to the amount of 2302 metric tons. The neighboring province of Monthon Chumpon, situated a little to the north, but in the gulf of Siam, is rich in tin, especially in the districts of Lang-Suan and Lampun. The district of Monthon Chumpon will soon be provided with a railway, by which it is hoped its resources will be quickly developed.

Spain.—In the provinces of Orense and Pontevedra occurs a broad belt of schists, overlapping into Portugal, which is possibly the same formation as that in which tin is found in Cornwall. Near Rizadavia, Orense province, the Arnoya Mining Co., Ltd., has worked two tin mines since 1907, and concentrates the ore. The Balcobo Tin Mines, Ltd., is 35 miles north of Orense. Tin ore is found both in veins and in placers; the former are being further explored by drilling.

In Portugal, the same zone extends as far south as the Tagus River; the principal occurrence of tin is at Miranda, on the Douro River.

United Kingdom.—The principal events of 1911 were the completion of Williams' shaft, Dolcoath, at a cost of considerably over $500,000; the installation of an up-to-date tube mill at East Pool; the appointment of an Australian engineer to the management of Carn Brea, and the subsequent resignation of the chairman, directors and secretary; the ever-increasing use of electricity supplied to the mines by independent power companies; the extension of the magnetic separator and consequent higher price for the concentrates; the Royal Mines Commission to investigate all the principal mines; the lowering of the death rate due to miners' phthisis; the successful working of the largest suction dredge; not to mention the bitter feelings of the miners against some members of the Metal Exchange, who, it was said, were about to penalize unduly such impurities as wolfram, antimony and arsenic.

The total output of concentrated tin ore in 1911 was 8,200 long tons valued, at the works, at £617,376, as compared with 7572 tons, worth £655,871 in 1910.

The price, which in 1910 averaged $515, advanced to $563.75. The mines that paid dividends increased from three to seven and the amount of dividends proportionately. Dolcoath paid 12 1/2 instead of 6 1/4 per cent. and South Crofty paid 25 cents quarterly. In 1911, the mines produced less, but the quarries and surface works more.

Three suction dredges and one bucket dredge were at work, but the quantity of black tin recovered did not exceed 70 tons, on account of the difficulties of adopting the dredges to local circumstances. The future of the dredging for tin is encouraging, and other dredges will be built to work Restrouget and other creeks during the coming year.

Imports of tin and tin ore into Great Britain were as follows, according to trade and navigation returns:

IMPORTS OF TIN ORE INTO GREAT BRITAIN.

	1909.		1910.		1911.	
	Long Tons.	£	Long Tons.	£	Long Tons.	£
South America	18,951	1,291,563	19,982	1,362,134	21,439	1,738,059
British South Africa	2,675	190,826	3,339	290,427	3,407	342,031
Other countries	2,460	138,786	2,751	168,221	3,991	226,106
Total	24,086	1,621,175	26,072	1,820,782	28,837	2,306,196

For more detailed figures of 1910 and preceding years, see table in Vol. XIX of THE MINERAL INDUSTRY.

IMPORTS OF METALLIC TIN INTO GREAT BRITAIN.

	1909.		1910.		1911.	
	Long Tons.	£	Long Tons.	£	Long Tons.	£
Straits Settlements, Malay States and Labuan.	36,084	4,857,798	40,604	6,294,052	39,691	7,566,633
Australia	4,674	628,912	3,872	601,132	3,571	679,960
Other countries	967	117,134	1,809	264,598	2,644	492,879
Total	41,725	5,603,844	46,285	7,159,782	45,906	8,739,472

The Tin Markets in 1911

London.—The market had closed on Dec. 30, 1910 with a decline to £173 15s. per ton upon the announcement of increasing supplies of Banka tin for 1911. Large sales were privately effected at advancing prices, and when the market reopened on Jan. 3, there was keen competition among buyers for the little which was on offer. The result was a rebound in the price of no less than £5 10s. for 3 months' delivery, and £5 for prompt. Liberal offerings from the East from time to time were readily absorbed. The advance culminated on Jan. 30 when £204 was paid.

February fluctuations continued wide and wild. The price went during the month from £194 10s. down to £171 and up to £200. This was again the signal for aggressive selling, resulting in a fall to £187 per ton.

In March, fluctuations continued wild, the lowest being touched on March 16 when 3 months' warrants were booked at £173 10s.; after which the tendency was generally upward until the end of the month when cash warrants closed at £188 17s. 6d. and 3 months, at £187 2s. 6d. The most notable features of the month were the constant stream of orders for consumption, the persistent diminution of stocks, and the capricious attitude of Eastern sellers whose policy ranged from free selling to complete reserve. The closing for April was £203 10s. for prompt delivery, and £190 10s. for 3 months.

In May the market continued active and eventful. Some effort was made to initiate business on a new basis whereby sellers reserved option to deliver Straits or Australian tin at contract price, or Banka, Billiton, English or Chinese brands at certain allowances. Some transactions passed on this basis, but it lacked official recognition by the Committee of the London Metal Exchange, and no official quotation or settlement price could therefore issue, so that the project failed to influence the speculative market to any appreciable extent. Fluctuations in price

were incessant and often sensational, closing prices being £210 and £190 respectively.

Opening prices in June were £213 per ton for prompt warrants, and £190 10s. for 3 months. Consumers were not disposed to buy more than they absolutely needed in such an inflated market, but trade requirements gave sufficient support to maintain the foregoing prices and occasionally to augment them. The crisis came about on June 9, when "bears" became alarmed at the tightening squeeze and paid up to £233 per ton for early delivery, this representing a backwardation of £43 per ton. The syndicate then reduced its prices almost sensationally by lending tin for 3 months' delivery at a backwardation of £10 only, as compared with about £42 which had ruled on the previous day. Simultaneously they lifted the price of forward tin from about £186 to £190, and thus brought cash values to about £200. On June 16 further quantities of prompt metal were provided at a premium of £5, and gradually the backwardation was wiped out entirely, prices for all dates coming to about the same level around £188. Thereafter ensued an active business with consumers, particularly in America. Eastern sellers supplied the market so liberally that short selling was encouraged and values consequently relapsed.

In July the market opened with a fall of £1 per ton in the price of the prompt metal, but with no appreciable alteration in that of forward delivery. In August the market was sensitive at the outset, particularly for early delivery although the volume of business was restricted. Speculation was checked by pervading industrial unrest, complications in international politics, and particularly by the persistent depression in the stock exchanges. The consuming industries were everywhere well employed, but there was some apprehension of unfavorable statistics at the end of the month.

September statistics showed a diminution in the visible supply. There was little speculative business in the market but trade demand—especially in America—continued active, so that the market was very sensitive although the turnover was relatively small.

In October the market opened actively; the main tendency was upward, top prices being reached on Oct. 23, when cash warrants were quoted £191 12s. 6d. and three months' £191 7s. 6d. These figures were generally regarded as artificially inflated, business being restricted in view of the control exercised by the Syndicate.

November opened wit an apparently weak market, in spite of encouraging statistics. The new form of contract for standard tin had come into vogue and was operative for transactions maturing on and after Feb. 1, 1912; under which conditions a fair volume of business took

place. The 3 months' price drifted down to £183 10s. on Nov. 3. Meantime business was transacted in Straits and Australian sorts for delivery up to Jan 31 at prices considerably above those ruling for February delivery in standard grade; the Syndicate brokers accumulated large quantities of the former with the evident intention of holding it for a substantial premium. Tin now took a decided turn upward, and the advance in values continued with little interruption to the end of the month. The advance in prices was further stimulated by strong American demand. The Dutch sales took place on Nov. 29, when 2500 tons of Banka fetched the remarkable equivalent of £208 10s. per ton. The result in London was eager buying of December supplies, mostly from the Syndicate, and ranging up to £205. Standard tin was also in better request.

In December fluctuations in price were moderate during the early days with increased activity in standard tin under the new contract. Business in Straits and Australian for December and January dates was confined to consumers' orders, the prices rising at one time to £207 for prompt cash, and £201 15s. for late January.

AVERAGE MONTHLY PRICE OF TIN IN LONDON. (a)
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.
	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d
1899	99.16 .4	108.16 .3	107.16. 8	114 .1 .1	117 .9 .6	117.12 .0
1900	118 .9.11	137.18 .4	142 .0 .0	137.15 .0	135 .1 .8	139 .9 .3
1901	120 .9.10	122 .6.11	116.15 .6	116 .3 .0	123.13 .0	129.16.11
1902	105. 6 .5	114 .4 .9	115.10 .6	125.14 .2	134.13.10	129.12.10
1903	127.12 .6	133 .8 .1	137 .0 .6	136.19 .2	133.12 .0	127.11 .0
1904	130.10 .4	125.13 .6	126 .9 .8	127 .5 .1	125 .7 .2	119.11 .1
1905	131 .5.11	131 .3 .6	134.17 .2	140.11 .8	136.11 .8	138 .3 .6
1906	164.11.10	166 .0.10	166 .1 .2	176.14 .5	192 .6 .4	178 .0 .7
1907	190 .4 .0	191.18 .9	188.17 .6	187 .1 .2	191 .1.10	187.10.11
1908	128 .9 .0	128.14 .1	137.19 .8	143.12.10	135.11 .6	127.12 .2
1909	127 .7 .3	127.15 .3	130 .6 .7	133 .8 .3	131.16.10	133 .4 .0
1910	148 .3 .6	149.13 .0	147 .4.11	149.19 .3	150 .1 .8	148.13 .1
1911	186.17.11	189.12.10	182.12 .4	193 .0.10	197.15 .4	207 .7 .9

Year.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d
1899	132.13 .1	142 .1 .4	146 .7 .2	144.10 .2	129.16 .0	113 .0 .7	122 .8 .7
1900	142.16.10	140.19 .1	132.13 .9	130.14 .3	127 .3 .8	119.14 .9	133.11 .6
1901	127.19 .9	116 .1 .7	114.10 .6	113 .1 .5	114 .0 .7	108.17.10	118.12 .8
1902	127 .3 .2	126.10 .0	121.10 .7	117.11 .3	115 .2 .3	115.13 .5	120.14 .5
1903	125 .1 .7	127.16.10	120 .9 .6	115.17 .1	116.13 .9	125.15 .0	127 .6 .5
1904	119.18 .6	122 .5 .9	126 .7 .7	130.11 .6	133 .0 .5	133.15 .6	126.14 .8
1905	144 .6 .8	150 .5 .6	146.11 .9	148 .3 .6	152 .5 .3	162.14 .3	143 .1 .8
1906	170.12 .5	180.19.11	184.15 .3	195.15.11	195.15.10	195.19 .9	180.12.11
1907	188 .0 .2	170 .5 .9	166 .6 .6	146 .7 .7	138 .8 .8	125.10 .4	172.12 .9
1908	131 .6.10	134.16 .2	131 .6 .8	133 .8 .8	137 .8 .3	132 .4 .7	133 .2 .6
1909	131.19 .1	135.18 .3	137.14 .6	138.13 .2	140 .0 .3	149 .2 .3	134.15 .6
1910	149 .2 .1	155 .8 .2	160 .8 .7	164 .4 .9	166.18 .0	173.17 .3	155 .6 .2
1911	193 .2 .0	190 .9 .7	180.16.11	187 .2 .9	194.19 .4	203 .7 .2	192 .7 1

(a) As reported by Metallgesellschaft, Frankfurt-am-Main.

The new tin rules of the London Metal Exchange went into effect on Feb. 2, 1912. They were intended to broaden the market by admitting several brands which were not good delivery under the old rules, and by so doing to increase the difficulty of making corners in the metal.

Deliveries.—Sellers have the option to deliver each 5 tons either in: *Class A*—(1) Straits or Australian tin of good merchantable quality at contract price. (2) Refined tin of good merchantable quality, assaying not less than 99.75 per cent. at contract price. *Class B*—Common tin in ingots or slabs of good merchantable quality, assaying not less than 99 per cent., subject to a rebate of £7 per ton from contract price.

Rule 1—The warrants tendered shall be those of a public warehouse in London, Liverpool, or Swansea, the name of which shall appear on the official list of the committee (of the Metal Exchange). The tin shall have been weighed in drafts of 10 cwt. each, and the warrants shall be as far as possible for 5 tons each. Each parcel of 5 tons shall be of the same description and class and shall lie at one wharf; but sellers shall be at liberty to tender parcels of 5 tons composed of metal of the same description and class, but lying at different wharves in the same port, in which case they shall allow buyers 5s. per ton for each 5 tons so divided. Warrants tendered against a contract for a multiple of 5 tons shall, either separately or together, be capable of subdivision into parcels of 5 tons each.

Rule 2 provides for the sampling of Straits and Australian tin by taking one sample from each lot to be delivered to the buyers.

Rule 3—Straits and Australian tin, if below the standard of good merchantable, shall not be considered a valid tender; and sellers shall substitute a similar quantity of standard tin. The rest of this rule provides for the time allowed for substitution.

Rules 4 and 5 provide for the methods of drilling, sampling and testing. Rule 6 provides that all warrants shall be accompanied by certificates from assayers approved by the committee. If there is delay in furnishing the certificates, buyers may hold back 10 per cent. of the price until the certificates are furnished. Rule 7 provides for the rejection of tin not coming up to the standard, and for the substitution of other tin; also for arbitration in case of dispute.

Rule 8—Sampling shall, in all cases, take place independently by the wharfinger at the public warehouse, and the entire cost thereof and of the assaying and testing shall be borne by the sellers.

New York—The market for tin was entirely dominated by the Syndicate—consisting of London and continental firms—which, through its successful operations during the preceding years, has become very powerful. In describing the fluctuations of prices of this metal, one is

therefore compelled simply to give a description of the manipulations of the Syndicate, which were but little disturbed by the quantities of the metal not under their control. The Syndicate was assisted in its bull operations throughout the year by the favorable statistics of the metal, which disclosed the fact that actual production has hardly come up to consumption, so that existing stocks had constantly to be drawn upon.

The New York market opened strong at around 40 cents, advancing during the month of January to about 43 1/4 cents. Fluctuations in

AVERAGE MONTHLY PRICES OF TIN PER POUND IN NEW YORK. (a)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1897	13.44	13.59	13.43	13.34	13.44	13.77	13.89	13.80	13.98	13.88	13.79	13.71	13.67
1898	13.87	14.08	14.38	14.60	14.52	15.22	15.60	16.23	16.03	17.42	18.20	18.30	15.70
1899	22.48	24.20	23.82	24.98	25.76	25.85	29.63	31.53	32.74	31.99	28.51	25.88	25.12
1900	27.07	30.58	32.90	30.90	29.37	30.50	33.10	31.28	29.42	28.54	28.25	26.94	29.90
1901	26.51	26.68	26.03	25.93	27.12	28.60	27.85	26.78	25.31	26.62	26.67	24.36	26.74
1902	23.54	24.07	26.32	27.77	29.85	29.36	28.38	28.23	26.60	26.07	25.68	25.68	26.79
1903	28.23	29.43	30.15	29.81	29.51	28.34	27.68	28.29	26.77	25.92	25.42	27.41	28.09
1904	28.85	28.09	28.32	28.13	27.72	26.32	26.57	27.01	27.78	28.60	29.18	29.292	27.99
1905	29.325	29.262	29.523	30.525	30.049	30.329	31.760	32.866	32.095	32.481	33.443	35.835	31.358
1906	36.390	36.403	36.662	38.900	43.313	39.260	37.275	40.606	40.516	42.852	42.906	42.750	39.819
1907	41.548	42.102	41.313	40.938	43.149	42.120	41.091	37.667	36.689	32.620	30.833	27.925	38.166
1908	27.380	28.978	30.577	31.702	30.015	28.024	29.207	29.942	28.815	29.444	30.348	29.144	29.465
1909	28.060	28.290	28.727	29.445	29.225	29.322	29.125	29.966	30.293	30.475	30.859	32.913	29.725
1910	32.700	32.920	32.403	32.976	33.125	32.769	32.695	33.972	34.982	36.190	36.547	38.199	34.123
1911	41.225	41.614	40.157	42.185	43.115	44.606	42.400	43.319	39.755	41.185	43.125	44.655	42.281

(a) From *Eng. and Min. Journ.*

February became very violent, due to some urgent liquidation on part of a member of the Syndicate, which brought down prices during the first 10 days of the month to about 38 cents. After this liquidation was completed, the market advanced just as rapidly as it had declined, reaching 44 cents by the middle of the month and remaining fairly steady until the end of February.

March showed a continuous decline to about 39 1/2 cents. advancing again to about 42 cents by the middle of April. From this level it moved irregularly during the next few weeks, without, however, displaying any violent fluctuations until the middle of May. Then a buying movement on part of the Syndicate, especially in spot tin, advanced the market rapidly in London, which was naturally reflected in the domestic market. The market advanced quickly to about 46 cents early in June.

The corner in spot tin, which advanced the price thereof in London about £44 over the 3-months quotation, thereby disclosing a very unhealthy condition of things, induced sellers in this market to liquidate their holdings of near-by material at about 3 cents below the price at which it could be imported. The squeeze of the shorts lasted until the middle of June, when spot tin suddenly declined about £41, while 3-months tin scored a small advance, thus bringing both quotations

closely together. Spot tin was selling in this market at about 42 3/4 cents, advancing gradually from this level to 43 3/4 cents by the middle of July. Some outside selling then appeared, which broke the price temporarily. After this selling had subsided, the market became strong again, and reached 45 cents toward the latter part of August. This meant a premium of about 4 cents for spot tin, of which supplies in this market were small and closely held. This premium disappeared almost entirely at the beginning of September, and the market declined further in sympathy with London, where the unsettled political situation had a disturbing effect.

At the end of September the market was established at about 38 5/8 cents, New York. The market became firm again during October, and reached 42 cents the latter part of the month, advancing further the beginning of November.

Secondary Tin.—There is a considerable quantity of tin recovered from scrap, as tinplate clippings, old plates and other used material. The Geological Survey estimates the quantity thus recovered in the United States in 1911 at 7,749 tons. This material is not marketed to any considerable extent as pig tin, but chiefly in alloys—such as bearing metals, solder and bronze—and as chloride and other salts of tin. The quantity thus recovered in Europe is estimated at about 14,000 tons, making the total production of secondary tin in 1911 about 21,750 tons.

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TITANIUM

By Bradley Stoughton

The metal is derived from the oxide which occurs with iron, as in titanic iron, or free from iron as in rutile. The rutile deposits near Roseland, Virginia, furnish a fine concentrate of 75 per cent. titanic oxide and other deposits in the vicinity are of low grade and contain phosphorus. The production of concentrates in 1910 is stated[1] to be 566 short tons, valued at $44,480 and was produced from 8000 tons of ore. The large deposits of titaniferous iron ore in the Adirondacks furnish the mineral for the making of ferro-titanium.

The American Iron & Steel Association reports that during the calendar year 1911, 3,679,335 lb. (1643 gross tons) of ferro-titanium alloy were shipped by an American producer to 79 manufacturers of steel in the United States. From statistics the average consumption appears to be 9 lb. of ferro-titanium per long ton of steel treated.

In foreign countries a small production of rutile (50 tons in 1910) comes from the deposits at Kragero, Norway. The deposits in South Australia require much concentration and are not promising.

In America, in 1911, 410,600 gross tons of steel were treated with titanium, of which something less than one-half was rail steel made by both the Bessemer and the basic open-hearth processes. In brief, the amount of steel treated with titanium exceeded the amount treated with vanadium, chromium, tungsten, nickel, and all other elements combined, except manganese and silicon (aluminum). In addition to steel, a large tonnage of cast iron—the exact amount being unrecorded—was treated with titanium, as were also substantial amounts of copper, brass and other non-ferrous metals and alloys. In England and Europe, much cast iron and steel was so treated commercially, as indicated by the sales of the alloys used, but more exact statistics are lacking. The most significant changes during the year are the increasing amount of structural steel and open-hearth steel, as distinguished from rail steel and Bessemer steel respectively, and the increasing amount of gray cast iron treated, as distinguished from chilled cast iron. In other words, the titanium addition had previously established itself in the rail steel and in the chilled iron industries, no doubts remaining as to its advantages for these products. The year 1911 sees also its prominent entrance into the structural steel, open-hearth steel and gray cast iron

[1] U. S. Geol. Survey.

industry, and into the steel casting industry. Of course it has been used in the past in notable quantities in these classes of product, but it could hardly have claimed to be established there until recently. Even at the present time it has many unfriendly critics, but present information indicates that the ill results obtained from its use, which are the foundation of the arguments advanced against it, were the consequence of improper methods of using the alloys, rather than lack of virtue in the treatment.

The chief commodities treated in 1911, outside of rails, were iron and steel rolls, tires, steel castings, gray iron and malleable iron castings, axles, plates, gears, etc. It is also interesting to know, even though not yet commercially important, that metallic titanium has been used at least experimentally for electric light filaments, and titanium carbide, for electrodes.

A lively correspondence was carried on during the year in the issues of *The Iron Age* on the relative merits of titanium containing aluminum without carbon, and that containing carbon without aluminum (see appended Bibliography). On the one hand, W. R. Hulbert claimed that the presence of carbon, by its union with titanium, decreased the availability of the latter for deoxidation, whereas the presence of aluminum made the alloy dissolve in the bath better by reducing the melting-point of the alloy, and also cleared the bath more thoroughly from occluded solids by fluxing off the titanium oxide. In opposition to him, G. B. Waterhouse claimed that the carbon present was in the graphitic form, and not united with the titanium, and gave analyses to establish this contention; he admitted the presence of aluminum was an advantage in so far as it lowered the melting-point of the titanium alloy, but claimed that the alumina formed increased the amount of occluded solids which the titanium oxide had to flux off, and therefore was a hindrance, and stated his experience as a steel-maker with several years' familiarity with the use of the alloy, to the effect that the iron and other oxides present fluxed off the titanium oxide without the necessity of any alumina being present.

Further notes on the action of titanium are given in MINERAL INDUSTRY, Vol. XIX, p. 658. The use of titanic oxide in fused silica is given under Quartz, p. 649.

According to U. S. patent 1,003,806, Sept. 19, 1911, A. J. Rossi of the Titanium Alloy Manufacturing Company produces an alloy of nickel and titanium by mixing the oxides and metallic nickel with aluminum or carbon and treating in the electric furnace. A number of patents were issued during the year on the use of titanium carbide for electrodes.

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TUNGSTEN

A notable decline occurred in the output of tungsten ore during 1911, owing to smaller demand for ferro tungsten in the steel business. Many former producers closed down entirely, while others piled their output at the mines, waiting for a rise in the market. Figures reported to us by the producers of the United States, and recalculated to a uniform basis of 60 per cent. tungstic oxide, account for an output of 1293 short tons, valued at $500,247, in 1911. Prices ranged from $5.80 to $7.50 per unit per ton, the largest tonnage averaging $6 per unit. No new producers entered the field in 1911, and those already engaged in it did not operate at more than one-half their capacity.

PRODUCTION OF TUNGSTEN CONCENTRATE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.	Value.	Average per Ton.	Year.	Production.	Value.	Average per Ton.
1902.....	184	34,040	$185	1907....	1640	890,048	543
1903.....	292	43,639	149	1908....	671	229,955	343
1904.....	740	184,000	249	1909....	1619	614,370	379
1905.....	803	268,676	334	1910....	2020	844,526	418
1906.....	928	348,867	376	1911....	1293	500,247	387

(a) Statistics reported by the U. S. Geological Survey, except for 1910 and 1911.

Tungsten Mining in the United States

Arizona.—Operations at the close of 1911 were inactive in the Dragon Mountains, the region where wolframite was first found and which has produced most of the state's output. There were several reported sales of tungsten properties in the Guijas Mountains, in Pima County, where the ore occurs in strong quartz veins, up to 30 ft. wide, cutting an aplitic granite. These veins also carry some gold. The only deposit of scheelite yet uncovered in the state was in process of development by the Cody-Dyer Mining Co. This deposit is in the Catalina Mountains, 50 miles north of Tucson, and occurs as a lenticular quartz vein in shales, close to the margin of the intrusive Oracle granite. The oreshoot is small but has been opened to a depth of 200 ft. A few carloads of ore were mined near Arivaca, 70 miles south of Tucson, hauled to railroad and shipped to Denver, Colo., for concentration.

California.—The output of tungsten in California in 1911 was less than in 1910, owing to a falling off in demand. The principal production came from the Atolia Mining Co. in the Randsburg district in San

Bernardino County. The plant was not operated full time during the year, but the company had a good supply of ore on hand, so the suspension of operations gave opportunity to make some needed repairs and improvements. The usual run of about 40 tons per month of 65 to 67 per cent. concentrates was made while in operation. Other tungsten ores were mined in the district during the year. Tungsten was discovered in the Forbestown district in Butte and Nevada Counties, associated with the gold in the sulphide ores; but it was not determined of commercial value, owing to the presence of sulphur and pyrite. At Nipton, considerable development occurred, but no shipments.

Colorado.—The ore mined in Boulder County is mainly ferberite (iron tungstate), which is not obtained in large quantity at any other place in the world. In spite of the poor market several new companies were formed to mine ore in the Boulder field and commenced operations during the year. Two companies profitably worked over tailings left from former operations. A considerable quantity of the ore mined both in this and in other fields was not marketed. The total output of this district in 1911 was about 1000 tons of concentrate worth $360,000.

Complete details of milling practice in the district are given by H. C. Parmelee in *Met. and Chem. Eng.* for July and August, 1911. The tungsten minerals in the Boulder field are the three iron-manganese tungstates, ferberite, hubnerite, and wolframite, and the calcium tungstate, scheelite. Ferberite predominates, but scheelite occurs in many mines. These minerals are of high specific gravity, scheelite being 6 and the other three ranging from 7.1 to 7.5. They are generally soft and friable. The ferberite occurs in three well-defined forms which, however, frequently grade into each other. These are (1) well-crystallized crusts and layers covering the surface of rock fragments and cementing them together; (2) massive granular ore occurring in more dense seams; and (3) the highly silicious ore in which ferberite occurs in fine grains scattered through the gangue.

In breaking ore in the mine the highest grade is sorted and shipped without preliminary milling. At present this grade must average 40 per cent. or more tungstic acid. The lower grades, ranging from 3 per cent. to 15 per cent. tungstic acid, are concentrated by wet methods. Two grades of concentrate are recovered and shipped; the first contains 60 per cent. or more tungstic acid, and the second ranges from 35 per cent. to 40 per cent. The former is usually recovered as coarse concentrate and the latter as slime.

The town of Nederland is centrally located with respect to the tungsten mills of Boulder County. It is reached by stage, 18 miles from Boulder, or by railroad to Cardinal station and thence by stage

for 2 miles. At Nederland is the mill of the Wolf Tongue Mining Co. At present this company is running its mill on a custom basis. At Cardinal, 2 miles west of Nederland, is the Cardinal mill of the Primos Chemical Co. This plant has been retreating the tailings made during 4 years' previous run, when the methods of saving were not as efficient as at present. The Lakewood mill of the Primos Chemical Co. is at Lakewood, about 2 1/2 miles northeast of Nederland. It is the newest and most attractive of the mills in the district. It treats the company's own ore at present, although a great deal of custom ore has been treated. Between Cardinal and Nederland is the mill of the Alton Mining and Milling Co., a small plant for company or custom work. On Beaver Creek, about 2 miles southeast of Nederland, are two small mills, one belonging to the Tungsten Mining and Milling Co., treating company ore, and the other owned by Smith & Ardourell, at present retreating the old tailings of the Wolf Tongue Mining Co. The Clarasdorf mill on lower Middle Boulder Creek is idle. In addition to the regular milling, an effort is made in a small way to concentrate some of the old mine dumps, either by screening and saving the fine mineral, or by hand jigging. Material obtained in this way is sold to the mills.

The mills are of small capacity. Four of them can treat about 5 tons each in 24 hours, one of them 24 tons and another 40 tons in the same period. With two exceptions they are built especially for tungsten work and are new. The two old mills are remodelled gold mills. A notable feature is the large table area required per ton of ore treated, which naturally reduces the tonnage capacity. The Monell slimer is a general favorite. In no instance is screen sizing being practised with a view to eliminating coarse material which might prove worthless. In view of the friable nature of the tungsten minerals it would seem that sizing should prove worthy of investigation, and certainly classification should be more widely practised.

Idaho.—The deposits of Paterson creek, Lemhi County, are owned by the Ima Mining Co. and are operated under a long lease by the Idaho Tungsten Co., which has built a new mill. There are two tunnels, 700 and 900 ft. respectively, each showing ore as wide as 9 ft. The ore is hard white quartz full of sulphides of lead, zinc, copper and iron, and also hubnerite and wolframite. The ore is said to average 4 per cent. tungsten and a saving of 87 per cent. is made in concentration. Another deposit in the same neighborhood is being developed by the International Company.

Montana.—The occurrence of scheelite at Jardine, and of hübnerite in Butte, was described in Vol. XIX. A new discovery of hübnerite is reported in the Potosi mining district, an abandoned silver camp which

lies in an air line about 7 miles southwest of Pony, Madison County, Mont. The district is about 11 miles by wagon road which passes Potosi hot springs. The tungsten ore occurs in aplite dikes which are a part of a large igneous intrusion extending along the eastern slope of the Tobacco Root range. The ore occurs in distinct veins from 1 to 12 ft. in width. The elevation is 8000 ft. above sea level. The tungsten occurs in white quartz in long brownish needle-like crystals of hübnerite ($MnWO_4$). Samples which have not shown a large proportion of the crystalline mineral have given encouraging results by analysis. This seems to furnish evidence of the presence of tungstic acid in addition to hübnerite. The district is near railroad facilities, which will be an advantage in the development of the camp should the deposits prove of economic value.

Nevada.—The new mill erected in 1910 by the United States Tungsten Corporation, of Osceola, near its mine at The Hub, 12 miles south of that place, went into operation for a short time. The same concern owns other deposits on Williams creek, south of Mt. Wheeler. At the close of 1911, a new mill was being built at Round Mountain. Other new discoveries were made in the Sonoma range, south of Golconda, and in the New Goldfield district, near Rebel creek. At the last-mentioned locality a new mill was begun. The American Rare Metals Co. planned a mill to treat ore from a deposit of tungsten-bismuth ore near Central Peak.

Washington.—The Tungsten Consolidated Co., of Spokane, is completing a concentrator at its mine on Blue Grouse Mountain, 6 miles northeast of Loon Lake. With ore carrying 5 per cent. tungstic oxide, the mill is designed to produce 4 tons of concentrate per day.

A new tungsten mineral, ferritungstite, is reported (*Am. Jour. Sci.*, Aug., 1911) from the Germania Tungsten mine, in the Deer Trail mining district, in northeastern Washington.

OUTPUT OF CONCENTRATED TUNGSTEN ORE BY THE LEADING PRODUCING COUNTRIES.
(In Metric Tons.)

Country.	1908.	1909.	1910.	1911.
Argentina	497	817	1,912	(a)
Australasia:				
New South Wales	244	325	321	398
Queensland	426	617	873	553
Tasmania	5	29	68	71
New Zealand	88	79	179	140
Austria	40	39	40	(a)
Bolivia	170	152	210	(a)
England	237	382	278	267
France	113	54	30	(a)
Germany (Saxony)	42	96	95	(a)
Portugal	106	673	943	(a)
Spain	226	129	153	(a)
United States	599	1,469	1,833	1,173

(a) Statistics not available.

Tungsten Mining in Foreign Countries

Argentina.—Ore from a mine at Cordoba contains niobium in addition to tungsten: WO_3, 74.86; Nb_2O_3, 1.22; FeO, 13.45; MnO, 11.02 per cent. Other ores occur at St. Thomas and Santa Barbara, and in spite of the difficult transport, a few hundred tons have been shipped to Germany in past years.

Australia.—The output of tungsten ores in the commonwealth in 1911 was 1145 long tons valued at £115,512, of which Queensland contributed 544 tons, nearly all wolframite, and New South Wales 392 tons, of which 109 tons was scheelite and 283 tons wolframite. New Zealand's output was 138 tons, and Tasmania's, 70 tons.

In Queensland, the Chillagoe district yields tungsten ore mainly at Wolfram Camp and Mt. Carbine, the Irvinebank company being the leading operator at both localities. In its new mill at Mt. Carbine, this concern, between May and December, 1911, treated 6435 tons of ore, for a recovery of 148 tons of wolframite concentrate, at a cost of $1.64 per ton of ore. Mining cost reached a maximum of $8.28 per ton.

In New South Wales, scheelite is produced on a small scale at Hillgrove. Wolframite production is made at Torrington, where four fairly large concerns are actively engaged. A new mill is to be erected by the Broken Hill Block 14 Company near the principal workings, on Fielder's Hill, and as other mills have enlarged their capacity, increased production is to be anticipated. For geological data as to occurrence of the tungsten ore in this locality, see the *Mining Magazine*, Jan., 1912.

In New Zealand the producing centers are Lake Wakatipu, and Macrae's, both in Otago. The ore is scheelite and at the latter locality is recovered in the treatment of a gold ore. The material is consumed in Hamburg, Germany.

Bolivia.—Deposits of wolframite are found in the province of Caluyo, Department of La Paz. As a rule, these ores contain 56 per cent. of tungstic acid. Fluctuations in the European markets have caused many losses so that most of the mines have shut down and only the ownership is maintained.

Canada.—In New Brunswick, near the confluence of the southwest Miramachi River and Burnt Hill brook, certain quartz veins, long known to carry molybdenite, have recently been found to contain coarsely crystallized wolframite, of which an analysis gives: FeO, 16.90; MnO, 8.37; WO_3, 74.43 per cent. The locality is hard to reach, being distant from the railroad and practically without roads. The veins thus far uncovered reach a maximum width of 2 ft.

In Nova Scotia, the scheelite deposits on Moose River, Halifax County,

were actively developed in 1910 and 1911. The mineral occurs in quartz veins, seldom over 4 in. thick, which also contain varying proportions of arsenopyrite. A number of shallow shafts have been put down, disclosing eight or ten different veins, from which 100 tons of ore were extracted. The Scheelite Mines, Ltd., chartered in 1911, erected a mill of 30 tons capacity at Scheelite, which went into irregular operation early in 1912.

Portugal.—The output of tungsten ore concentrates amounts to something over 1000 metric tons per year, or about one-fourth the world's supply. According to R. S. Lozano, of Madrid, the mines could increase their output sufficiently to supply the present demand of all Europe, but their development would have to be undertaken in a more rational manner, with sufficient capital. The area through which the deposits are scattered is extensive, covering practically the whole western part of Spain and the northern part of Portugal. The veins consist mainly of quartz, and intersect granulites, or follow the contact of these with the surrounding sedimentaries. They are very irregular, and rarely extend to great depth. The tungsten mineral is disseminated with extreme lack of uniformity through the quartz, necessitating unusual care in acertaining its richness. Cassiterite is mingled with the tungsten minerals in some veins, but in other localities constitutes separate veins. The tin is practically confined to the northern part of the area.

According to the manager of the Mines de Borralha, the ores of Portugal are remarkable for their purity, containing neither tin, sulphur, phosphorus, nor bismuth; after being dressed by that company, they contain from 70 to 72 per cent. of tungstic acid. By reason of this purity these ores are largely used in French electrometallurgical factories for the production of ferro-tungsten. The mines at Borralha are the most important in the country, their production for 1910 having been 384 metric tons.

The Traz-os-Montes mine, in North Portugal, was developed entirely in 1911, but has already taken third place among the wolfram mines of Portugal, according to *Revue de Chimie Industrielle*, Mar., 1912. The product shipped in 1911 amounted to 51 metric tons, of an average of 70 per cent. WO_3, while at times the contents ran as high as 73 per cent. WO_3, although only crude concentrating methods were employed. An analysis of a typical lot was: WO_3 70.56; SnO_2, 0.32; insoluble, 3.40; Cu, 0.05; FeO, 13.04; Al_2O_3, 0.89; MnO, 10.60; CaO, 0.20; S, 0.04; H_2O and volatile, 0.38 per cent. There is no arsenic nor phosphorus. The average price received was 45.50 francs per unit. The wolfram is chiefly found in quartz stringers, but on one claim also disseminated in granite.

The concentration, consisting of crushing and washing, is done by women, hand labor. The ore is taken by rail from either Pinhaô, 17 km., or Alambres, 9 km., to Porto, where it is put on ships. Transportation is by bullock carts to Alambres and Pinhaô, the costs being about $1.60 and $0.80 respectively, and the railroad haul to Porto costs $1.20 per ton.

United Kingdom.—A fairly uniform output of 300 tons per year of wolframite concentrates is maintained by the Cornish tin mines and works. The average contents of WO_3 in these concentrates is 65 per cent. South Crofty is the largest single producer, followed by East Pool and Agar United. The output from mines in 1911 was 260 long tons, to which must be added a few tons from stream works.

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URANIUM

An active demand for uranium ores, particularly from European sources, characterized the uranium situation in 1911. At first it was thought that some new and important use for the metal as an alloy or in the chemical industries had been discovered, but later it was practically admitted by the buyers that the ore was wanted for the purpose of the extraction of radium and radio-active salts.

Interest in the carnotite deposits of Colorado and Utah was revived and great activity in shipping and in exploration in this locality resulted during the summer and autumn. It is probable that these deposits will be still more extensively exploited during 1912. The ore contains both uranium and vanadium. About 30 cars of carnotite were shipped, most of it to England but some of it to Pittsburgh. A small output of pitchblende was made from mines in Gilpin County, Colorado, and discoveries were reported from Chihuahua, Mexico. The chief output of pitchblende continues to be from Germany and Cornwall, England. Reported deposits of complex uranium ore in Guerrero, Mexico, were examined by the Mexican Geological Institute and were reported as not important. A deposit of radio-active uranium ore was reported in South Australia, and its development and exploitation were begun.

Colorado.—Early in 1912 a car load of carnotite ore was shipped to Europe from Placerville by the Colorado Carnotite Co., of Telluride. The shipment totalled 27 1/2 tons of hand-sorted ore, taken from the company's property at Long's Park, in the Paradox Valley.

Utah.[1]—Carnotite deposits are traceable along the east side of the San Rafel swell for about 60 miles. It is also found 25 miles southeast of Green River, where claims have been located by Cook and Roberts of Green River. This property seems to have a high-grade ore, and it proves that the carnotite field is much greater than had been supposed. The only property from which regular shipments are now being made is that owned by Angus Cameron and associates. This property is located 14 miles southeast of Green River, and consists of 20 claims, on which much development has been done. Regular shipments of one car per week are being made. All of the ore so far developed in this property lies on the surface, and is easily mined, the ore being soft and readily broken. The Hite-Loveless property is located about 10 miles south of the Cameron group, and consists of 10

[1] *Salt Lake Mining Rev.*, May 30, 1912.

claims; plans are being made to begin shipments at an early date. J. C. Morris has six claims located about 3 miles south of the Loveless holdings. The property belonging to Jessie E. Inman, of Green River, consists of a group of claims which lies between the Morris and the Loveless properties, and has every evidence of developing into a large producer. Shipments will be commenced not later than July, 1912. The contract for the ore from this property has been made as follows, with Salt Lake parties: $1.35 per unit for 3 per cent. uranium and $3.50 for 3 per cent. vanadium, and 2 1/3 times these prices for 6 per cent. or better.

The Cook-Roberts property, located 25 miles southeast of Green River, and 10 miles from the nearest shipping point, consists of six full claims, on which the vein outcrops for 1/2 mile. Shipments could easily be made from the Cook-Roberts mine on very short notice, the ore having been opened at a number of places along the outcrop.

The Temple Mountain property controlled by the Green River Engineering & Development Company, is one of the best properties in the middle carnotite section. Results from a shipment of 4 tons, to Ledoux & Co., New York, were 1 1/2 to 4 per cent. uranium and 3 to 7 per cent. vanadium.

Germany.—Bohemia, in 1910, produced 6 1/2 metric tons of uranium ore from the Government mine at Joachimsthal. The output of uranium preparations amounted to 10.3 tons, in which were consumed 14.7 tons mined during 1909 and 1910.

South Australia.—The government geologist, H. J. L. Brown, has reported[1] on the occurrence of uranium ores and other rare metals on the ground held under right of search by the Radium Extraction Co., of South Australia, near Mount Painter in the Flinders Range. Uranium ores have been found in several localities. Several discoveries have already been made within the radius of a few miles. The presence of niobium and other rare metals, associated with the uranium ores, adds value to these discoveries.

Analysis of Ferro-uranium.—From 0.4 to 0.6 gram of the powdered alloy is heated with *aqua regia* until solution is complete, after which the liquid is diluted, cooled and made alkaline with solid ammonium carbonate. After standing for 24 hours, with frequent stirring, it is filtered and the filtrate boiled to decompose the remaining ammonium carbonate, precipitating most of the uranium. The traces of uranium in the filtrate are removed by adding a few drops of ammonia solution, and the liquid is again filtered. The filter containing the uranium precipitates is washed with ammonium nitrate solution until only faintly ammoniacal, and the precipitates ignited in a porcelain crucible and weighed as U_3O_8.

[1] "Occurrence of Uranium Ores in South Australia." Printed by R. E. E. Rogers, North Terrace, Adelaide, 1911.

VANADIUM

Vanadium steel found many new applications and its use was greatly extended in 1911 as a result of scientific and practical demonstrations of its special advantages. Interesting results are expected from experiments under way with the use of vanadium as an additive agent with other metals than steel.

An important source of vanadium ore supply continued to be the patronite mines of the American Vanadium Co. at Minasragra, Peru. Vanadium is also being obtained from carnotite deposits in western Colorado, and from eastern Utah. The principal operators of the carnotite ores in the Paradox Valley, Colorado, are the International Vanadium Co., of Liverpool, Eng., and the Standard Chemical Co., of Canonsburg, Penn. Some of these ores carry (besides vanadium) uranium and radium and the utilization of salts of these metals will give miners of carnotite ore a decided advantage over producers of straight vanadium ores.

The mill at Vanadium (Newmire), Colo., controlled by the Primos Chemical Co., treated about 30 tons of roscoelite ore daily throughout 1911, with reported satisfactory metallurgical results. Much development was carried on in this district but practically no ore shipments were made nor was any new treatment plant established. Plans for at least one new mill are under way and the district promises to continue an increasingly important producer of vanadium.

The vanadinite (lead vanadate) deposits at Cutter, New Mexico, were operated in 1911 by the Vanadium Mines Co. of Pittsburg but operations were limited by reason of metallurgical difficulties with this ore. Other vanadinite deposits in New Mexico, Arizona, California, and Chihuahua received attention during 1911, but no important production resulted.

The reported discoveries of rich deposits of vanadium in Russia apparently did not materialize, for at the close of the year the European consumers were still buying largely from the American producers and were increasingly concerned in acquiring vanadium supplies.

Satisfactory statistics of the vanadium trade are impossible to obtain. The consumption in a measure is controlled by the state of the steel trade, but by reason of the expanding uses of vanadium steel it is likely that the consumption for 1911 was greater than any previous year not-

withstanding the lessened steel output. The Vanadium Sales Co. of America reported shipments, during 1911, of 689,510 lb. of ferro-vanadium to 57 plants in the United States and Canada, engaged in the manufacture of steel ingots or castings. A contributor to *Engineering* estimates that the annual consumption of metallic vanadium in the United States is 250,000 lb., and that 5 lb. of the metal is consumed for 1 ton of Vanadium steel.

The price of the vanadium product as usually marketed—ferro-vanadium—at the begining of 1911 was $5 per pound for the metallic vanadium content. During the year one of the large operators made price concessions and the alloy was openly quoted at $3.50 per pound and sales were reported at $3 per pound.

The Ores of Vanadium[1]

(By T. F. V. Curran).—The principal ores containing vanadium are patronite, carnotite, roscoelite, vanadinite and the hydrocarbon asphaltite.

Patronite is a sulphide of vanadium, associated with pyrite, in a carbonaceous substance resembling slaty coal, but abnormally rich in sulphur. It contains 35 per cent. vanadium sulphide, 5 per cent. iron sulphide, 35 per cent. free sulphur, 15 per cent. silica, and small quantities of molybdenum, nickel, alumina, lime, etc. It occurs at Minasragra, 30 miles west of Cerro de Pasco, Peru, in the Andes at altitude of nearly 16,000 ft. in a lens-like vein having a dip of about 63°, thickness of from 3 to 6 ft. and length of 300 ft. The patronite ore is free-burning, is easily calcined and loses 45 per cent. of its weight in the process. The calcined ore contains about 52 per cent. vanadium oxide, (V_2O_5). The clays in the vicinity of the patronite vein are impregnated with solutions of oxidized vanadium. Possibly 5000 tons of these materials have been extracted.

Carnotite is a complex vanadium and uranium mineral of variable content, the usual commercial quality having about 4 per cent. vanadium oxide and 2 per cent. uranium oxide. In the western United States it occurs as a canary-yellow ocherous pigment impregnating sandstone and is to a large extent made up of calcium and barium compounds admixed with an amorphous silicate containing vanadium in the trivalent state, probably replacing aluminum. The strata are flat, and vary in thickness from 1 to 4 ft. The color changes in proportion to the content of the vanadium or of the uranium. In ores containing 14 per cent. vanadium oxide and 3 per cent. uranium oxide (U_3O_8) with 3 to 4 per cent. sulphur and some carbonaceous matter, the color is a dark olive

[1] *Eng. and Min. Journ.*, June 1, 1912.

green, with yellowish specks, while in ores free from carbonaceous matter and high in calcium, the vanadium content is appreciably higher and the color is a brick-red, somewhat like soft iron oxide, but interspersed with minute particles of the yellow carnotite, or containing thin layers of the blended vanadium carnotite of duller yellow hue. The principal deposits operated are in the Paradox Valley, Montrose County, Colo., and are over 50 miles in length and 12 miles in width. The occurrence is in the La Plata sandstone of Jurassic age. Carnotite is found also in San Miguel, Mesa, Routt and Rio Blanco Counties, Colo.; in Grand and Emery Counties, Utah; in South Australia, Portugal and Turkestan. The shipments from Paradox Valley have totaled probably 2000 tons.

Roscoelite, which is a muscovite in which vanadium has partly replaced aluminum, or a closely related compound, occurs in blanket deposits in the Lower San Miguel mining district of San Miguel County, Colo., as a replacement of the original calcareous cement. The deposits are persistent, thick and easily worked, and contain an average of about 3 per cent. vanadium oxide. The ore is milled close to the mines and the production so far has been probably 10,000 tons.

Vanadinite, $Pb_5V_3O_{12}Cl$, occurs in a great variety of isomorphous mixtures, indicative of hydro-chemical reactions. Endlichite is a mixture of the arsenic and vanadium compounds; mottramite, a vanadate of copper and lead, is found at Alderley Edge, England, as an impregnation of the Keuper sandstone. A further variety is descloizite, close kin to ramirite found in many parts of Mexico. The vanadates occur principally in metalliferous veins, and their formation is attributed to percolating vanadiferous waters acting on other compounds, usually the compounds of lead. Several deposits occur near Cutter, New Mexico, associated with descloizite, lead molybdate, fluorspar, galena, barite and copper minerals. The average ore contains about 1 per cent. vanadium oxide and 5 to 7 per cent. lead. Considerable difficulty is experienced in mechanically concentrating the ore. Deposits also occur at Bayard, Hillsboro and Cook's Peak, while a somewhat higher grade vanadinite occurs near Tucson, Ariz. In Mexico, vanadinite is found near Villa Rasales, Chihuahua, also at Coyame, Iturbide, and at Zimapan, Hidalgo; at Cardonal, in San Luis Potosi, at Charcas in the Sierra de Coronados, and at Catorce, as well as in Ojocaliente, near Berriozabal, Zacatecas. The vanadinite from Chihuahua is fairly pure and easily concentrated to 18 per cent. vanadium oxide. The ore from Zacatecas contains over 4 per cent. vanadium oxide and 64 per cent. lead but is frequently associated with arsenic. Cuprodescloizite, or ramirite, is probably akin to chileite, which contains 13.5 per cent. vanadium oxide, 55 per cent. lead oxide, 14.6 per cent. copper oxide, and 4.6 per cent. arsenic oxide,

2 per cent. sand and clay, the remainder being lime, iron, alumina, etc It occurs in Guanajuato at San Luis de la Paz, in San Luis Potosi, Matehuala, and Charcas. A variety that occurs near Zacatecas contains the vanadium in association with wulfenite. In the descloizite as obtained from the River Plate district, Argentina, manganese replaces the copper and the vanadium oxide content is 22 per cent., with 11 per cent. combined manganese oxides. Many varieties of vanadates occur in the silver-lead mines of Cordoba, some of the mines being extremely rich but flooded by underground streams. The lead ores of Northern Spain contains from 4 to 5 per cent. vanadium oxide and by crude hand-dressing are concentrated to 12 per cent. vanadium oxide, with 35 per cent. lead, 13 per cent. ferrous oxide, 35 per cent. siliceous matter and about 3 1/2 oz. silver to the ton.

Asphaltite is an extremely soft, friable, lustrous substance, of very low specific gravity, containing about 80 per cent. hydrocarbon; it breaks into minute squares or cubes and burns with a long, smoky flame, yielding much tar. The vanadium content is quite uncertain, ashes from coal from the same deposit giving from 5 per cent. to 50 per cent. vanadium oxide. It occurs near Page, Okla., and at Palisade, Nev., a sample from the latter place giving 1.39 per cent. ash and 25 per cent. vanadium oxide in the ash. There are many asphaltite deposits in Peru, those of the Yauli, being best known. The surface consists of three layers which at a slight distance unite and form a single layer from 7 to 9 millimeters thick, with average content of 1 per cent. vanadium oxide, which being burned to an ash yields from 25 to 40 per cent. vanadium oxide. Other mines are situated near Huanuco, a sample ton of ashes received recently giving 17 per cent. vanadium oxide. A lignite coal of this nature also occurs near San Rafael, Argentina. An ordinary steaming coal from Rockvale, Colo., gave 1.5 per cent. ash with 27 per cent. vanadium oxide in the ash. In operating the Jodandy carnotite mine in Paradox Valley, a small pocket was opened in the solid sandstone that was filled with pea-size pieces of red calcium vanadate, yellow siliceous carnotite, and vanadiferous coal. In the Minasragra deposit, the patronite is enveloped with veins of quisqueite and natural coke. The elevation of the Yauli deposits is about the same as those at Minasragra.

Vanadium Mining

Colorado.—The Primos Chemical Co., Primos, Pa., operated its plant at Newmire throughout 1911, at a daily capacity of 25 tons of vanadiferous sandstone. In addition, ore of the richer grades was shipped

regularly to Europe, the total freight charges, via Pueblo and Galveston, being $12 per ton. Storage bins were built at New Placerville for the convenience of the several small operators in the Paradox valley, who haul out their product by wagon. It is reported that 72 carloads, valued at $432,000, were shipped out of Paradox valley during 1911. Discoveries of high-grade ore were reported from Magnolia and Eldora, Boulder County.

New Mexico.—In the Caballo Mountains, just north of Palomas pass, veins of galena are prominent, with which are associated molybdate and vanadate of lead. The most extensive developments of vanadium ores have occurred on the Red Top, White Swan, Benjamin and Dewey claims. The Vanadium Mines Co., of Pittsburg, Pa., has been experimenting in an old lead mill of the vicinity, and expects to build a new mill for the concentration of the ore. The concentrate, averaging 8.63 per cent. Vd and 59 per cent. Pb, is hauled 12 miles to Cutter, on the Santa Fe railway, where the company has established a plant for separating the vanadium from the lead. The concentrator has a capacity of 150 tons per day, while the treatment plant can produce 6 tons of lead sulphate and 1500 lb. vanadic oxide per day. The former is sold, but the vanadic oxide is shipped to the company's plant at Rankin, Pa., where it is converted into ferro-vanadium. The first shipment of finished product was made from Rankin in March, 1911. The ore, as mined, is very soft and brittle, and most of the difficulty has been encountered in the milling process; the metallurgical process was devised by S. Diescher & Sons, of Pittsburg.

At Georgetown, Grant County, leasers have lately been working over the dumps from the once flourishing lead-silver mines and recovering a product containing 7.5 per cent. Vd, 40 per cent. Pb, and 300 oz. Ag, which they have been shipping to El Paso; they get no pay for the vanadium.

Utah.—A good deal of activity occurred during 1911 in the Vanadium deposits of eastern Utah. San Rafael, in Emery County, and Cisco, in Grand County are the centers of interest. A shipment of 21 tons, carrying 7.3 per cent. Vd, was made to Liverpool from the former district, via Green River, and 30 tons were shipped from Cisco by the Utah Rare Metals Mining Co., in the course of the year. It is reported that F. J. Cameron, of Philadelphia, will build an extraction plant in the vicinity.

Peru.[1]—The mines of Minasragra, in the province of Pasco, belonging to the American Vanadium Company, represented in Lima by Eulogio Fernandini, produce sulphides and oxides of the metal and con-

[1] By Eduardo Higginson, Peruvian Consul General at New York.

stitute the principal source of that substance in Peru. There are also deposits of asphalt and vanadiferous coal, the ashes of which sometimes contain as much as 40 per cent. of vanadic acid. Deposits of this character are worked in the provinces of Yauli, Jauja and Canta, among them the Lucha mine, in Huari, near Oroya, the property of Alejandro Aza, of Lima. The most recently worked deposits of this class are those in Yautac, between the provinces of Canta and Yauli, where Augusto Gildermeister, Alberto Noriega, Fausto Valdeavellano and Carlos I. Lisson, of Lima, and the Praeli brothers, of Tarma, have different holdings. The coal here contains from 1 to 1 1/2 per cent. of vanadic acid, which is concentrated in its entirety in the ashes. In the provinces of Huancayo and Dos de Mayo, there are several such deposits which are not worked, although those of the last-named district have been acquired by the firm of C. Weiss & Co., of Lima, for mining purposes.

During 1911, the mines at Minasragra shipped monthly 200 to 400 short tons of calcined patronite, carrying 25 per cent. V_2O_5, all of which went to the United States. The output of 1907 was valued at $24,330 and of 1908 at $340,650; in 1909 the output amounted to 1749 metric tons and in 1910 to 3130 tons.

Technology of Vanadium

Bleeker Process.—U. S. patent No. 1,015,469 was granted to W. F. Bleeker, of Boulder, Colo., for the following process: The first steps consist in grinding the material fine, roasting for at least 15 minutes at a heat a little above redness, and fluxing with sodium chloride and a fixed alkali, such as potassium or sodium hydroxide, or sodium or potassium carbonate, either during or after the roasting process. This converts the vanadic acid into soluble form. The fluxed ore is then leached, either with or without agitation, with water, and the solution of alkaline vanadates removed. The residue is then leached with a dilute mineral acid (such as hydrochloric), which leaves the ore practically free from vanadium. It is necessary to produce a neutral solution, which is then done by mixing the aqueous and the acid extracts, although either of these solutions may be taken and neutralized separately. The final step consists in precipitating the vanadium from the neutral solution by adding a soluble salt of copper, iron or lead. While other insoluble vanadates may also be produced in this way, those mentioned are the most important.

Koenig Process.—U. S. patent No. 986,180 was granted to G. A. Koenig, of Houghton, Mich., for the following process, applicable to ores containing roscoelite or similar minerals. The crushed ore is put into

a tight, rotating drum, and subjected to treatment with 20 per cent. solution of sulphuric, hydrochloric or other acid, at a temperature of 200° C. and under pressure of 225 lb. per square inch. This decomposes and dissolves the mineral in a few hours. The solution is then filtered and evaporated to a mushy state; the salts are then heated to bright red in retort or muffle, to expel the remaining acid. The resulting mixture of oxides and sulphates is then mixed with sodium carbonate and roasted at red heat in an oxidizing flame, with or without addition of oxidizing agents. The roasted mass is next disintegrated with boiling water, and while the solution is still in the boiler is treated with carbon dioxide, which precipitates aluminum hydroxide.

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ZINC

The zinc industry of the United States in 1911 was profitable both to miners and smelters, to the former because the price for spelter, and consequently of ore, was materially above what is considered to be a normal average; and to the latter because the margin between the value of zinc in crude and finished form was larger than usual. The latter condition was due, of course, to the bountiful supply of ore; because of this same fact the joy of the producers of ore was not unalloyed.

The total production of spelter in 1911 was 295,836 tons, of which 9030 tons is estimated to have been derived from dross, scrap, etc. (against 7793 tons in 1910), and the remainder from ore. This does not include the production of spelter by concerns which treat dross and scrap only, whose output is estimated at 13,500 tons in 1911 as compared with 11,350 in 1910.

In addition to this production there was an output of 254 tons of zinc dust in 1911. The importation of zinc dust in 1911 was 1713 tons, 296 tons being entered through Pacific Coast ports and the remainder through other ports.

In 1910 there were produced 3398 tons of spelter in bond. In 1911 the production in this way was 9598 tons. At the end of the year no bonded spelter was reported in stock. The zinc ore received by smelters was 28,596 tons from Mexico and 2356 tons from Canada in 1911. The imports reported by the Government were 69,800 short tons.

Along with the increase in the production of spelter in 1911 there was naturally an increase in the production of ore. The striking features of the ore statistics are the decrease in the production in the Joplin district and the increases in Colorado, Wisconsin and Montana. The extraordinary increase in the case of Colorado is due to the large shipment of calamine from Leadville. This being of very low grade it does not make as much new spelter as would be indicated upon the face of the returns. The increased output of Montana is of course due to Butte. Further increases in the production of Butte in 1912 and 1913 are to be expected. Most interesting, perhaps, is the continued increase on the part of Wisconsin, the ore of which state is of high grade.

The statistics of ore production, Table II, refer only to the ore received by the smelters, and do not include the ore received by manufacturers of oxides.

In 1911 there were seven zinc smelters who reported the production of sulphuric acid from blende roasting. The total amount of their output was 252,192 tons against 202,000 tons in 1910. The figures of sulphuric acid produced are reduced to the basis of 60° B. Six of these smelters produced 115,310 tons of spelter against 100,791 tons in 1910. They received 71,168 tons of blende from Wisconsin and 107,995 tons from the Joplin district, besides less supplies from the Rocky Mountains. It is noteworthy that these works buy nearly the whole output of blende in Wisconsin and about 40 per cent. of the total production of zinc ore in the Joplin district. A noteworthy feature is the increasing use of western ore by some smelters making sulphuric acid. Several of the smelters in this group, however, continue to use only Missouri and Wisconsin ore.

I. PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1903.	1904.	1905.	1906.	1907.	1908.	1909.	1910.(c)	1911.(c)
Colorado	877	4,906	6,599	6,260	5,200	3,079	6,115	6,564	7,477
Illinois (a)	49,526	47,607	45,357	48,238	56,103	50,244	75,229	79,570	88,681
Kansas	87,406	103,721	114,948	129,741	133,561	99,136	103,390	105,697	106,173
Missouri	9,894	12,056	11,800	11,088	11,594	10,196	8,418	6,485	
Oklahoma	.....	.....	.....	.....	5,094	14,867	28,840	34,760	46,333
South and East (b)	10,799	13,513	23,044	30,167	38,060	32,989	44,470	43,989	47,172
Total tons of 2000lb.	158,502	181,803	201,748	225,494	249,612	210,511	266,462	277,065	295,836
Total tons of 2240lb.	141,520	162,324	180,132	201,343	222,868	187,776	237,913	247,364	264,139
Total metric tons	143,792	164,921	183,014	204,548	226,398	190,933	241,730	251,348	268,378

(a) Up to 1903, inclusive, includes also the production of Indiana. (b) New Jersey, Pennsylvania and Virginia, and (since 1903) West Virginia. (c) From *Eng. and Min. Journ.*

II. PRODUCTION OF ZINC ORE IN THE UNITED STATES.
(In tons of 2000 lb.)

State.	1905.	1906.	1907.	1908.	1909.	1910.	1911.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons. (g)	Tons.(g)
Arizona	.....	.....	.....	.....	7,908	7,568	6,395
Arkansas	2,200	4,200	4,088	2,582	(d) 320	190	860
Colorado	105,500	114,000	142,510	85,052	90,288	77,065	158,528
Idaho	1,700	2,150	11,847	1,558	2,784	10,248	9,667
Kentucky	(b) 414	975	1,005	341	(d) 94	179	575
Mo.-Kan	(a)258,500	(a)280,260	297,126	273,420	304,581	289,913	268,500
Montana	2,000	4,900	1,218	2,783	12,037	33,514	56,593
Nevada	*Nil.*	7,080	4,593	1,445	4,609	4,915	5,666
New Mexico	17,800	30,000	4,281	2,290	19,163	15,959	10,184
New Jersey	(b)361,829	404,690	368,710	399,232	479,699	(f)	(f)
Oklahoma	.....	.....	3,240	9,300	17,089	1,640	8,750
Tennessee	.....	.....	.....	.....	1,710	2,775	3,439
Utah	9,265	10,700	9,043	709	18,130	27,318	19,933
Wisconsin	32,690	42,130	53,011	58,135	69,000	51,383	71,565
Others	(c) 3,800	(c) 850	(c) 2,241	(c)1,530	572	46,905	48,650
Totals	795,698	905,175	902,923	838,377	1,027,984	569,572	669,305

(a) Production of Joplin district, plus output of southeastern Missouri, the latter as reported by the State mine inspector. (b) Report of State Geologist; crude ore. (c) Arizona, Nevada, Illinois, Iowa, Tennessee and Virginia. (d) These figures may be a little too low, there being a possibility that some ore originating in these States has been credited to other States, especially Missouri and Kansas. (e) Tennessee, Arizona and California. (f) Included in others. (g) The statistics for 1910 and 1911 include only the zinc ore received by producers of spelter, wherefore some apparently peculiar changes from the figures for previous years, which include the receipts by oxide manufacturers. Reports missing from three small smelters. In 1911 California produced 3754 tons.

DELIVERIES OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

	1905.	1906.	1907.	1908.	1909.	1910.	1911.
Stock, Jan 1....	6,500	4,000	4,550	32,883	25,000	11,500	23,000
Production.....	201,748	225,494	249,612	210,511	282,459	288,415	309,336
Imports.....	521	2,203	1,778	881	9,670	3,452	1,637
Total supply.	208,769	231,697	255,940	244,275	317,129	303,367	333,973
Exports(a).....	5,515	4,670	563	2,640	2,900	8,457	18,142
Stock, Dec. 31..	4,000	4,550	32,883	25,000	11,500	23,000	9,323
Deliveries.....	199,254	222,477	222,494	216,635	302,729	271,892	316,508

(a) Includes domestic and foreign.

EXPORTS OF ZINC ORE AND ZINC OXIDE FROM THE UNITED STATES. (a)

Year.	Ore.			Oxide.		
	Short tons.	Value.	Value per ton.	Short tons.	Value.	Value per ton.
1898....	11,782	$299,970	$25.50	3,925	$252,194	$64.25
1899....	28,221	725,944	25.90	5,343	366,598	68.61
1900....	42,062	1,134,663	26.98	5,656	496,380	87.76
1901....	44,146	1,167,684	26.45	4,561	393,259	86.22
1902....	55,733	1,449,104	26.00	5,358	433,722	80.93
1903....	39,411	987,000	25.04	7,215	578,215	80.14
1904....	35,911	905,782	25.22	8,157	628,494	77.05
1905....	30,946	848,451	27.41	11,280	810,203	71.83
1906....	27,720	733,300	26.45	15,578	1,149,297	73.78
1907....	20,352	579,490	28.47	13,256	1,069,924	80.71
1908....	26,108	877,745	33.60	12,008	845,070	70.37
1909....	12,456	412,300	33.10	14,846	1,026,377	69.14
1910....	19,711	649,425	32.95	13,166	943,968	71.69
1911....	14,573	642,884	44.12	14,618	1,051,311	71.92

(a) In addition to the exports of ore, 15,887 short tons of zinc dross (galvanizers' waste) were exported in 1906, 9593 short tons in 1907, 8405 short tons in 1908, 7069 short tons in 1909, 4728 short tons in 1910, and 4246 tons in 1911.

EXPORTS OF DOMESTIC SPELTER FROM THE UNITED STATES. (a)

Year.	Plates, Sheets, Pigs and Bars.		Wares.	Total Value.
	Short Tons.	Value.	Value.	
1897.....	14,245	$1,356,538	$71,021	$1,743,049
1898.....	10,499	1,033,959	138,165	1,724,188
1899.....	6,755	742,521	143,232	1,978,295
1900.....	22,411	2,217,963	99,288	2,317,251
1901.....	3,390	228,906	82,046	310,952
1902.....	3,237	300,557	114,197	414,754
1903.....	1,521	163,379	71,354	234,733
1904.....	10,073	1,094,490	117,957	1,212,447
1905.....	5,516	682,254	159,995	842,249
1906.....	4,670	583,526	204,269	787,795
1907.....	563	75,526	186,283	261,397
1908.....	2,640	250,254	88,485	338,739
1909.....	2,566	263,010	69,751	332,761
1910.....	3,990	426,500	101,336	527,836
1911.....	6,872	810,099	134,518	944,617

(a) There is also a comparatively insignificant re-export of foreign-made spelter and zinc wares.

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES.
(In pounds.)

Year.	Sheets, Blocks, Pigs and Old.		Manufactures.	Total Value.	Oxide.	
	Amount.	Value.			Dry.	In Oil.
1897..	2,557,341	$95,883	$19,431	$115,314	5,564,753	502,357
1898..	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899..	2,985,463	151,956	14,800	166,756	3,012,709	41,699
1900..	2,013,196	97,772	36,836	134,608	2,618,808	38,706
1901..	775,881	30,920	42,643	73,563	3,199,778	128,198
1902..	1,238,091	46,713	37,191	83,904	3,271,385	163,081
1903..	728,614	30,900	18,938	49,838	3,487,042	166,034
1904..	933,474	44,326	11,918	56,244	2,585,661	224,244
1905..	1,042,081	51,052	12,390	63,442	3,436,367	342,944
1906..	4,407,481	253,310	17,385	270,695	4,191,476	292,538
1907..	3,555,890	210,322	16,282	226,604	5,311,318	362,418
1908..	1,762,627	85,885	7,474	93,359	4,635,101	210,166
1909..	19,340,029	826,588	19,176	845,764	6,119,328	535,024
1910..	6,904,389	289,689	27,475	317,164	6,133,840	336,257
1911..	3,275,340	170,634	124,983	295,617	(a)5,561,016	.....

(a) Both dry and in oil.

Smelting Capacity

In spite of the fears that were entertained in the latter part of 1910 that there would be a deficiency in smelting capacity in 1911, there appears to have been actually an increase.

At the end of 1910 the smelters reported a total of 91,291 retorts, of which 9740 (Lanyon Zinc Co.) were about to be permanently dismantled, leaving 81,551 available, of which 3208 were in works that were idle.

At the end of 1911 the total number of retorts was 87,800, of which 4744 were in works then idle, one of which was in operation during a part of 1911, and another of which was being made ready for operation in January, 1912.

The works that went permanently out of use in 1911 were those at Pulaski, Va., and Bethlehem, Penn. New works that went into operation were those of the Bartlesville Zinc Co. (five furnaces, 3200 retorts) at Collinsville, Okla.; Tulsa Fuel & Manufacturing Co. (six furnaces, 3320 retorts) at Collinsville, Okla.; and Grasselli Chemical Co. (six furnaces, 3456 retorts) at Meadowbrook, W. Va.

Additions to smelting capacity were made by the Bartlesville Zinc Co., at Bartlesville, Okla., Lanyon-Starr Smelting Co., National Zinc Co. and United Zinc & Chemical Company.

At the end of 1911, among operating smelteries, 5120 retorts were under construction at Collinsville, Okla., 2880 at Palmerton, Penn., 1800 at Danville, Ill., and 876 at Lasalle, Ill., a total of 10,676 comprised in 23 furnaces. New works under construction were those of the Granby Mining & Smelting Co., at East St. Louis, Ill., and of the American Zinc, Lead & Smelting Co. and the Robert Lanyon Zinc & Acid Co., at Hillsboro, Ill., with an aggregate of 8480 retorts projected.

III. ZINC SMELTING CAPACITY OF THE UNITED STATES AT THE END OF 1911.

Name.	Location.	Furnaces.	Retorts.
American Zinc, Lead & Smelting Co	Deering, Kan.	6	3,840
American Zinc, Lead & Smelting Co	Caney, Kan.	6	3,648
Bartlesville Zinc Co.	Bartlesville, Okla.	9	5,184
Bartlesville Zinc Co.	Collinsville, Okla.	5	3,200
Chanute Zinc Co.	Chanute, Kan.	4	1,280
Beer, Sondheimer & Co.	Altoona, Kan.	6	3,840
L. Vogelstein & Co.	Gas City, Kan.	4	(a) 2,560
L. Vogelstein & Co.	La Harpe, Kan.	3	1,856
Collinsville Zinc Co.	Collinsville, Ill.	6	(b) 1,536
Edgar Zinc Co.	St. Louis, Mo.	9	2,000
Edgar Zinc Co.	Cherryvale, Kan.	24	4,800
Granby Mining & Smelting Co.	Neodesha, Kan.	6	3,840
Grasselli Chemical Co.	Clarksburg, W. Va.	10	5,760
Grasselli Chemical Co.	Meadowbrook, W. Va.	6	3,456
Hegeler Bros.	Danville, Ill.	2	1,800
Illinois Zinc Co.	Peru, Ill.	7	4,640
Lanyon-Starr Smelting Co.	Bartlesville, Okla.	7	3,456
Matthiesen & Hegeler	Lasalle, Ill.	5	4,380
Mineral Point Zinc Co.	Depue, Ill.	6	4,520
National Zinc Co.	Bartlesville, Okla.	7	4,256
Nevada Zinc Co.	Nevada, Mo.	3	(c) 648
New Jersey Zinc Co.	Palmerton, Penn.	12	2,892
Pittsburg Zinc Co.	Pittsburg, Kan.	2	448
Prime Western Spelter Co.	Gas City, Kan.	9	5,384
Sandoval Zinc Co.	Sandoval, Ill.	4	896
Tulsa Fuel and Manufacturing Co.	Collinsville, Okla.	6	3,320
United States Zinc Co.	Pueblo, Colo.	7	1,680
United Zinc and Chemical Co.	Springfield, Ill.		3,680
		..	87,800

(a) Inactive. (b) Inactive during latter part of year. (c) Not operating in 1911, but preparing for production in 1912.

PRODUCTION OF ZINC IN EUROPE AND AMERICA. (a)
(In metric tons.)

Year.	Austria.	Belgium.	France.	Germany.	Holland.	Italy.	Russia.	Spain.	United Kingdom.	United States.	Totals.
1897 ..	6,236	116,067	38,067	150,739	6,600	250	5,868	6,244	23,805	91,070	444,946
1898 ..	7,302	119,067	37,155	154,867	6,700	250	5,664	6,031	28,387	103,514	468,937
1899 ..	7,192	122,843	39,274	153,155	6,235	251	6,331	6,184	32,322	117,644	491,331
1900 ..	6,742	119,315	36,305	155,799	6,845	547	5,963	5,611	30,207	111,794	465,438
1901 ..	7,558	127,170	37,600	166,283	7,855	511	6,090	5,354	29,877	127,751	516,049
1902 ..	8,309	124,780	36,300	174,927	9,910	485	8,280	5,569	40,244	143,552	552,356
1903 ..	8,949	131,740	37,416	182,548	11,515	126	9,901	5,134	44,110	143,792	569,971
1904 ..	9,159	137,323	41,600	193,058	12,895	189	10,607	5,887	46,218	164,921	621,857
1905 ..	9,204	142,555	43,200	198,208	13,550	5	7,520	6,184	50,125	183,014	653,565
1906 ..	10,711	148,035	46,536	205,691	14,650	69	9,610	6,209	52,587	204,548	698,646
1907 ..	11,208	154,492	(c)49,733	208,195	14,990	88	10,409	(c)6,000	55,595	226,398	737,108
1908 ..	12,770	161,940	47,880	216,490	17,257	(d)100	9,960	6,357	54,473	190,933	718,160
1909 ..	12,638	167,100	(c)49,718	219,766	19,548	(b)	7,949	(c)6,400	59,350	241,730	784,199
1910(f)	13,305	172,578	(c)51,527	227,754	20,975	(b)	8,128	(c)6,604	(e)63,587	251,348	815,806
1911 ..	13,925	195,102	55,170	235,776	22,734	(b)	9,652	9,053	(e)67,907	268,378	877,697

(a) From the official statistics of the various Governments except 1906 to 1911 inclusive, for which years the figures reported to Henry R. Merton & Co. have been used where the official statistics were unavailable. In addition to the production reported in this table, Australia produced 286 long tons in 1903, 299 in 1904, 544 in 1905, 1008 in 1906, 980 in 1907, and 1086 in 1908. (b) Included in Austria. (c) An approximate separation of the total which is reported for "France and Spain." (d) Estimated. (e) Includes 508 and 1000 tons from Australia. (f) *Eng. and Min. Journ.*

IMPORTS OF ZINC ORE INTO THE UNITED STATES.
(In tons of 2000 lb.)

Source.	1906.	1907.	1908.	1909.	1910.	1911.
British Columbia..	600	1,157	6,157	10,024	11,795	2,356
Mexico.....	(a)88,900	(a)108,800	66,383	106,245	29,198	28,596
Totals.....	89,500	109,957	72,540	116,269	40,993	30,952

(a) The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore which for statistical purposes has been reduced to the zinc ore equivalent. This table is based on reports from the smelters of the ore received by them from these countries.

Zinc Dust (By William J. Dwyer[1]).—Notwithstanding the facts that prices rose to a high level and that a duty of 1 3/8 cents per pound is now collected on importations of zinc dust, the consumption of this product in the United States in 1911 revealed a marked increase. The total consumption was somewhat in excess of 2500 tons.

The imports for the fiscal year totaled 2135 short tons valued at $204,836. The average import price per pound, weight and package ex-vessel, port of shipment, was $0.048. As the government did not separately classify the imports of zinc dust previous to Aug. 9, 1909, figures covering a complete previous fiscal year for comparison are not available. According to the "Statistik des Deutschen Reichs, Auswärtiger Händel," the shipments from Germany, the principal source of supply, in 1910 to the United States amounted to 1492 tons valued at $136,749.

Zinc dust was recovered in commercial quantities in the United States in 1911 by the United States Zinc Co., at Pueblo, Colo., and the Grasselli Chemical Co., at Clarksburg, W. Va. The production of the Pueblo smeltery early in the year reached 1000 lb. per day, but later on the output was considerably reduced. The production of the Grasselli company was probably much in excess of its competitor. The recovery of this product, coincident with the broadening markets, will probably receive more attention hereafter upon the part of zinc smelters in this country.

The production of zinc dust in Germany amounts to approximately 10,000 metric tons annually. Upper Silesia produces the bulk, but Westphalia and the Rheinland are also large producers. Belgium is the next important shipper to this market. In September about 17 tons arrived at New York from Austria. A large majority of the American imports came through the port of Copenhagen last year but this material was undoubtedly of Silesian origin. The exports from Germany to all parts during the first three-quarters of 1911 amounted to 2338 metric tons as against 2234 metric tons during the corresponding period of 1910.

[1] *E. and M. J.*, Jan. 13, 1912.

Slightly more than 25 per cent. of the American import business is done through the port of New York. San Francisco is a large importer.

The principal use for zinc dust is now as a substitute for zinc shavings in cyaniding. Eastern importers, however, find their local markets with sherardizers, dyers and bleachers. The consumption of the product in sherardizing is increasing steadily.

The American Spelter Market in 1911.—Among the commercial metals, spelter continues to be the only one whereof the market is not dominated by one paramount factor. The forces of supply and demand have free sway. Fluctuations, therefore, are apt to be frequent, indexing as it were changes in general sentiment and the vicissitudes of trade at large. The market is still gaged by the prices ruling in St. Louis, where quotations hovered around 5.35 cents at the beginning of the year. The ease in the ore market, and the consequently favorable smelting margins, caused smelters to realize freely. The decline in May was hardly justified. The free selling of first hands was due to a desire to clinch not only a satisfactory current smelting profit, but also to dispose of stocks of spelter at the zinc works. This opportunity continued to exist upon an increasingly attractive scale throughout the remainder of the year.

Supplies in Europe had been reduced to a minimum and consumption had reached a maximum rate. Quotations in London climbed until they reached the export parity. The domestic market, not having a surplus to spare, was driven ahead. The advance was almost uninterrupted on both sides of the Atlantic until the market crossed 6 cents during the closing days of October. Reports of large exports were current, but the statistics will probably show that the bulk of the quantities sent out of the country by western smelters consisted of bonded spelter obtained from Mexican or British Columbia ore which had not been retained for domestic consumption since the Payne-Aldrich tariff became effective. Nevertheless, the foreign syndicate became frightened, and to keep out American spelter lowered its prices. This action had no effect on the home market, which proved its independence by a further advance during November and into the first days of December, when the market had climbed to 6 1/4 to 6 1/2 cents. Coincidentally there prevailed an acute scarcity of spot spelter, for which premiums were realized by the few fortunate holders. By the middle of December, the tension was relieved. Spelter which had been held back from the market was offered for sale, and as the immediate demand had been satisfied, the market experienced a sudden decline to about 6 cents, and closes uncertain at 6.10 to 6.20 cents, St. Louis.

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN ST. LOUIS.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1903....	4.688	4.681	5.174	5.375	5.469	5.537	5.507	5.550	5.514	5.350	4.886	4.556	5.191
1904....	4.673	4.717	4.841	5.038	4.853	4.596	4.723	4.716	4.896	5.033	5.363	5.720	4.931
1905....	6.032	5.989	5.917	5.667	5.284	5.040	5.247	5.556	5.737	5.934	5.984	6.374	5.730
1906....	6.337	5.924	6.056	5.931	5.846	5.948	5.856	5.878	6.056	6.070	6.225	6.443	6.048
1907....	6.582	6.664	6.687	6.535	6.291	6.269	5.922	5.551	5.086	5.280	4.775	4.104	5.812
1908....	4.363	4.638	4.527	4.495	4.458	4.393	4.338	4.556	4.619	4.651	4.909	4.987	4.578
1909....	4.991	4.739	4.607	4.815	4.974	5.252	5.252	5.579	5.646	6.043	6.231	6.099	5.352
1910(*a*)	5.951	5.419	5.487	5.289	5.041	4.978	5.002	5.129	5.364	5.478	5.826	5.474	5.370
1911(*a*)	5.302	5.368	5.413	5.249	5.198	5.370	5.545	5.803	5.719	5.951	6.223	6.151	5.608

(*a*) From *Eng. and Min. Journ.*

AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900....	4.65	4.64	4.60	4.73	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901....	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.07
1902....	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903....	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904....	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905....	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882
1906....	6.487	6.075	6.209	6.078	5.997	6.096	6.006	6.027	6.216	6.222	6.375	6.593	6.198
1907....	6.732	6.814	6.837	6.685	6.441	6.419	6.072	5.701	5.236	5.430	4.925	4.254	5.962
1908....	4.518	4.788	4.665	4.645	4.608	4.543	4.485	4.702	4.769	4.801	5.059	5.137	4.726
1909....	5.141	4.889	4.757	4.965	5.124	5.402	5.402	5.729	5.796	6.199	6.381	6.249	5.503
1910(*a*)	6.101	5.569	5.637	5.439	5.191	5.128	5.152	5.279	5.514	5.628	5.976	5.624	5.520
1911(*a*)	5.452	5.518	5.563	5.399	5.348	5.520	5.695	5.953	5.869	6.102	6.380	6.301	5.758

(*a*) From *Eng. and Min. Journ.*

AVERAGE MONTHLY PRICE OF SPELTER IN LONDON.
(Pounds sterling per ton of 2240 lb. of good ordinary brands.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£.	£.	£.	£.	£.	£.	£.	£.	£.	£.	£.	£.	£.
1905....	25.063	24.594	23.825	23.813	23.594	23.875	23.938	24.675	26.375	28.225	28.500	28.719	25.433
1906....	28.225	25.844	24.563	25.781	27.000	27.728	26.800	26.938	27.563	28.075	27.787	27.938	27.020
1907....	27.125	25.938	26.094	25.900	25.563	25.469	23.850	21.969	21.050	21.781	21.438	20.075	23.771
1908....	20.563	20.875	21.075	21.344	19.906	19.000	19.031	19.350	19.563	19.750	20.875	20.625	20.163
1909....	21.425	21.563	21.438	21.531	21.975	22.000	21.969	22.125	22.906	23.200	23.188	23.094	22.185
1910(*a*).	23.350	23.188	23.031	22.469	22.100	22.094	22.406	22.800	23.165	23.900	24.083	24.019	23.050
1911(*a*).	23.887	23.276	23.016	23.743	24.375	24.612	25.006	26.801	27.750	27.256	26.795	26.849	25.281

(*a*) From *Eng. and Min. Journ.*

ZINC MINING IN THE UNITED STATES

California.—The first important production came from this State in 1911.

Colorado (By G. E. Collins).—While the lead production of the state continued to decrease, the production of zinc has increased, owing to the important discoveries of zinc carbonate at Leadville. These discoveries were made during 1910 but have only now begun fully to show

their effect on the production. No great extension has yet been added to the productive area, which still centers in the Wolf Tone, Maid of Erin, Morning Star and Evening Star. So far, there is only a market for such ores when they contain over 30 per cent. of zinc. It is, however, certain that Leadville has great resources in zinc-carbonate ores of much lower grade, running from 15 to 30 per cent., which at present are awaiting some method of utilization.

The Smuggler Union mill is being remodelled to save zinc.

The oxidized zinc ores at Leadville are discussed by George O. Argall.[1] Zinc ores in various parts of Colorado are also described in *E. and M. J.*, Sept. 2, 1911; F. L. Clerc[2] gives plans for the utilization of the new zinc ores of Leadville.

Kansas (By A. L. Morford[3]).—Under the plan of Manager Playter the Boston Land and Mining Company at Galena, Kansas, furnishes all equipment for mining and mills the "dirt" for half the yield. Five subleasing companies are now hauling dirt to the plant and the first turnin of blende was made recently. Near the mill site a 35-ft. face of zinc-bearing dirt has been opened above the 90-ft. level and development work shows that the length of the face is 600 ft. All shafts are within 100 yd. of the mill, which has made the supplying of steam from a central boiler plant to the five hoists an easy matter. Within a week from the time of starting, the plant was placed on a double-shift basis, this being made possible by the installation of electric-lighting apparatus at the gas-engine pump station.

This is the first time such a plan has been tried by a land-owning company and it is working perfectly. Other companies have begun work at different places along the ore belt, which has been traced for three-quaters of a mile across the company's holdings. Mill tests show a recovery of 7 per cent. blende which assays 61 per cent. metallic zinc with no iron.

Idaho (Coeur d'Alenes).—The Federal Mining & Smelting Co., from the Macquisten-tube installation, at the Morning plant, produced over a million pounds of zinc and the Success Mining Co. operated several months during 1911, with a heavy output.

Much development was made in the concentrator practice of the district. The new Hercules Mining Co. concentrator was put in commission at the town of Wallace and the Federal Mining & Smelting Co. made extensive improvements and extensions to its concentrator at the Morning mine, at Mullan. These improvements were along the line of finer grinding in ball mills and the use of Macquisten tubes for recover-

[1] *E. and M. J.*, Aug. 26, 1911, p. 399.
[2] Proceeding Colo. Scientific Society, and *Mining Science*, Feb. 22, 1912.
[3] *E. and M. J.*, Dec. 17, 1911.

ing zinc which heretofore went to waste in the fine tailings. The picking of clean shipping ore from the mill feed was extended and an elaborate picking plant was erected by the Federal Mining & Smelting Co. at the mouth of the Arizona tunnel, of their Wardner mine.

Missouri, Joplin District (By Jesse S. Zook[1]).—Shipments of both sulphide and silicate ores decreased. Missouri sustained a reduction of 19,000 tons zinc sulphide and 5000 tons zinc silicate: Kansas 4960 tons zinc sulphide and 80 tons zinc silicate; Oklahoma 4220 tons zinc sulphide and 24 tons zinc silicate. For final figures see the following table.

ZINC- AND LEAD-ORE PRODUCTION OF THE JOPLIN DISTRICT.
(In tons of 2000 lb.)

	Zinc Blende.		Zinc Silicate.		Lead Ore.		Total Value.
Webb City-Carterville	101,515	$4,138,578	.....	.....	25,591	$1,475,732	$5,614,310
Joplin	53,474	2,255,010	25,225	$505	7,175	409,603	2,665,118
Alba-Neck	16,138	704,477	.....	.....	263,280	15,279	719,756
Duenweg	11,770	485,239	2,164	56,245	1,150	65,735	607,219
Oronogo	10,085	393,672	.....	.....	1,377	75,631	469,303
Carthage	2,040	79,833	108	2,429	43	2,395	84,657
Sarcoxie	1,394	55,797	376	7,245	13	741	63,783
Carl Junction	4,650	209,801	.....	.....	16	876	210,677
Cave Springs	4,101	188,362	.....	.....	269	16,805	205,167
Jasper County	205,162	.....	2,673	.....	35,894	.....	10,639,990
Granby	4,212	146,326	9,356	236,403	647	33,219	415,948
Spurgeon	3,246	129,044	3,653	96,561	1,169	66,429	292,034
Jackson	5,752	236,592	.....	.....	913	52,667	289,259
Saginaw	.....	.....	375	11,470	116	6,579	18,049
Wentworth	288	11,591	.....	.....	.....	.....	11,591
Senaca	.....	.....	.....	.....	23	1,290	1,290
Newton County	13,499	.....	13,384	.....	2,870	.....	1,028,171
Aurora	3,053	122,514	1,458	34,087	96	5,402	162,002
Stott City	547	21,872	.....	.....	8	405	22,277
Lawrence County	3,600	.....	1,458	.....	104	.....	184,280
Dade County	.....	.....	39	.....	.....	.....	695
Green County	.....	.....	.....	.....	37	.....	2,174
KANSAS.							
Galena	16,811	683,441	81	2,424	3,497	198,905	884,770
Badger	1,338	53,248	.....	.....	87	5,210	58,458
Lawton	848	36,914	.....	.....	52	5,125	40,039
Cherokee County	18,998	.....	81	.....	3,636	.....	983,267
OKLAHOMA.							
Miami	7,988	243,450	.....	.....	2,781	157,123	400,573
Quapaw	2,097	75,211	22	620	313	17,561	93,392
Ottawa County	10,085	.....	22	.....	3,094	.....	493,967
Totals.							
Missouri	222,262	.....	17,555	.....	38,911	.....	11,855,310
Kansas	18,998	.....	81	.....	3,637	.....	983,267
Oklahoma	10,085	.....	22	.....	3,095	.....	493,965
Joplin District, 1911	251,346	.....	17,657	.....	45,643	.....	13,332,542
Joplin District, 1910	270,534	.....	23,874	.....	43,707	.....	14,129,022

[1] Abstract *E. and M. J.*, Jan. 6, 1912.

Early in the year the Zinc Producers' Association adopted a contract selling base for zinc-sulphide ores grading 60 per cent. and under. This created a dissention resulting in a disruption in which the opponents secured control of the Association and denounced the contract plan. Talk of exporting ore and of erecting a producers' smeltery assisted in boosting spelter prices, and while ore prices continued to rise, the advance was meager in proportion to the metal advances. However, the collapse of metal at the year-end indicated that no higher level of ore prices was warranted.

The old association offered to place contracts covering 4500 tons for each week's delivery, if there were sufficient sellers to accept that amount, but less than 1000 tons weekly have been sold on contract. The contract placed the price of zinc-sulphide ore at $37 per ton on a St. Louis spelter market of 5 cents per pound with an advance or deduction of 8 1/2 cents for each 1 cent up or down in the metal market. The base was formulated on a recovery of 87 per cent. of the metallic contents shown by assay. The close of the year ended the life of these contracts. A substitute contract is offered, for 6 months, based on a metal recovery of 87 per cent. of the metallic contents shown by assay, with a smelting charge based at $21 per ton.

AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN, MO. (a)
(Dollars per 2000 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1902....	26.75	27.00	28.00	28.85	29.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903....	31.50	32.50	35.75	37.75	36.60	36.50	36.00	36.00	34.40	34.40	30.75	30.00	33.44
1904....	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905....	51.94	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40
1906....	49.33	49.25	45.60	44.00	41.50	44.20	43.88	44.38	43.20	42.50	44.43	45.55	44.82
1907....	46.90	48.30	49.75	49.25	46.90	47.00	46.80	44.56	41.00	41.75	38.60	31.50	44.38
1908....	37.60	36.63	36.19	35.40	34.19	34.06	34.55	36.53	37.63	35.95	39.13	42.75	36.63
1909....	38.48	34.46	34.77	35.99	37.82	40.00	41.28	44.56	44.78	46.04	48.29	47.97	41.08
1910....	45.32	39.47	39.78	39.35	37.51	38.11	36.74	37.32	39.74	40.80	43.40	43.10	40.42
1911....	40.44	39.16	38.62	37.54	37.04	37.15	38.32	40.35	41.53	40.92	42.61	40.76	39.90

(a) Base prices for 60-per cent. zinc ore.

SHIPMENTS OF ORE FROM THE JOPLIN DISTRICT.
(In tons of 2000 lb.)

Year.	Zinc Ore.	Lead Ore.	Year.	Zinc Ore.	Lead Ore.
1894.....	147,310	32,190	1903.....	234,773	28,656
1895.....	144,487	31,294	1904.....	267,240	34,362
1896.....	155,333	27,721	1905.....	252,435	31,679
1897.....	177,976	30,105	1906.....	278,930	39,189
1898.....	234,455	26,687	1907.....	286,538	42,065
1899.....	255,088	23,888	1908.....	258,628	38,532
1900.....	248,446	29,132	1909.....	296,453	43,659
1901.....	258,306	35,177	1910.....	294,408	43,707
1902.....	262,545	31,625	1911.....	269,004	45,643

In the Thoms station area, northwest of Joplin, a dozen new and remodeled mills were in operation at the end of the year where four mills were standing and but two operating a year ago. New mills were erected in the Webb City, Carterville, Sarcoxie, Carl Junction and Jackson fields. Two were built west of Joplin, and several were built in Kansas, around Galena. During the year a number of mills were removed from one to another part of the district and remodeled.

Buyers uniformly agree that the assessing of penalties has had good effect on ore-concentrating methods, but the contract clause of $1 premium for zinc ores carrying less than one-half of 1 per cent. lead has brought better results than penalizing methods. It was a definite offer to producers in a form and manner so ever-present that it could not be ignored. Human nature prefers premiums to penalties, anyway.

The shipment of zinc ores the past years was nearly 30,000 short tons less than 1910. At the beginning of the year the reserve stock was but a little over 3000 tons while at the close it approximated 11,000 tons. The year's production was, therefore, close to 8000 tons greater than the shipment.

Nevada.—The Yellow Pine mine, in southern Nevada, is down 700 ft. on the vein. A narrow gage railway from Jean to the mine is contemplated. The Potosi mine hauls its ore to the S. L., Los Angeles and San Pedro Railway. The outputs of the district are shipped to Joplin, Mo.

New Jersey.—The Mine Hill zinc mines, at Franklin Furnace, in Sussex County, New Jersey, shipped 308,353 tons[1] to the concentrators (producing 263,606 tons of concentrates), and 67,324 tons of crude ore to the smelteries. Figured as metallic zinc, the total recoverable output was 137,355,219 lb. of spelter.

The mines are opened by a vertical shaft 965 ft. deep, a slope to the 600-ft. level, and a shaft 1500 ft. deep on an incline of 47 1/2 degrees. The crushing plant has a capacity of 2240 tons per 24 hours and the separator a capacity of 1344 tons.

New York (By D. H. Newlands).—Exploration of zinc blende deposits near Edwards, St. Lawrence County, were in progress last year, and some shipments of selected ore, amounting to several hundred tons in all, were sent to the smelter. The work was carried on by the Northern Ore Co. of Philadelphia. The developments have encouraged the company to provide a mill which is now in course of erection and which is planned to treat 50 tons crude ore a day.

The zinc ores are found along a northeast-soutwesth belt that parallels the talc range with the same crystalline limestones for county rock. They occur as disseminations along bands of the limestone and also in

[1] U. S. Geol. Survey.

compact bodies which are lens-shaped and range up to 4 or 5 ft. thick. Though narrow as compared with most commercial deposits, they average very rich, in some places over 40 per cent. zinc. In addition to blende they carry more or less pyrite and in the southwestern part of the belt considerable galena. Thus far exploration has been limited to one outcropping lens in the northern section near Edward, and further work will be necessary before the importance of the district can be estimated.

Oklahoma.—Notes on mining and production are given by George W. Bingham, *E. and M. J.*, Jan. 6, 1912.

Tennessee.—A new zinc production is expected to come before long from the Holston River district in Tennessee, where the American Zinc, Lead and Smelting Co. claims to have developed 11,000,000 tons of ore and is now sinking a shaft to a depth of 1000 ft. for its extraction and is planning to erect a concentrating mill of 2500 tons daily capacity at Mascot. The chief production of zinc ore in this district in 1911 was made by the Grasselli Chemical Co., which erected a new mill there.

Washington.—The lead-zinc ore production from the Oriole, near Metaline Falls, Stevens County, decreased 20 per cent. in 1911.

Wisconsin.—The table of shipments for Wisconsin-Illinois is given by J. E. Kennedy, Platteville, Wis. No zinc ore was reported shipped from Iowa during the year 1911.

WISCONSIN-ILLINOIS ZINC ORE SHIPMENTS.

	Gross from mines. Pounds.	Net to Smelteries. Pounds.
Wisconsin:		
Highland	9,612,430	9,143,130
Linden	16,945,482	753,150
Harker	7,798,995	7,469,525
Mineral Point	2,314,169	36,838,929
Montfort	3,011,060	1,008,120
Livingston	12,979,290	285,000
Rewey	6,111,660	1,765,120
Platteville	29,315,420	36,190,240
Cuba City	4,384,090	4,056,700
Benton	52,188,830	13,873,610
Hazel Green	39,469,020	12,052,220
Shullsburg	9,067,800	1,478,220
Dodgesville	460,910	460,910
Illinois:		
Galena	21,287,980	26,313,570
Totals	214,947,136	151,688,444

Some Notes on Foreign Countries

Africa (By H. F. Mariott[1]).—The recent advance in the market price of zinc served to direct attention to the ore to be found in the Transvaal, and, although the production is at present very small, it is

[1] *E. and M. J.*, Jan. 7, 1912.

anticipated that the output will be increased to some extent in the near future.

Australia, New South Wales.—Silver, lead, and zinc mines produced 1,484,397 tons of ore in 1911 and 1,243,648 tons in 1910; zinc concentrates were produced as follows:

	Tons.	
Amalgamated Zinc	142,083	De Bavay process
Proprietary	92,302	Potter process
Zinc Corporation	92,527	Minerals process
Minerals Separation	94,015	Minerals process
British Broken Hill	76,585	Elmore process

The Proprietary Company began the production of spelter on a larger scale at their Port Pirie works. In the last half of 1911, 1037 tons of spelter were produdced and it is anticipated that a production of 9000 to 10,000 tons will result with the installation of the Hegeler roaster, and with 10 furnaces. The ore treatment and results for the first half of 1911 are given in *Engineering and Mining Journal*, Oct. 21, 1911.

Western Australia produced $500 worth of spelter in 1911.

Austria.—In 1910 the spelter produced, partly from foreign ores, amounted to 12,458 metric tons, of which 47 per cent. was in the form of zinc dust. Zinc dust is specially made at the state owned works at Cilly, Styria.

France.—At Malines, the Murex process of concentration is producing 48 per cent. zinc concentrate from an ore containing zinc and lead in barytes gangue.

Germany.—The exports and imports of zinc ores and products of Germany for the year 1911, are reported, in metric tons, as follows:

	Imports.	Exports.
Spelter	48,355	77,088
Sheets	467	36,093
Scrap	2,252	3,953
Zinc ore	262,398	48,998
Zinc dust	787	3,689
Zinc sulphide	2,719	13,740
Zinc white	4,977	20,677

In Prussia, 714,855 metric tons of zinc ores were mined. Of this 522,440 metric tons were sulphide and the balance calamine. The centers of the industry are in Upper Silesia and Rhineland and Westphalia. While prices rose the tonnage declined but the increase in value of zinc produced amounted to 2,280,480 M. The amount of spelter produced from 22 zinc works, employing 11,784 persons, was

216,362 metric tons, which includes that used up for zinc white and zinc manufactures, and also that obtained from foreign ores of which there is a considerable importation.

Italy.—The great bulk of zinc and lead ores comes from Sardinia; but at Bergamo, the English Crown Spelter Co., the Vieille Montague, the Societá Austro Belga and the Societá Orobia together returned 17,320 metric tons of 43 per cent. Zn ores in 1910; these are smelted in foreign countries. Zinc ore exported from Italy in 1911 was 78,948 tons to Belgium, 32,259 tons to France, and 14,167 tons to Great Britain.

Japan.—The zinc ore produced in 1910 was 21,108 metric tons.

Russia.—Poland has the only zinc industry near Olkusch, apparently an extension from Silesia. Production of ore at the Boleslav and Uliss mines was: 1907, 3,441,000 poods; 1908, 3,647,000 poods; 1909, 3,027,000 poods; 1910, 3,302,000 poods. Smelting of zinc was done at works named as follows, in poods:

	1907.	1908.	1909.	1910.
Paulina	235,000	217,000	192,000	232,000
Constantine	145,000	145,000	152,000	156,000
Bendsin	214,000	175,000	141,000	139,000
	594,000	539,000	485,000	527,000

Rich deposits exist also at Sadon, Northern Caucasus, production of zinc being 50,000 poods.

Recently at Primorsk, Far East, 43 per cent. zinc calamine has been mined. This is worked by the Tetyuch Company. From four mines, 914,000 poods of ore were mined in 1910, and 362,000 poods in the first quarter of 1911. Ore shipped was 1,233,800 poods in 1910 and 674,800 poods the first quarter of 1911.

The Russian production of zinc is given as 586,434 poods in 1909 and 526,914 poods in 1910.

Sweden produced 558,484 kilos of spelter valued at 318,336 crowns in 1910.

Turkey.—The amount of zinc ore produced in Turkey in 1911 was 40,000 metric tons. Ore deposits occur at Kirasliyaila and Berghama.

United Kingdom.—The year 1911 saw the successful working of the most up-to-date mill for the concentration of blende in England. It has a capacity of 200 tons per 12 hours, and was designated and constructed by German contractors for a mine in Cumberland. The blende mines are situated in the north of England and north Wales. Derby yields about 600 tons per year. The output was stationary as the increase in Cumberland was counterbalanced by the decreases elsewhere.

THE METALLURGY OF ZINC

BY W. R. INGALLS

In the metallurgy of zinc there was no radical change in 1911; conditions were distinctly in favor of the smelters.

The construction of several new works in the coal field of Illinois was begun in 1911. Two new plants were also built in the gas field of Oklahoma. Smelteries to use natural gas as fuel are relatively cheap and quick to construct. Smelteries to use coal are more permanent, but are more costly not only for zinc smelting proper but also because a plant of this kind in order to compete successfully must be provided with an accessory sulphuric-acid works. It is interesting to note that the first furnace of the new works of the Bartlesville Zinc Co. at Collinsville, Okla., was put in operation on Dec. 18, just 5 months after the beginning of construction of the works. So far as I am aware, this is the shortest time on record.

Interesting events of 1911 were the inauguration of zinc smelting by the Broken Hill Proprietary Co. in New South Wales, at Port Pirie; and the beginning of zinc smelting, on a small scale at Osaka in Japan, where the production of about 4000 tons of spelter in 1912 is projected. The operation of these new plants will be watched with interest.

Ore-Dressing.—The enormous output of zinc ore at Broken Hill continues to be made by the oil-acid flotation process, the patent rights to which are the subject of bitter litigation. In 1911 this process was introduced by James M. Hyde at the mill of the Butte & Superior Copper Co. at Basin, Mont., where ore from Butte containing about 20 per cent. zinc is raised to a product assaying 45 to 50 per cent. zinc. Mr. Atwater, manager of the company, informed me that an extraction of 90 per cent. was being made, the cost of the process being about 75 cents per ton. This company is paying no royalty. Suit on the ground of infringement has been brought against Mr. Hyde by Minerals Separation, Ltd. The Butte & Superior is engaged in the construction of a new mill at Butte, which will be of capacity of about 200,000 tons of ore per annum.

The Macquisten tubes have been introduced in the Morning mill at Mullan, Idaho, for the treatment of middlings, and it has been reported that an installation of them is to be made by the Grasselli Chemical Co. at Park City, Utah. Henry E. Wood operated the flotation separator of his invention in his testing works at Denver, Colo.

A good deal of attention was devoted in 1911 to what may be called the pyrometallurgical concentration of zinc ore by burning off the zinc after the fashion of the Wetherill process of zinc oxide manufacture, but without the refinements that are necessary when the oxide is to be

collected for use as a pigment. Such a process of concentration may find a useful field in the treatment of ores and metallurgical products that are not amenable to mechanical concentration and consideration of this possibility has been inspired by the success of Pape in extracting zinc from the slags of Oker, in Germany. Among others who have been experimenting on this line are Babé, Blum, Hommel, Witter, and Huntington.

Grades of Spelter.—At the annual meeting of the American Society for Testing Materials in June, 1911, Prof. William Campbell, of Columbia

Grade.	Pb.	Fe.	Cd.	Total not over.
A. High grade[1]	0.07	0.03	0.05	0.10
B. Intermediate[1]	0.20	0.03	0.50	0.50
C. Brass special[1]	0.75	0.04	0.75	1.20
D. Prime Western	1.50	0.08	.....	.....

University, chairman of the committee on "Non-ferrous Metals and Alloys" reported a classification of the grades of smelter as above, the figures representing the maximum percentages of impurities allowable.

This classification is rational and corresponds substantially to the understanding among American zinc smelters.

Roasting Furnaces.—In the United States the furnaces in common use for blende roasting continue to be the Hegeler and the Zellweger, the former being employed invariably when it is desired to use the roast-gas for sulphuric manufacture, while in the natural gas field the Zellweger furnace is employed. At Cherryvale and Neodesha, where the supply of natural gas has seriously diminished, these furnaces are now, I understand, being fired with petroleum.

Many of the zinc smelters in Germany have spent large sums of money in the trial of furnaces of their own design and construction. It is difficult to see why the European smelters have not seen fit to adopt the Hegeler furnace, which has been well proved in American practice, and although being far from perfect from the economical standpoint is, nevertheless, immensely superior to hand work.

At certain works in Swansea, Wales the Merton furnace has found successful employment. With reference to hand-raked furnaces the Rhenania continues to be the standard, but in Belgium and France the new furnace of Delplace has found more or less use. This is an elaboration of the old Maletra shelf-burner. As to the results accomplished with it, some remarkable stories are told. It is said that in a furnace roasting 15,000 kg. of ore per 24 hours the consumption of coal is as low as 90 to 120 kg. per 1000 kg. of raw ore.

[1] To be free from aluminum.

The counter-current type of furnace is a favorite of the New Jersey Zinc Co., at whose Palmerton works the Convers & De Saulles furnace is employed. The Schmidt & Desgraz furnace, which has lately found rather extended employment in Europe, is also of the counter-current type. Furnaces of the Schmidt & Desgraz construction have been installed at the works of the Broken Hill Proprietary Co. in Australia.

As to the furnaces of the reversing-regenerative type, the Overpelt furnace is generally in use among European smelteries. Furnaces of this kind exist in the United States at the works of the United States Zinc Co. at Pueblo, Colo. A new design of reversing-regenerative furnace is that of Dor-Delattre, which is in use in Europe. This furnace is said to have been operated with a consumption of heating coal of only 900 kg. per 1000 kg. of ore, which figure is undoubtedly better than anybody else has been able to achieve. Heretofore a fuel consumption of 1000 to 1100 kg. has been considered extraordinary work in the case of reversing-regenerative furnaces, while in the case of the counter-current furnaces the figure is 1300 to 1400 kg. and upward.

At the Birkengang works, near Stolberg, Rhenish Prussia, experiments have been made on a large scale with a new furnace containing vertical retorts. The results attained have not yet been made public.

The tendency among European zinc smelters to drive their furnaces harder, and at high temperatures increases.

As to the arrangement of the retorts of the distillation furnace, European practice tends toward the Rhenish system; American toward the Belgian. Among American works using natural gas the Iola type of furnace still prevails, in spite of certain disadvantages.

Electrothermic Smelting

At the Scandinavian works electric zinc smelting has been carried on after a fashion for 5 or 6 years back, the DeLaval arc furnace having been first adopted. The works at Sarpsborg still have those furnaces, but at Trollhättan a resistance furnace is now employed. Metallurgically both furnaces are said to behave in the same way, but the resistance furnace at Trollhättan is very much more economical of power than the arc furnace of Sarpsborg.

For many years it was impossible to smelt ore at these works and obtain spelter in one operation. The product of ore smelting was a metallic powder. By resmelting that powder a spelter containing a large percentage of lead was obtained. Finally by a third smelting spelter of high grade was produced. Such spelter as has come from these works has been chiefly derived from the treatment of dross, scrap, etc.

Ore smelting was not, however, abandoned. Lately a charge, consisting mainly of powder, with a small proportion of ore was smelted. The product from this charge is partly a very leady spelter, and partly powder, the latter going back for resmelting. The leady spelter is refined by redistillation. The recovery of metals is low and there would not be possibility of commercial success were it not for the extremely low cost of power available in Scandinavia.

In the smelting of 537 metric tons of ore at Trollhättan the consumption of electric power was 2078 kilowatt-hours per ton of ore, this not including the refining of the leady spelter. At $10 per kilowatt per year (0.116 cents per hour., which is approximately the rate at Trollhättan, the cost per ton of ore is $2.41.

In the United States experiments in electric zinc smelting were conducted in 1911 by W. McA. Johnson at Hartford, Conn. At Hohenlohehütte, Silesia, an experimental plant of Imbert-Thompson-Fitzgerald was erected and run with results that have not been publicly reported. These were the most ambitious attempts in 1911. Besides them, experimental work on a small scale has been done by numerous metallurgists.

In spite of all the experimental work in electric zinc smelting that has so far been done, and the reports of success achieved, it must still be pronounced that this art is absolutely in its infancy and it is impossible yet to say definitely that certain metallurgical obstacles are surmounted, much less that the process will offer any commercial advantage.

Hydrometallurgical Process.—Several processes for zinc extraction were exploited in 1911, but none of these commanded any very serious attention. Among them are the Malm process (chlorination) for which a mill (not yet in operation) has been erected at Georgetown, Colo.; a chlorination-electrometallurgical process that it so be tried at Helena, Mont.; and the so-called bisulphite process that is being exploited in Great Britain. The process has been developed at a works in Swansea, and a plant for the treatment of the ore of the Roseberry mine is now reported to be in course of erection in Tasmania. The bisulphite process consists essentially in dead-roasting the ore and then digesting it with water into which sulphur dioxide from the roasting is introduced in a special apparatus. The solution of zinc bisulphite thus obtained is separated, the residue going to lead-smelting furnaces and the solution to an apparatus in which the zinc is precipitated as monosulphite, the sulphur dioxide that is evolved at the same time being returned to the process. The zinc monosulphite is calcined to oxide, which it said to be of the density required for reduction and distillation in retorts in

the ordinary way. As thus described, the process possesses no features differing from the proposals of many metallurgists in the past, and it is to be supposed that the novelty of the present process lies in the methods of conducting the several operations and the design of the apparatus employed.

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By L. E. Young

One might expect that owing to lack of business expansion in general during 1911, and also to a stoppage in the expansion of the mining industry in particular, that the literature relating to this industry would reflect such shortage of new material. This, however, is true only in that fewer new enterprises and undertakings were described in the technical journals. At the same time existing practice could be given much thoughtful study, and in this respect the industry has profited more than usual during this period. It does not necessarily follow that a great deal has found its way into print and so has given the man in the mining fraternity the advantage of learning by his brother's experiences and errors, but it does appear that the general literature has been during the year of a specially conservative and sound nature. The workers in the mining field cannot follow the literature excepting perhaps a journal or other and then there is much that never reaches the stage of digestion and probably should not. The following is a list of references covering the year 1911 and giving a selection of articles that are not complete treatises but have certain merits recommending them to the busy man, especially if desiring to read about any one branch in a short space of time. Many articles have been omitted as they are not of equal merit or are already covered in others given.

In preparing this bibliography, the author has had in mind not to make an exhaustive review, but to give such articles that are really worth while, new, or at least a good review of approved practice. This would mean that, though only just published, as far as can be judged the articles are expected to live. It is expected that such articles will be found among those published by the American journals of large circulation, and these naturally will fulfil one requirement that seems important, namely, to be accessible to the average American mining man. For this reason also articles in foreign journals are not generally given except in the form of reviews of which the author has selected the best and most accessible ones. Original titles are not in all cases used but are modified so that they describe the subject matter that appeals to the mining engineer. In this way the reference always indicates the true contents of the article. The author believes that the articles given can be found

in the average library of the larger centers to which the practising engineer or mining man would have access. The classification has been subdivided but the headings are arranged alphabetically and can be readily looked over.

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PLACER MINING

The two forms of metalliferous alluvial deposits that are of greatest commercial importance are those of gold and tin. The following pages refer principally to gold placers. The application of dredging for the recovery of gold from some classes of such deposits was quickly followed by a similar application for the recovery of tin, which has heretofore been confined substantially to some of the Australian colonies, although attempts to introduce the process have been made in the Malay peninsula. Dredging for alluvial tin is going on in both east and west Cornwall, where dredges are now at work. For platinum dredging see page 606.

Gold Dredge Installations During 1911

By John Tyssowski

There were 24 gold dredges erected in North America in 1911, distributed as follows: Alaska, 9; California, 8; Canada, 2; Idaho, 2; Montana, 2; South Dakota, 1. These figures are verified by information from manufacturers; the totals, however, include several dredges reconstructed or built previously but first operated in 1911, as will be seen by referring to the accompanying table. For comparison with 1910 see article in Mineral Industry, XIX, p. 723. The total digging capacity represented by the new dredges is approximately 3,147,000 cu. yd. per month. The digging capacity of the 24 dredges installed in 1910 was approximately 2,745,000 cu. yd. per month.

Dredging in Alaska

By John Power Hutchins

The physical characteristics of the dredging ground and the climate are alike in both Alaska and the Klondike. Some consideration will therefore be given to the latter in the following article. The high cost of labor and supplies due to lack of agriculture and stock raising, and the necessity of importing food and fodder, are likewise the same in both of these placer regions. Though the latter is in Canadian territory and therefore not properly Alaska, it is largely exploited by American capital under American management and its technical problems are similar.

NORTH AMERICAN DREDGE INSTALLATIONS IN 1911.

Locality.	Operating Company.	Bucket Capacity, Cu. Ft.
Alaska (9):		
Solomon River, S. P.	Casadepaga Mining Co.	2½
Solomon River, S. P.	Solomon Dredging Co.	3½[1]
Shovel Creek, S. P.	Charles Kimball	2½
Ophir Creek, S. P.	Wild Goose Mining and Trading Co.	3½
Seward Peninsula	Star Dredging Co.	2½[2]
Nome District	Saunders Dredging Co.	.....
Nome (Osborn Creek)	Gold Beach Dredging Co.	5[1]
Nome (Osborne Creek)	Julien Gold Mining and Dredging Co.	2½
North Alaska	James Kelleher	2½
California (8):		
Folsom	Union Dredging Co.	9
Sacramento	Beaver Gold Dredging Co.	3½
Natoma	Natomas Consolidated of California (No.9)	15
Natoma	Natomas Consolidated of California (No. 10)	15
Thermalito	Feather River Dredge	15
Oroville	Natomas Consolidated of California (Feather No. 3).	15
Oroville	Lawrence Gardella	5
Trinity County	Alta Bert Gold Dredging Co.	7½
Canada (2):		
Dawson, Y. T.	Yukon Gold Co.	7½[1]
Dawson, Y. T.	Dominion Goldfields, Ltd.	.....
Idaho (2):		
Idaho City	Boston and Idaho Dredging Co.	14½
Salmon	Kirtley Creek Gold Dredging Co.	9
Montana (2):		
Canon Ferry	Magpie Development Co.	5
Kennedy placers (Huson)	California Dredging Co.	5½
South Dakota (1):		
Mystic	Castle Creek Dredging Co.	7[2]

[1] Reconstructed.
[2] Built previously, 1911 first operating season.

Tin Dredging on Seward Peninsula.—Soon after gold was discovered on Seward Peninsula, prospectors, who were looking for gold-bearing streams, discovered the alluvial tin deposits between Cape York and Cape Prince of Wales. Several times during more than 10 years, these deposits were examined by experienced engineers but their reports were such that no considerable capital was invested. When these examinations were made, however, it had not been demonstrated that dredging could be successfully applied, either for gold or tin, on Seward Peninsula. Attempts were made to work by small scale methods and these have been continued, sporadically, ever since the discoveries of the deposits.

With the successful application of dredging to the gold-bearing gravels near Nome, in 1905 and subsequently, and particularly, of the internal combustion engine, using distillate or similar fuel to run dredges, it seemed possible to dredge successfully on Buck creek, some 14 miles north of Cape York. As on other parts of the Seward Peninsula, there are no forests and the fuel problem had been the greatest obstacle.

In 1911 a tin dredge of the York Dredging Company was installed and operated about 30 days. This is quite noteworthy in many respects. First, it is the only successful tin dredge on the Western Continent, some dredging for tin in South America having been unsuccessful. Second, it is the first and only tin dredge in North America. Third, the dredge was erected in one month.

This dredge has 2 1/2 cu. ft. buckets and, though designed and built by Californians, the dredge is not exactly of the California type, inasmuch as it was necessary to modify the parts intended to save the tin and in doing this, it allowed a long, inclined stacker to be eliminated. The tin-saving surfaces are arranged so that part can be cleaned up while the rest is being used and this is an important advantage, for frequent clean-ups are necessary and otherwise the dredge would be shut down during these frequent clean-ups. The dredge operates on a headline.

It was found that the excavating capacity of the dredge about 1200 cu. yd., was greater than the extraction capacity by about 25 per cent. and it was noted, also, that considerable losses occurred while dredging at the top speed. Tin is not saved as easily as gold, because of its lower specific gravity, and a greater degree of care is necessary in operating the tin-saving devices and more frequent "streaming down" is required than in saving placer gold.

Operating costs are not yet available, the 30-days' run having been insufficient time to get data as to what may be expected of average running, it is probable that these will be somewhat higher than for gold dredging with California type dredges under similar conditions on Seward Peninsula, for, not only does this type of dredge require more power per cu. yd. than the California type, but more labor is needed in the more frequent clean-ups and for handling the larger amount of concentrate.

It is said that the average recovery of tin concentrate was over 5 lb. per cu. yd. and that in all about 100 tons were saved. The selling price of concentrates per ton is not known but it is quite evident that, with the present price of tin, the owners will be well rewarded for their enterprise, skill and courage.

The following details of design illustrate the difference between the parts of gold and tin dredges that must be modified to save the tin.

The revolving screen has 3 in. holes, as some of the tin concentrate is of this size. The California dredge usually has holes in the screen only about 0.5 in. diameter, as nearly all the gold in California dredging fields will pass a 0.5-in. hole. Inasmuch as such a large proportion of the material excavated, passes through the screen apertures and into the

slimes and, therefore, only an inconsiderable amount goes to the stacker, this latter need not be long as it is not necessary to carry it far from the dredge or to lift it high. This dredge rather resembles the so-called single-lift dredge, formerly used at Ruby, Montana, than it does the California dredge and tailings from it do not expand to great degree, as with the California dredge, which descharges first, the fine material into the bottom of the pond, and afterward stacks everything, over 0.5 in diameter, on top of the fine. As a result of this latter method of stacking, there are large interstices and usually irregular and high tailing piles. With the method in vogue on the tin dredge, it is quite necessary to keep the dredge pond almost full or tailings will be washed back under the stern of the dredge and cause trouble.

In order to allow the large pieces of tin concentrate to be saved, the Hungarian riffles in the sluices are 2.5 in. deep and 2.5 in. apart. The holding capacity of the riffles must be large to also furnish room for the greater amount of material to be caught than is necessary when dredging for gold.

The ore occurs as cassiterite and it is said to be about 50 per cent. pure tin; this would make the concentrate, as cleaned up on the dredge, worth between $400 and $500 per ton with tin at its present price.

Gold Dredging on Seward Peninsula.—Of all the gold produced on Seward Peninsula in 1910 about three-fifths was produced by dredging; in 1911 over seven-eighths of all the gold was from dredging. This is largely explained by the decrease in the product from other kinds of mining, as well as by the increased number of dredges that operated. There was an increase in the number of operating dredges from 14 to 19 and there are six others in course of construction which should all be operating in 1913.

Experience, during the past year, had demonstrated again that the types of dredging machinery that have been evolved, as a result of experience, are the most reliable and, particularly, is it extremely hazardous to try new devices in places like Seward Peninsula, where, not only are machine shops of small capacity, and machine work very expensive, but with such short working season, any time lost, because of trying new devices, is of much greater consideration than in regions in temperate zones where dredging is carried on all or most of the year.

The application of engines of the internal combustion type for operating dredging machinery has proven successful and has been a boon to Seward Peninsula. The first dredges used coal which was very costly, about $25 per ton at the dredge site, because principally of cost of transport. Fuel for internal combustion engines, being cheaper to

transport, per horse power produced, has extended the dredging horizon considerably.

While the internal combustion engine is not so flexible in its control as a steam engine, and is much less satisfactory than the electric motor for driving dredging machinery, it has still proven a success, particularly, on the smaller dredges on which it has been so far installed.

It is anticipated that, as the placers on Seward Peninsula now being operated by one or another hydraulicking method are exhausted, the water may be used for generating electric power for dredges. This would result in a considerable reduction in operating costs.

Projects for Dredging.—There are several projected installations for Seward Peninsula and, in one notable instance, the advisability of erecting electric dredges on and near Solomon river, to be operated from a central generating station using California petroleum, is being seriously considered. Should this be done, it is likely that a new low record for dredging cost will be made for Seward Peninsula.

Some parts of the old beaches, back of Nome, are thought to be susceptible of successful dredging. The problems are a frozen condition and depths so great as to make steam-thawing difficult, if not impossible. It is not usually very difficult to thaw with steam to a depth of about 20 ft., but beyond this depth it becomes very difficult to drive the steam points, used to transmit the steam into the frozen gravel, and encountering boulders results in deflecting the points in a direction not intended. This results in an unequal distribution of heat and unthawed masses being left. Should the cost of thawing be reduced, and the method improved so as to allow its successful application to deeper ground, it is probable that a large additional volume of ground will become available for dredging near Nome.

The present beach at Nome is being considered and it is thought that it may be worked with dragline excavators. These devices have the advantage of (1) long reach to excavate the gold-bearing beach sand while set on a track out of reach of high tide; (2) to dump onto a washing plant out of reach of waves during storms and (3) to haul themselves up onto the tundra to a safe place, quickly enough to avoid damage from storms.

The problem of saving the fine beach-sand gold, which does not amalgamate freely and which occurs associated with much black sand, can probably be solved. The circumstance, that there is no way at present of controling a considerable length of beach as title cannot be acquired, is a serious drawback to large scale operations. The recent successful application of the drag-line excavator to placer mining suggested its possible use at Nome.

This excavator is also being considered for working frozen gravel. It is thought that it may be used, somewhat as the steam-scraper has been used, in Klondike, taking advantage of its long digging radius to scrape off, each day, what gravel thaws by exposure to sun and air across a wide face. It is yet to be determined whether gravel will thaw fast enough over a face about 250 ft. wide, which is a convenient width for this excavator, to supply enough unfrozen material to keep the machine working at full capacity, even during mid-summer. There is the possibility of using this type of machine over wider face and so having larger thawing surface which, however, makes more frequent moving necessary and complicates the track system.

Dredging Costs on Seward Peninsula.—Cost for operation alone is said to have been as low as 10 cents per cu. yd. in some instances. It is interesting to note in passing that operating costs for a 7.5 cu. ft. bucket California-type dredge installed in Siberia, in 1911, were about 11 cents per cu. yd. where the climate conditions are similar to those of Seward Peninsula. High administration costs, which are inevitable with the short working season of the far north, will probably raise the total operating costs to, perhaps, more than double the operating cost given above. It is said that steam thawing was done in one instance for about 10 cents per cu. yd., where the gravel was not deeper than 12 ft. This is cheaper than the cost for thawing in Klondike where although done on a larger scale, it is applied to deeper ground.

Dredging in Forty-mile District.—This district is nearest Klondike of all dredging fields in Alaska. The season of 1911 was one of checkered careers for the dredges in this region. There is a very irregular distribution of payable gravel along the stream beds and on the adjacent bars, and the resulting ups and downs of clean-up returns was noted as usual. Some very rich gravel was worked, but some very poor gravel had to be worked while moving from one good place to another.

From the Forty Mile to Seward Peninsula is a distance of about 600 miles as the crow flies, and it is some 800 miles to the mouth of the Yukon River following its meanderings. There are no successful dredges on the Yukon and its tributaries between Forty Mile and the mouth of the Yukon. While there are important placer districts on the tributaries, and rich placer is being worked, the physical conditions, notably that of frozen ground, are such as to make it difficult or impossible to dredge many of them with the present difficulties and high cost of steam-thawing. One dredge has been installed in the Fairbanks region, during 1911. Data are not at hand as to its operations and it cannot be considered more than an experiment, until it shall have operated for a longer time.

It is thought that, with improved transport, and thawing methods, dredging will be extended in this region.

The importance of improved transportation, as an essential to any considerable expansion of dredge-mining operations in Alaska cannot be much exaggerated. It is almost, if not quite, impossible to transport heavy machinery at all in summer, except over a few roads near the most important mining centers. Winter transport, while easier, is still fraught with trouble and expense. The need of roads in Alaska is a crying one.

Dredging in Klondike.—There was no particularly interesting installation in Klondike except that of a dredge with 16 cu ft. buckets; these are as large as those on the largest dredges in California. There has been a rather strong tendency to install medium and small-size dredges, as the difficulties of transporting, handling and repairing the heavy parts of the larger dredge is supposed to be a greater drawback in the far north than elsewhere and most of the dredges in Klondike are therefore of medium size. This big dredge seems to have been a success, but data, which can only be had after its longer operation, will be needed before its operating cost can be compared with those of smaller dredges.

It has been found that the floating dredge has such advantages of mobility that frozen ground can be worked more cheaply than by means of hydraulicking with mechanical elevators even though steam-thawing must be done while dredging, and is not necessary for the other method, as the exposure to sun, air and running water has a thawing effect.

Preparations are being made to work Dominion Creek on a large scale by a powerful corporation backed by English capital. On this stream, as in most parts of Klondike, frozen ground occurs and various devices and methods are being considered to overcome the difficulty and cost of steam-thawing. Some of this ground is more than 20 ft. deep and this feature as rated elsewhere in this article adds to the ordinary troubles of steam-thawing. The possibility of removing or of stripping part of the top material, by hydraulicking, by steam or electric shovelling or by drag-line excavating, is being considered and, as there will be a considerable amount of cheap electric power available, it is expected to drive such machinery at a low cost per cubic yard worked.

Dredging Cost in Klondike.—This has not changed materially during 1911 and from about 15 to 25 cents per cubic yard for unfrozen and about 30 to 40 cents per cubic yard for frozen ground was noted with large scale operations of such magnitude as to make low administration cost per cubic yard.

Economic Conditions as Effecting Dredging Costs.—In all of Alaska and the Canadian Yukon, there has been little attempt made to make the country self-supporting by agriculture and stock raising. Comparing this with Russia where the temperatures are similar, near Omsk and Tomsk, these industries are of such magnitude that large imports are made to Europe. Siberian food and fodder is cheap and so is the labor. In the Nerchinsk region, where perpetually frozen ground is found probably as frequently as in North America, common labor costs 50 cents per day and a horse can be hired for 50 cents per day. It may be said that the inhabitants of Alaska and Klondike are practically inexperienced in living in the extremely cold winters. Traveling in Siberia is generally done in closed sleighs with proper clothing. The writer recalls an exhausting trip from White Horse to Dawson without a fur coat as this was advised unnecessary. This suggests that there may be much about living, farming and stock raising in the cold climates for the Alaskans and Klondikers to learn.

CALIFORNIA.

During 1911 one of the largest dredges was burned and others were sent to the scrap heap, but several large new dredges were built. The modern dredging machines, in the more extensive fields, are working ground of an average recovered value of 21 cents a cubic yard, at a working cost of 5.67 cents a yard; and ground averaging 9.64 cents a yard at a cost of 4.52 cents. In the Yuba River field, now the most important in the State, the year 1911 was a record one, as it was also in the Folsom field of Sacramento County. The Oroville field where the best ground has been worked out showed some falling off in yield.

(By Lewis H. Eddy.[1]) The increase in bucket capacity to 15 cu. ft., advancing the digging depth from 35 to 65 ft. below the water line, increase in yardage from 350 to 600 cu. yd. per hour, extending the area of gold-saving tables from 900 to 8000 sq. ft., and the introduction of the steel hull are details of material comparison in construction of the largest dredges built in 1908 and the largest in construction in 1911 in the three chief dredging divisions. The increased demand for dredges built especially for operation of old hydraulic and placer mines at interior points, and for the handling of mine tailings indicates that there is no decline in the smaller type of dredges. While the improvement in efficiency and increase in aggregate capacity appear to meet all present demands, the limit of possible capacity of the gold dredge is not admitted to have been reached in the present construction.

[1]Abstracted from *E. and M. J.*, July 8, 1911.

The first of the 15-cu. ft. boats was Natoma No. 8; then followed Yuba No. 13 and Feather No. 3. These are similar in design and general construction.

Natoma No. 8 is equipped with 88 and Yuba No. 13 with 90 close-connected, 4200-lb. buckets; Feather No. 3 with 68 plus 13 close-connected, 3500-lb. buckets. Total power equipment of Natoma is 720 h. p. plus 250 h. p.; Yuba and Feather, each, 720 h. p. The Natoma is equipped with two monitor pumps, accounting for the added horsepower. The Feather is designed for digging two depths, 35 and 50 ft., accounting for the additional number of buckets and the additional length of digging ladder.

In June 1911, 61 dredges were operating on 14 streams, including rivers and creeks, and at two interior points off stream. In Feather River division: 19 in Oroville field; two in Thermalito; one on Honcut Creek; two in Wymans Ravine; two on Butte Creek; total, 26. Yuba basin: 12 at Hammonton; three at Marigold; total, 15. American River division: four in upper and four in lower Natoma field; one at Folsom; total, 9. In the other counties there were three in Placer, two in Shasta, and one in Monterey.

The Oroville and Thermalito fields combined increased the average monthly output from 1,560,000 to about 1,900,000 cu. yd. in the last half of 1911 by the addition of new dredges.

Only two companies operate gold dredges in Yuba basin, the Yuba Consolidated Goldfields at Hammonton, and the Marysville Gold Dredging Company at Marigold. These companies own all the gold-bearing dredging lands, as far as prospected and proven. Yuba Consolidated operates 12 dredges, two of which are of 6 cu. ft. capacity, equipped with shaking screens; the 10 others are of 7 1/2 cu. ft. capacity, and have revolving screens; the spray pipe for washing is being displaced on these larger boats by the nozzle washing device.

The dredges are digging 40 to 63 ft., and it is expected that Yuba No. 13 will go deeper, probably to 70. The value of the gravel for the two fields is reported at from 10 to 30 cents per cubic yard.

Both companies operating in Yuba basin work in conjunction by contract with the United States Débris Commission in the construction of retaining walls describing the new channel of the river.

Natomas Consolidated operates also two rock-crushing plants on the American River fields, and one at Oroville. The land department of this company operates a reclamation project both on the American and the Feather Rivers, though on a smaller scale at Oroville than in the vicinity of Natoma.

The interesting feature of dredge operation in the Natoma fields is

at Rebel Hill, situated on the north side of American River. The dredge, which began operation at Rebel Hill in January 1911, is equipped with a monitor with a 12-in. pump with 3-in. nozzle, under 200-ft. head, capable of delivering 3600 gallons per minute. No. 9 dredge is also equipped with monitor, and the yardage output is 130,000 cu. yd. per month. The boat was digging 60 ft. The gravel at Humbug Hill is similar to that at Rebel Hill. No. 10 dredge is not equipped with monitors for breaking, but will have a steel hull, and be especially designed to withstand the strain of tight digging 65 ft. below the water line with especially strong 15-cu. ft. buckets, and a yardage capacity of 300,000 cu. yd. per month.

Oro Vista No. 1 dredge,[1] installed at Oroville, Calif., by Lawrence Gardella was put into active commission on Jan. 22, 1912. It is probable that this installation will be the last new boat in the Oroville field of the Feather River district, as the field is now covered with a full complement of dredges necessary to the area remaining to be worked.

This is a 5-cu. ft. bucket-elevator Risdon-type dredge designed to dig 35 ft. below the water line in average operation, but equal to a depth of 42 ft. below the water line under extreme conditions and to handle a bank 12 ft. above, making a total depth of 54 ft. The ground which is situated between two sections of the town of Oroville is hard-digging material, necessitating especial strength in design and construction to withstand the severe strains to be encountered. The dredge is equipped with revolving screen, belt conveyor and standard-type gold-saving tables.

The cross-system plan of constructing gold-saving tables has been adopted by a number of dredge operators in California within the last year or two. In some instances the system was installed with the original construction of the dredge; in others it has displaced the Holmes type of longitudinal tables.

(J. Tyssowski.)[2] The 9-cu. ft. dredge, built for the Union Dredging Co., and installed at Folsom, Cal., deserves particular mention. This boat is designed with especially heavy machinery and will dig to a depth of of 55 ft. It is equipped with double gold-saving tables and hydraulic giants fed by a 10-in. centrifugal pump and has a total power equipment of 625 h. p.

The California gold dredge has been described in full detail by Robt. E. Cranston of Sacramento, Cal. in a paper before the American Society of Mechanical Engineers.[3] The mechanical features of this type of dredge, which was evolved from a New Zealand and other types, are criticized.

[1] See article by Lewis H. Eddy in *E. and M. J.*, Feb. 10, 1912.

[2] "Dredge Installation During 1910," by John Tyssowski, *Eng. and Min. Journ.*, Jan. 7, 1911; "Notes on the Construction of California Dredges," by John Tyssowski, *Eng. and Min. Journ.*, Oct. 15, 1910.

[3] Published in Feb., 1912.

He points out the best construction of buckets and the material used for their parts; also the dimensions and type of tumbler shafts, digging ladders, drives, grizzlies, hoppers, spuds, etc. He shows how to add strength at small cost for the results obtained.

Colorado (Geo. E. Collins).—At Breckenridge the Revett and Reiling dredges, on French Gulch, were actively and profitably operated, and the large dredge on the Swan, operated by the Colorado Gold Dredging Co., was also kept at work. These dredges handle from 3000 to 6000 cu. yd. of gravel per day and the cost is said to be five cents per cubic yard. In 1910 the Colorado Company's dredge is said to have recovered $250,000 at a cost of $65,000. The Twin Lakes placer, near Granite, has been idle, and its future operation is likely to be hampered by opposition of those owning water rights on the Arkansas River. The other placers in Colorado have been unproductive; but plans have been made for re-working on a large scale some of the alluvial ground near Fairplay. The discovery of placer gold in commercial quantity is reported at Newlin's Gulch, near Denver.

F. L. Ransome describes the character of the placer gold of the Breckenridge district.[1] At the Swan at Snyder's camp, the gravels are 27 ft. deep. Among the nuggets found from working on the Mascot placer, are some as long as 15 mm. and 5 mm. wide. Lower down the Swan, the proportion of nuggets increased and gold, recognizable as of the Farncomb Hill derivation, becomes less abundant. The dredge now at work would probably lose over the stacker any nugget exceeding an inch in diameter.

Georgia.—An electrically driven Bucyrus dredge belonging to the Seawell Dredging Company was at work on the Chestatee River in 1911. Black sand amounts to approximately 1 per cent. of the total gravel and carries from $3 to $6 per ton in gold, but this, and some of the heavier minerals which occur, such as rutile and monazite, are not saved.

Idaho.—At the new plant at Idaho City, 40 miles north of Boise, of the Boston and Idaho Gold Dredging Company, a 10,000-cu. yd. dredge (13 3/4 cu. ft. bucket), has been in operation since June 1911 and during the winter. The ground averages 10 to 20 cents per yard. The large area owned will make up for exhaustion of ore as in Oroyhee County. Another dredge built by the Yuba Construction Company, with 9 cu. ft. buckets was put in operation on the Kirtley Creek placers near Salmon, Idaho, in September on ground of similar value. These two large plants have extensive resources of gravel and should continue to make a rich and profitable yield for years. The Willis Placer Company on Stanley creek, Austin County, has added disintegrating devices to its dredge, in

[1] Professional Paper 75, United States Geological Survey, Geology of Breckenridge District, Colo.

order to overcome the troubles with a layer of talcy gravel containing rusty gold. The Sturkey placer, an old steady producer, was somewhat limited in its output because of water shortage. This property averages 25 cents per yard in gravel 15 to 20 ft. deep. These properties also show cinnabar (the sulphide of mercury) pebbles. The Elmore Placer Company expects to produce during the coming season and has made improvements in its appliances.

Montana.—The California Dredging Company operating at Kennedy Placer, Huson, Montana, uses a 5 3/4 cu. ft. dredge. Of interest regarding this dredge is the use of open-connected bucket chains and the steam power. Water tube boilers and compound condensing engines are used.

Near Ruby, Montana, gold dredges have been operating since 1899. Ordinary sluicing and various devices were tried but were found unsuccessful. The first two dredges known as No. 1 and No. 2., were found unsuccessful about 10 years ago. No. 3 was put in commission in 1907, and many of the old features of the previous dredges, such as the long tail sluice, single lift, open connected bucket line and the mooring line system of anchorage, were retained. Buckets having 12 cu. ft. capacity gave fairly good service. The Conrey Mining Company, at Ruby, Madison County, Montana, operates the final dredge, No. 4, which is one of the largest dredges in the world. It has 16-ft. buckets and the design of the machinery throughout is in proportion. This dredge is 150 ft. long, 13 ft. deep and 68 ft. wide. The total weight of the ladder and rollers is 112 tons which was shipped in sections in order to bring it on the ground. The spuds were the largest and heaviest pieces weighing 42 tons each. A special track was made to transport these from the rail road station. This dredge is driven by 1045 hp.; the motors are direct-geared 2200 volt.

Nevada.—The opening of the Nevada gold fields to dredging operations was begun in the middle of November, 1910, in Spring Valley, Humboldt County, 130 miles northeast of Reno. Dry Gulch and American Canyon have been worked for the last 35 years by Chinamen who sank shafts and hoisted gravel by a whim and used rockers. The ground remaining is stated to run 30 cents per cu. yd. and estimates were shown to be correct by subsequent operations.[1] The dredge operated is built by the Risdon Iron Works and has a capacity of 2000 cu. yd. per day. The dredge is driven by gasoline engines. The gravel after washing in revolving screens, courses over tables with angle-iron riffles sloped one foot in fifteen. The tables are 40 ft. long and 6 ft. wide in addition to 32 ft. of riffles in the sluices. The gold is coarse and $3 to $5 nuggets are common.

[1] H. G. Walker, *Engineering and Mining Journal*, June 7, 1911.

South Dakota.—The first dredge in the Black Hills was put in commission in May by the Castle Creek Hydraulic Gold Mining Company, at Mystic. The bucket line comprises 78 buckets of 5 cu. ft. capacity each, geared to discharge 26 buckets per minute. The dredge will dig 35 ft. below water line. The machinery is all electrically driven, power being obtained from a steam plant comprising four 100-hp. boilers and 300-hp. 28x48-in. Corliss engine, driving a 250 hp. motor.

Australia.—The principal dredging in Australia and New Zealand has been on a steady decline. In 1910, 104 dredges were at work as compared with 167 in 1905. The failure of the Ross Goldfields has involved the postponement at least of the Rium Flat sluicing project.

In Victoria the area under lease that was worked by dredging was about 17,630 acres in 1910. In dredging are included 53 bucket-dredging plants, 46 pump hydraulic sluices and eight jet elevators all actually working at the beginning of 1911. About seven dredges were under reconstruction in addition. The total quantity of material treated by all these plants during 1910 was 19,838,552 cu. yds. yielding 87,156 ounces in gold. In 1909 about the same number of plants treated 19,971,131 yards and recovered 88,340 ounces.

Special appliances to comply with the leases as to the clauses relating to restoration of soil-covered areas have been in use. The earthy material has to be advance-stripped to a depth of 7 ft. if necessary, in as dry a state as possible and replaced on top of the material dumped from the boxes. Provision is also made in the leases since 1907, that proper impounding of mining dèbris must prevent its discharge into any watercourse. For resoiling the Briseis dredge No. 2 is provided with a belt conveyor. The buckets of 4-ft. capacity travel at a rate of 12 1/2 per minute. They come up full of earth and may discharge imperfectly into the drop chute from which the conveyor is fed. Revolving tynes are installed to help empty the buckets, but any material sticking will pass down into the well and into the operating pond. While overburden is handled, a shaking chute bumps the material onto the belt conveyor. In this way about one-half the material is delivered on the ground in a dry and useful condition. Another plant, the Myrtle Queen, uses a screw conveyor, the chute being of sheet iron 2 ft. wide.

The trouble with dredging in spoiling the character of the land for agriculture has been much exaggerated. At Harrietville, near Bright, Victoria, in the district where complaints were most common, magnificent crops of grass and clover have been grown on similar ground.

New South Wales.—In 1911 the output from alluvial dredging was the lowest since 1902. There were twenty bucket and nine pump dredges which handled respectively 5,122,476 and 1,260,429 yds. The

values per yard were 3.66d. and 3.81d. The output was 23,364 ounces in 1911. Only one of the Tewkesbury proprietary's dredges was working. The costs are given as 1.083d. per yard.

Tin dredging was prosperous, 1742 tons of black tin being recovered from 577,977 yards treated by four bucket dredges and 2,533,782 yards treated by 28 pump dredges. The tin per yard was .57 lb. and 1.25 for these respectively. The fine returns are due to plenty of water and good prices. The Y Water Tin Company's dredge in the Emmaville district treated 539,300 yards and produced 328 tons of tin.

New Zealand.—At the southern province of Otago, where dredging practice started, the Mystery Flat Gold Dredging Co. is carrying out a novel method of dredging. The dredge is now working its way up the gentle slopes of the terraces and foothills surrounding the river flat. To do this, water has been brought onto the claim by means of a race several miles long and at an elevation of about 50 feet above the river level. The dredge works in a "paddock" supplied by this race and continually stacks the tailings behind, forming a dam to hold the water. In starting the dredge, it was found necessary to build a low dam of tailings and brush, and the same procedure may be necessary, from time to time, as the dredge rises. After passing the water race, it is intended to proceed to a higher level by means of pumping.[1]

Bolivia.—Various placers on the Mapiri River are owned by the Incahuara Gold Dredging Company, Ltd., an English corporation. An American company's deposits are below this; this company is called the Kaka Mines Company. The most important are the Maquique and the Thalia. On these deposits boulders are practically absent and the gravel does not contain clay. The thickness of the overburden varies from 6 to 10 ft. The grade of the gold is high. Examinations of the ore were carried on in a systematic manner before working. Tests were made on one or two cubic yards from each shaft by means of rockers and the result was from $0.78 to $0.33 per cubic yard.

The Incahuara company, which was formed under the management of John Taylor & Sons, has brought in a dredge built by Lobnitz & Co., Renfrew, Scotland. In the rainy season, when the floods are frequent, the dredge will operate the inland bars such as the Maquique.

Colombia.—The native Indians were well acquainted with the art of placer mining long before the Spaniards came. Since that time Colombia has added $700,000,000 to the world's stock of gold, which is more than any other South American country. The primitive processes have exhausted all the ground that would yield to the treatment. The only grounds left are the low-lying, water-filled placers

[1] A Gordon McDonald, *Eng. and Min. Journ.* Dec. 9, 1911.

or river-bottom lands. The Spaniards worked these by water-wheel driven pumps and hand pumps to clear the pits of water. Some of the dredges that have been tried in late years could not even be made to pay in the best California fields. The Oroville Dredging Company, which has achieved great success in California, is to work at the Zaragoza and it is expected that the advent of this company will make for success in Colombian placer mining. Further details as to this region are given by R. D. O. Johnson.[1]

In prospecting dredging ground with Empire drills, much has been done lately in the Miel-Samaná region. The bed-rock consists of sandstone with a soft capping from 1 1/2 to 2 ft. thick. The drill holes vary from 34 to 56 ft. in depth and are spaced about 50 ft. apart. To a depth of 20 ft. the gravel runs about 15 cents per cu. yd., and the pay streak, from 12 to 15 ft., runs about 65 cents, increasing in grade upstream.

Ground sluicing has been commenced on a large scale on the Rosario, a tributary of the Samaná. The gravel averages from 80 cents to $1 per yard. To secure an outlet from the San Antonio portion of the Rosario mine, a tunnel about 200 ft. long will be driven. This is an ancient lake deposit and the ground averages about $2 per yard. Transportation facilities on the Miel River are good, the average depth of water being from 6 to 7 ft., with a current of from three to six miles per hour.

Gold-dredging in the Philippines (By Charles Janin[2]).—The Paracale district was the only field in the Philippine Islands up to a short time ago in which dredges had been even partly successful, for none of the present boats can be properly classed as entirely successful. As to the present time, according to R. Y. Hanlon,[3] there is only one dredge working in the district and another dredge idle on account of litigation. Both of these dredges are of New Zealand design and manufacture. They are equipped with 5-cu. ft. open-connected buckets, work on a headline, and are operated with steam-power, each using about three cords of wood per day. They handle less than 1000 cu. yd. per day. The dredge of the Paracale Gold Dredging Co. has worked about six acres of an average depth of 45 ft., making a total of about 450,000 cu. yd. The gold won from this acreage has been, according to Mr. Hanlon, $20,000, showing a saving of about 45 cents per yard. The extraction is estimated from 50 to 75 per cent. The property of this company, including the Melville, Fingal, and Nueva California properties, was acquired late in the year by New Zealand capitalists, who have had the ground

[1] *Eng. and Min. Journ.*, Dec. 9, 1911.
[2] *Min. and Sci. Press*, Jan. 6, 1912.
[3] *Mining and Scientific Press*, August 26, 1911.

drilled and are preparing to build additional dredges. The Malaguit River in Ambos Camarines offers some opportunities as a dredging field. The depth of the ground where the Philippine Gold Dredging Co.'s dredge is working on the river, about four miles from the sea, is from 18 to 24 ft. The dredge is a Risdon boat with 3 1/2-ft. buckets. The average amount handled per month is between 15,000 and 20,000 cu. yd., and the gold content of the ground is about 25 cents per yard. A new installation for the Islands, now under construction by the New York Engineering Co., is a 5 1/2-ft. steam-driven dredge for the Gumaus River in the Paracale district. This will be the first large modern dredge in the Islands, and its success will undoubtedly give an impetus to dredging in that country. According to the builders, the ground was prospected with Empire drills, and sufficient gold found to warrant the erection of this equipment. The dredge will cost about $150,000 complete, and will be fitted with a wooden hull, as the life of the property with the one dredge is estimated at five year.

Gold Production in the Russian Empire

By C. W. Purington

1911 *Product.*—According to the *St. Petersburg Mining Journal* "Gold and Platinum," the amount of gold dust and bullion turned into the eight Government Assay offices at Bodaibo, Irkutsk, Nikolaievsk, Blagoveshensk, Zeisky Pristan, Krasnoyarsk, Tomsk and Ekaterinburg, and representing the bulk of the product for the mines of the Russian Empire in 1911 was 2,384.75 poods (of 526.6 oz. troy). This gold is mostly in the form of dust of an average value of $18 per ounce. The total number of base ounces is therefore 1,255,809 worth $22,604,562. To this can probably be added 10 per cent. representing the gold dust which has been sold abroad, mostly at Aigun, Chefoo, and other Chinese cities and ports, by returning coolies.

Product of Districts.—The sources from which the gold comes are approximately shown by reference to the table below which shows in round figures the relative amounts recorded at the different receiving points. As an insignificant quantity comes from quartz mines, and very few of the placer miners use quicksilver in saving, or melt their gold, about 70 per cent. of the product is received in the form of dust of an average of $18 per troy ounce.

Another source of gold not accounted for by the Government Assay offices is that shipped by the Copper Mining Companies of the Ural and Caucasus in various forms of base bullion to private smelters. The total gold product may therefore be safely put at not less than $25,000,000.

ASSAY OFFICES

Bodaibo (Lena District)	$9,545,000
Irkutsk (Lena District)	3,660,000
Blagoveshensk (Upper Amur)	5,005,000
Zeiski Pristan (Upper Amur)	680,000
Nikolaievsk (Pacific Coast)	1,337,000
Krasnoyarsk (Central Siberia)	400,000
Tomsk (West Siberia)	1,390,000
Ekaterinburg (Ural, Russia)	1,256,000
	$23,273,000

The Lenskoi Gold Mining Company produces nearly one-third of the above entire Siberian output, by means of underground drift mining under exceptionally hard conditions. Nearly all the gold turned into the Bodaibo and Irkutsk Assay offices is the product of drift mining. So in a rough way it may be said that one-half the alluvial gold produced in Siberia is from drift mining. Over 80 per cent. of the remainder is produced by archaic hand methods of open-cut mining, another $1,000,000 from quartz mines, and less than $1,000,000 is produced by some sixty dredges which are operating.

Future Outlook.—Gold mining in Siberia may be described as a nearly stagnant industry. The annual product has not varied perceptibly for a period of over 20 years, nor is it likely to increase greatly for another 5 years to come. By that time some of the modern gold dredges now being erected in East Siberia may be expected to have an effect upon the product, and the completion of the railway from Irkutsk to Bodaibo—the center of the Lena district—will have stimulated production from many drift mines now idle.

Notwithstanding the small gold product of the Russian Empire at present, the potentialities of Siberia are enormous. It contains by far the largest continuous gold-bearing area in the world, and I have no hesitation in saying that Siberia is capable of sustaining an annual product larger than the present annual product of South Africa. It will certainly never produce on a large scale until its numerous deposits have been thoroughly sampled and afterward equipped with modern machinery. This will be done eventually. There are two main possibilities:

(1) *Deep Alluvial Leads.*—The so-called "Lena" gold region, comprising the district of 100,000 sq. mi. encompassed by the Olekma, Vitim and Lena Rivers is the richest alluvial gold district—considering its size—known in the world to-day. The conditions for working this field are extremely difficult at present, as is witnessed by the high cost

obtaining at the most important mines, namely, those of the Lenskoi Company. This cost for 1911, is stated to have been $6.21 per cubic yard, and when it is considered that there were 6600 men on the pay roll, this cost is not surprising. The average recovery was over $9 per cubic yard for the year; the total product being given as $8,047,000. It is not generally known that these mines have been in operation since 1863, when the Lenskoi Company was formed. From that date to the present time its production has been over $70,000,000 officially recorded, while the amount stolen and made away with during the early years in this wild region can only be surmised. Detailed reports of this year's operations are not to hand, but the following official report for 1910 of the Lenskoi Company, from a Siberian newspaper, and translated for the *Mining and Scientific Press* is typical of the work carried on through many years at this property, as conditions change very little there from year to year.

"THE LENA GOLD INDUSTRIAL COMPANY"

"The report of the Lena Gold Industrial Company for the fiscal year ended October 14, 1910, as published in the *Irkutsk Daily Sibir* of February 15, 1911, gives the following interesting facts: The property includes 43 placers of an average area of 250 acres each. The record of operations shows the handling of 0.38 cu. yd. of gravel per man per shift, the working of 1,884,710 shifts, and an aggregate of 724,573 cu. yd. of gravel washed during the year. The recovery of gold resulting from the Company's own work amounted to 765.6 poods, equivalent to 403,195.6 base ounces troy; there was recovered by tributers, 18.4 poods, or 9690.2 base ounces; gold purchased, for the most part paid for in trade, 33 poods, or 17,379.1 base ounces, making a total of gold delivered to the Government assay officer at Irkutsk of 817 poods, or 430,264.9 base ounces troy. While the average value of the gold is not given, it is commonly reported at $17.75 per ounce or $9360 per pood. Taking this average as a basis, the season's output of gold, including the small amount purchased, amounted to $7,637,200; and the average tenor recovered from 724,573 cu. yd. of material mined and sluiced was $9.87 per cubic yard. The cost per cubic yard is not given, but it has been sufficiently commented on in the reports of the Lena Goldfields, Ltd., which holds the bulk of the shares. The low duty per man, less than two-fifth of a cubic yard per shift, goes far toward accounting for the excessive cost known to have been incurred."

In 1907 an English Company was formed called the "Lena Goldfields" to take over 70 per cent. of the shares of the Lenskoi Company. After

this the shares rose rapidly, and last year were mostly bought back at St. Petersburg by the Russians, who at the last general meeting of the company, November 11, 1911, succeeded in placing two more Russian Directors on the Board.

The Lenskoi Company is obtaining the bulk of this product from the tributary of the Bodaibo river, which is a small stream about the size of the Klondike River. A glance at any good map will show that the Lena district contains hundreds of miles of gold-bearing streams, in the area above defined, less than 20 per cent. of which have been worked or even adequately prospected. A great impetus will be given to this district by the advent of the Lena Railway which is now under survey. This line will probably connect with the Trans-Siberian at Tulun, a few miles west of Irkutsk. The possibilities for the exploitation of the deep-lying partially frozen drift gravels will be thus greatly increased, and some of the river beds, such as the Patom, Thuya, and Homolko, will undoubtedly afford dredging areas, to which to-day it is practically impossible to transport machinery. This notwithstanding the fact that the areas are partially frozen. The district itself in which the Lenskoi holdings are situated has produced officially since 1844 (when the first claims were staked in the Lena district) in round figures, 27,500 poods of gold (.885 fine) or say $240,000,000. The average recovery from the pay gravel for the entire 68 years and the entire Lena district has been about $2.80 per cubic yard.

It should be said that drift mining in the Lena district is exceptionally difficult owing to the half-frozen condition of the overburden and pay gravel. The presence of water horizons between externally frozen layers of muck and clay renders a great amount of pumping necessary. The Lena Company uses station-pumps of large capacity operated by electricity generated by water-power. As a contrast the Fairbanks district of Alaska, for example, where both pay gravel and overburden are eternally frozen presents much easier conditions for deep gravel mining, as the workings are practically dry.

Steam-thawing methods have not as yet made much headway in the Lena district, the old system of thawing by wood fires being still largely in favor. As in Alaska, the pay gravel is mined all the year round stacked up in winter-dumps, and washed during the short summer.

Outside of the Lena district, no important area of deep-lying gravels workable by drifting has yet been discovered in Siberia. It is thought, however, with good reason that the boundaries of the Lena district will in time be considerably extended beyond their present workable limits.

Active prospecting with Keystone drills is now in progress in some parts of the area.

(2) *Gold Dredging.*—The old-fashioned methods of open cut mining are fast disappearing in Siberia through exhaustion of the rich shallow gravels, and with the exception of the Zeya district of East Siberia, north of the Amur, which still produces about $6,000,000 annually, by these methods, their scope is of insignificant importance. Numerous petty workings scattered over an immense area still furnish a considerable product. Many partially exhausted placers have been turned over to "starateli" or leasers, and practically ruined for future work by mechanical means.

It may be said that opportunities for gold dredging exist in all parts of the Ural and Siberia wherever unfrozen gravels have been hitherto worked by open-cut methods. But in Ural and West and Central Siberia the values are extremely low. It is safe to say the workable areas will not be found to contain an average of over 10 cents per cubic yard recoverable value. Of these low-grade gravels, however, there are enormous areas which unquestionably will be worked at a profit in the future. A notable exception to these low values in the Ural are the platiniferous gravels of the Iss and neighboring streams. This district is referred to in some detail in the article on Platinum, p. 605.

Taking up the three principal dredging districts into which I divided Siberia in the last year's review; namely: (1) The Lower Amoor field; (2) the Yenesei field; (3) the Ural field; progress in the (2) and (3) has been small. The dredges, some 35 in number, have continued at work, producing all told less than $1,500,000 in gold and handling, with the exception of those of the Nizhin Tajil Company, less than 200,000 cu. yd. per dredge for the 6 months' season. The 3 Nizhin Tajil dredges, of the New Zealand open-connected type, it is said handled over 1,000,000 cu. yd. altogether for the season; a very creditable capacity for these light dredges.

In division (3) namely the Lower Amur field, the two plants of the Orsk Goldfields Ltd. both operated successfully. The "Stackerscow"[1] plant handled 88,500 cu. yd. of gravel, in 158 working days, yielding $116,000 at a cost of 18 cents per cubic yard, leaving a working profit of over $100,000. The gold recovery was $1.30 per cubic yard. The gravel is a sticky mass of clay and finely broken angular porphyry and is washed in a series of revolving screens and a powerful log-washer. The total cost of this plant erected and running was about $90,000.

The 8-ft. electric dredge described in last year's review was successfully erected and had a trial run from October 25 to December 5, 1911. The dredge did not begin its trial until after the ice had formed

[1]A drag line bucket excavator on track dumping to hopper on a floating washingplant equipped with trommels, log-washer, gold saving tables, and belt conveyor stacker.

on the dredge pond, consequently its run of 433 hours was accomplished under difficulties. It also had to dig an undue quantity of bed-rock in working its way out of the construction pit. Nevertheless the recovery averaged 28 cents per cubic yard for 47,000 cu. yd. handled, at a working cost of 12 cents per cubic yard.

This first installation of the heavy California close-connected bucket type of dredge in the Amur region will undoubtedly be followed by more, as this type of dredge is best suited to the conditions, and during 7 months of the year a high yardage and low working cost should be maintained by such dredges in Siberia.

The Amgun Company, situated 200 miles west of Nikolaievsk, according to information kindly furnished by Mr. Schwartz, the manager, operated its 7 cu. ft., open-connected, New Zealand type steam dredge for 150 days digging 237,000 cu. yd. and recovered $78,000 or nearly 33 cents per cubic yard, at a cost of 8 cents per cubic yard, leaving $59,000 profit.

Several companies are now actively prospecting dredging ground in the Pacific Coast region of Siberia adjacent to the lower course of the Amur. The Pioneer Company of Siberia, Ltd. composed of Russian and English capitalists has recently acquired under special permits extensive properties along the tributaries of the Amur and Amjun Rivers. These holdings total over 100,000 acres of auriferous gravels suitable for the dredging industry. They are being systematically drilled under the direction of American engineers with a view to the installation of gold dredging on a large scale.

There are indications that both the Lower and Upper Amur regions, including the lower portion of the Zeya drainage, will become of immense importance in the gold-dredging industry. Practically 90 per cent. of this vast region which comprises over 300,000 sq. mi. remains to be explored. Gold-bearing slates extend over most of the area.

Hydraulic Conditions.—Outside of these two possibilities of drift mining and gold dredging, there is little promise in Siberia. Hydraulicking is possible in several isolated regions notably in the Kansk district east of the Yenesei, but none of these regions have been investigated to any extent. In *Russian Turkestan*, it is known that auriferous bench gravels exist at the base of Peak Kaufman, and in that vicinity. These gravels are similar in size and occurrence to the Tertiary bench gravels of California and are so situated that water can be got on them under pressure, and there is plenty of room for tailings. Foreign enterprise is however not encouraged in that part of the empire.

In *Northern Mongolia*, north and east of Urga, a Russian Company, the Tushetukhan and Tsetsenkhan Aimak Co. of Mongolia, generally

known as "Mongolór" is operating under the able management of Mr. von Grote a Russian engineer who has had wide experience in Alaska and the United States. Here Alaska sluicing methods are applied on a considerable scale, several thousand Russian and Mongolian laborers and tributers being engaged. The product for 1910 was about $900,000 and for 1911 is given as 118 1/2 poods, say $1,100,000. It is understood that the Company is operating at a profit, notwithstanding the fact that it has to pay a 16 per cent. royalty on the gross output to the Chinese government.

Recovery of Gold in Dredging (By Charles Janin[1]).—To determine the difference between the actual gold content of the ground and the recovery of the dredge, it is necessary to know: (1) either the actual gold content of the ground previous to dredging, or (2) the actual loss in the tailing.

In the actual sampling of a placer, each hole may be taken as indicative of the value of that portion of the ground where it is sunk, but the placing of holes is an important feature.

In estimating the value of dredging ground after determining the average value indicated by prospecting, most engineers take a percentage of this indicated value, generally 75 or 80 per cent., as the amount of gold recoverable by the dredge. These figures are based upon actual tests made to determine the dredge recovery and also upon the percentages of the prospecting value recovered from certain dredged areas.

Some engineers, in estimating the value of gravel recovered from drill holes, use a factor which they claim experience has demonstrated closely represents recovery. The use of this factor practically discounts the prospect value which would have been assumed by the use of the ordinary drill factor. Any engineers adopting such practice should, of course, mention the same in their reports; otherwise the value they claim for a property may be further discounted by those to whom the report is presented.

Another point is the difficulty of accurately sampling tailings. Some actual recoveries are cited. Lower operating costs will probably allow re-dredging of some ground at a profit.[2]

Placer Mining other than Dredging.—Hydraulic mining has had its day in most parts of the United States where extensive gravel deposits exist; not because the gravel was worked to the limit of the deposits, but because of the difficulty of disposing of the débris without injury to property, especially farm land and rivers. The great deposits of the United States are in California, and except in three counties, hydraulic

[1] *E. and M. J.*, Jan. 27, 1912.
[2] "Present-Day Problems in Dredging," the American Institute of Mining Engineers, San Francisco meeting.

mining is restricted. The La Grange in Trinity County is probably the largest hydraulic mine now regularly operated in this country. When hydraulic mining was at its zenith in California, before the days of débris restrictions, the North Bloomfield mine, in Nevada County, was the most noteworthy operation and was visited by many engineers who desired to post themselves on this method of mining.

Alaska (A. H. Brooks[1]).—In the Yentna district of the Susitna Basin, the shortage of water during the latter half of the summer cut down the production. Most of the gold output came from Cache and Peters Creeks and their tributaries. The largest production in 1911, as in 1910, was from Nugget and Willow Creeks, although the discovery of an old channel in the bench of Dollar Creek placed that stream among the important producers and gives promise of a still larger output for 1912.

Fairbanks continues to lead all the other Yukon camps in placer gold mining, though the output of 1911 was considerably less than that of the previous year. Circle was second in importance. Three hydraulic plants, all on Mastodon Creek, were installed during 1911. In addition to this, the two existing hydraulic plants on Eagle and Mammoth Creeks were operated so far as water permitted.

In the Hot Spring district the mines in the Sullivan Creek region had the largest production. Some new placer ground was discovered on American Creek, 15 miles west of Sullivan.

On Fortymile Creek a large ditch was under construction at Dome Creek.

In the summer of 1911 some placer gold was found near Red Mountain, on the headwaters of Indian River, which is in the Koyukuk River drainage basin. Considerable excitement was caused among the Yukon miners by the reported discoveries during the summer of 1911 of rich placers in the Ruby Creek region from 20 to 30 miles from the Yukon. These discoveries of placer gold were made in the headwater regions of Solatna River, which heads on the south side of the Ruby Creek divide and flows northeastward to join the Nowitna, a southerly tributary of the Yukon. The creeks reported to carry gold are Glenn and Birch, both tributary to Flint Creek, and Long Creek.

Little detailed information is available regarding the season's operations in the Iditarod-Innoko region.

Arizona.—Along the Hassayampa River, 45 miles northeast of Phoenix, dry placer machines have been worked. T. L. Carter describes these diggings in *E. and M. J.*, March 18, 1911. The method to obtain water is suggested by impounding in dams and using it over and over.

Hydraulic Mining in California.—Hydraulic mining is in operation

[1] Abstract *E. and M. J.*, Jan. 6, 1912.

on an extensive scale in the western end of Sierra County at Brandy City and Scales, the two companies operating at these points both having complied with the requirements of the anti-débris regulations. The Brandy City Company employed on an average of 40 men for over four years building miles of flume, immense retaining dams, pipe lines, reservoirs and a hydro-electric plant. The Neocene Company at Scales has been successfully operating for seveal years and is one of the largest hydraulic mines in the State. On the opposite side of the Yuba River from Brandy City is located the Snowden Hill gravel mine which has been a rich producer. On what is supposed to be an extension of this same channel, near Sleighville House, the Orient mine has recently developed rich pay gravel, the development work showing a large body of river gravel, supposedly a former channel of the Yuba river.

In northern Sierra County, near the Plumas County line, at Poker Flat, La Porte, Port Wine, Deadwood, Howland Flat, Round Lake, Gibsonville and at Salmon Lakes much development work has been done. At Haskell Peak very rich gravel has lately been uncovered and is now being washed.

There are in California many hundreds of acres of shallow placer lands that cannot be worked by the large modern dredge. These lands cannot be hydraulicked as the Caminetti law prevents the uncontrolled discharge of tailings. To fulfill the requirements, the Hearns-Martin placer mining machine is used near Oroville at Thompson flat. The surface has sufficient pitch for water used in the process to flow back to the supply reservoir without too much loss. A Marion steam shovel, model 35, of 1 cu. yd. capacity, delivers one bucket per minute to the hopper of the washing machine. Any digging apparatus could be used and would be applicable to any surface however steep. The washing apparatus consists of a portable hopper, mixing drum, revolving screen, together with chutes and riffles. It would appear that the plant is entirely successful and that it points the way for working this class of deposits successfully.[1]

Another plan is proposed by The Tarr Mining Company at Smartsville.

The gravel bank will be hydraulicked to a sump in front of a stationary dredge building of concrete and sheet iron, where a regular steel-girder dredge ladder, equipped with 52 seven-foot buckets elevates the gravel to a trommel 45×6 ft., with 1/2-in holes. From the screen the undersize flows to gold-saving tables with Hungarian riffles having an approximate area of 4600 sq. ft. The oversize passes to a belt conveyor 570 ft. long, in two sections, each operated by a 50-h.p. motor. A 100-h. p. motor

[1] *E. and M. J.*, Dec. 16, 1911.

is used on the digging ladder, and a 30-h.p. motor on the trommel. At the end of the belt-conveyor stacker two Bleichert tramways are being constructed to afford a larger dumping ground for the tailings.

The fine material, after passing the gold-saving tables, goes through a tunnel about half a mile long, and is elevated to revolving screens, the oversize going to the tailings dump, the undersize to Overstrom tables to save the black sand, with any accompanying gold and platinum. The concentrator stands several hundred feet from the Yuba River, and a concrete dam will be constructed to give a settling basin for tailings.

Where the limitations of the dredge are known, as in ground with heavy boulders or where cleaning up is required chiefly on bed-rock, a method has been used consisting of a box hydraulic elevator. This is an open inclined chute, usually at an angle of 15 degrees and long enough to elevate over a bank of 20 ft. The usual sluice has less capacity than this scheme which is in use on the Salmon river, although it originated in Oregon; and it is owing to the large capacity that it gives low costs per yard; six to seven cents for total operating cost.

Colorado (by Arthur Lakes).—A full description is given of the localities and methods of working near Breckenridge, Summit County.[1] The gold gravel in the vicinity of Blue River has run the gamut through gold pan, rocker, sluices, high-line ditches, hydraulic nozzles, elevators, booming, dredging and many experimental machines that have been found wanting.

Finally powerful bucket dredges were built and these, both on the Swan and Blue Rivers and on French Creek, have proved eminently successful. But some are still working with giant nozzles and others are contemplating resuming business, the ground being far from worked out.

Oregon.—Many efforts have been made to save values in beach sand along the shores of Oregon, which contain platinum, besides gold. The concentrator near Newport promises to do good work. A drag-line scraper hauls about one ton at a time up an incline to the mill; a canvas and burlap table extracts the gold by amalgamation, the platinum being then caught on a moving belt of rough material.

Canada.—One of the interesting placer grounds is that of Quebec where in 1835, gold was discovered on the Chaudiere River, Beauce County. In 1910 the ground was carefully sampled.[2] Quartz lodes in the neighboring slates do not appear to carry any gold. On the Plantes River the gold is of a flaky form showing that it has been carried for long distances. Years ago the presence of pipe clay and quicksand caused considerable trouble in underground mining, shown by shafts that

[1] *Mines and Minerals*, Feb., 1912.
[2] Fritz Cirkel, *E. and M. J.*, Nov. 25, 1911.

had to be sunk through 70 feet of bed. Of the $2,000,000 of gold produced in Beauce County $1,000,000 has been credited to the Gilbert River, according of the Government reports.

On the Ruisseau des Meules a miner took out $8000 in one season by open casting and drifting, with crude appliances. During August, 1911 a clean-up was made by the Champs d'Or Rigaud Vaudreuil Co. Ltd. from the tailings of these same diggings. About 7000 cu. yd. of the washed ground, subjected to hydraulicking, yielded 36 cents per cubic yard. The original ground, therefore, figures $1.50 per cubic yard. Formerly underground work was attempted even for depths of 20 ft., the ordinary sluice box and pick and shovel being the only appliances. Recently three satisfactory clean-ups have been made without use of block riffles.

Many of these placer deposits are too flat or have insufficient water for successful hydraulic mining and many contain boulders too large or the bed-rock is too hard and rough for dredging. At this property a tailings elevator is used to overcome these difficulties. Gravel is hydraulicked in the usual manner by a ground sluice to a sump. The elevator buckets pick up and dump this material into a sluice at sufficient height (40 degrees) above bed-rock to provide ample dump for tailings. The whole elevator apparatus is mounted on a movable car. The elevator buckets have a capacity of 1 1/2 cu. ft. and weigh 550 lb. To obtain water for the hydraulic nozzles at a sufficient head to disintegrate the gravel deposit, Lac Fortin, a body of water three-quarters of a mile wide and one and one-half miles long, was decided on, although it was about seven and one-half miles distant. From this lake 32,000 feet of ditch, 4500 ft. of flume, and 2400 ft. of pipe line were constructed. The ditch, which in section, is 3 1/2 ft. wide at the bottom, 5 ft. wide at the top, and 3 ft. high. The flume is 4 ft. ×4 ft.

British Colnmbia.—A most important hydraulic enterprise is the Quesnelle Hydraulic Gold Mining Co., financed by Philadelphia and New York capitalists. Through H. W. DuBois, of Philadelphia, who investigated platinum occurrence in the district, water rights on the Swift River were acquired from the Government. At least 15,000 miners inches are available. Forty miles of wagon road have been constructed to connect with Government roads. The undertaking will involve an expenditure of $1,000,000. The company's hydraulic leases cover about 800 acres of gravel bed on the lower part of Twenty-mile Creek at its junction with the Quesnelle River, where the new town of Hydraulic is situated. The water-supply area is 200 square miles and well wooded. Construction of about 13 miles of the water system has been completed since 1910. Of this about 10,000 ft. is 60 in. spruce wood stave pipe, spruce staves being

cut on the ground. It is used for crossing depressions of 80 to 110 ft. The flumes are 5 ft. wide by 4 ft., but there are only 2000 ft. of this, the remainder being lake and ditch. It is expected to move 12,000 cu. yd., with 3500 miner's inches per day. The double sluiceway, below the pits, will be 600 ft. long on a 5 per cent. grade, each sluice being 6x4 ft. The sluice is double to avoid delays, and it will be paved in the upper part with cotton-wood blocks, which have been found serviceable in this district heretofore. The lower part will be paved with stone. In the Atlin district of British Columbia, steel lined sluices have been used successfully. The sluices will be short, and the high carbon steel plates will be put in next season.[1]

Yukon.—Consul G. C. Cole, Dawson, Yukon Territory, Canada. reports a new gold discovery, regarded as one of the best since the days of the Klondike discovery in 1898, was made in December 1911, about 70 miles from the mouth of Sixty-mile River, 120 miles southwest of Dawson, and 25 miles from the Alaska line. The find is said to average about $1 to the square foot of bed-rock.

The discoverer reports the formations in sinking a shaft 19 1/2 ft. to bed-rock as follows: Six inches of muck; 6 ft. of gravel containing gold, three cents to the pan; 6 ft. of muck and 7 ft. of gravel to bed-rock where the gold is found. In cleaning up the surface of the bed-rock, a space of 5 by 8 ft., he obtained $40 worth of high-grade gold.

The working gravel is similar to that of the famous Bonanza Creek of the Klondike, and frozen solid from the surface to bed-rock, making a safe roof for mining. Yukon territory has produced $150,000,000 worth of gold. For further comments on the Yukon see p. 789.

[1] E. Jacobs, *E. and M. J.*, Sept. 23, 1910.

PROGRESS IN ORE DRESSING AND COAL WASHING IN 1911

By Robert H. Richards and Charles E. Locke

Crushing and Grinding Apparatus

Largest Gyratory Breaker in the World.[1]—The Gates breaker No. 24, built specially for the Biwabik iron mine, Biwabik, Minn., stands 28 ft. high and weighs over 500,000 lb. The two receiving openings are each 48×108 in. and will take 8-ton lumps of rock. The capacity is estimated at 1000 to 1200 tons per hour crushed through a 5-in. ring. It is driven by a 200 horse-power induction motor.

Calumet Jaw Breaker.[2]—For crushing Lake Superior native copper ore, jaw breakers are considered to be superior to gyratory breakers. The latest development in the Calumet and Hecla mills is a Blake breaker with jaw opening 24×48 in. The total weight is 40 tons. The frame is cast in one piece and has reinforcing bolts. The toggles are made of a special shape so as to obtain pure rolling motion at their ends. The angle of the face of the jaw is much less than in most breakers and consequently there is less clogging, less breakage and increased capacity. The movable jaw has a travel of 1 in. The weight of the pitman is taken from the bearings by suspending it by springs from a yoke bolted to the frame of the breaker. Smooth jaw plates are used and give higher capacity than corrugated plates. Manganese steel plates are used for conglomerate while chilled steel is more satisfactory for amygdaloid ore.

Symons Breaker.[3]—This breaker is of the fixed pillar type (figure 1) and has a hollow eccentric driving shaft *B* which fits like a sleeve over the central fixed pillar *A*. Over the hollow shaft *B* and attached to the crushing head is a casing *D* which is flanged at the bottom below the crushing head which flange covers the casing of the bevel wheel *C*. When in action the shaft *B* revolves inside the casing *D* which thus receives a gyratory motion. This breaker has equal movement on the large and small lumps.

Symons Disc Crusher.[4]—The principle of this machine is that of two shafts whose center lines lie in the same horizontal plane but in separate

[1] *Engineering and Mining Journal*, Vol. XCI (1911), p. 367; C. A. Tupper, *Ibid.*, Vol. XCII (1911), p. 1043; *Mines and Minerals*, Vol. XXXI (1911), p. 432.
[2] *Engineering and Mining Journal*, Vol. XCII (1911), p. 1026.
[3] B. Shubart, A. H. Reed, *Engineering and Mining Journal*, Vol. XCI (1911), pp. 307, 316, 602.
[4] *Mining Science*, Vol. LXIII (1911), p. 164; *Mines and Methods*, Vol. II (1911), p. 138; A. H. Reed, *Engineering and Mining Journal*, Vol. XCII (1911), p. 157.

vertical planes that intersect. The hollow shaft *A* (figure 2) is supported by fixed bearings *X*. This shaft is enlarged at one end and threaded and has screwed on it a cap *L* that has fastened concentrically into it one of the crushing discs *C*. This cap is split and has clamp bolts by means of which it can be locked in any set position and thus the size of the crushed product adjusted. In operation the cap revolves with the hollow shaft *A*. The tapered solid shaft *B* is also enlarged at the disc end into a half ball-and-socket bearing, constituted by the inner and outer shafts, to which enlargement is fastened concentrically the other

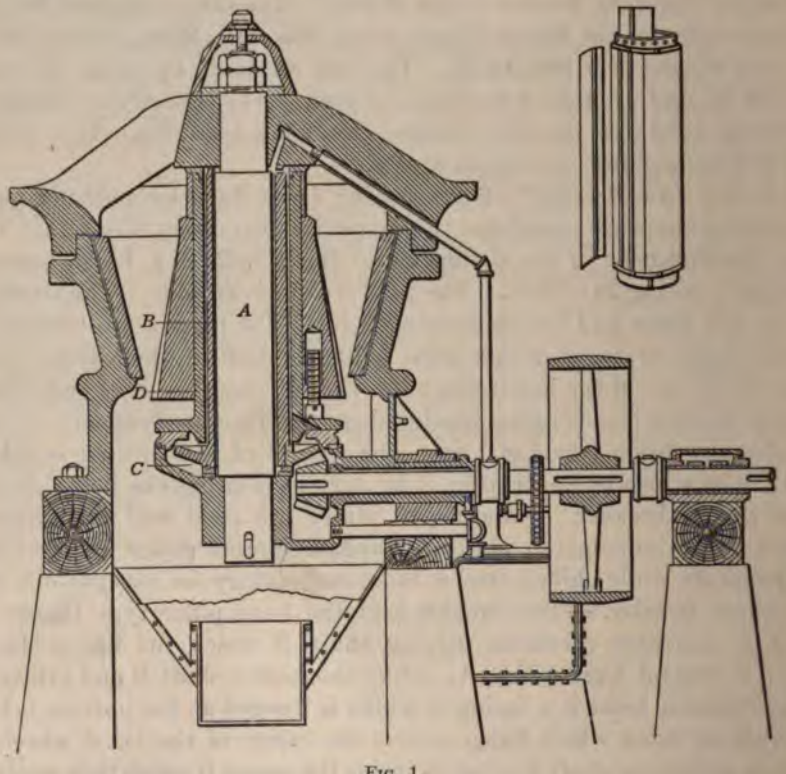

FIG. 1.

disc *D*. The other and small end of the solid shaft extends outside the hollow shaft and is carried on an independent bearing *Y* which is adjustable in a horizontal plane to regulate the throw of the crushing discs. A hood *M* carries the feed tube *S* delivering the ore between the discs. Discharge is at the periphery of the discs into the bonnet and outlet *T*. The crushing discs have smooth surfaces and constitute two dinner plates, so to speak, with their concave surfaces facing. When no ore is

fed only the outer shaft revolves, but the feeding of ore causes both shafts to revolve and in the same direction. The action of the shafts is such as to cause the space between the discs to be pinched on one side and open on the other and a piece of ore is crushed at the periphery of the discs while it passes through a semicircumference from the open side around to the pinched side. Centrifugal force tends to throw the rock to the periphery of the discs and to discharge it when crushed sufficiently fine. The machine crushes dry or with water. There are three sizes: 48 in. taking 7-in. feed and reducing to any size down to 5/8 in., capacity 50 to 90 tons per hour to 1 in.; 24-in., taking 3 1/2 or 4-in. feed and

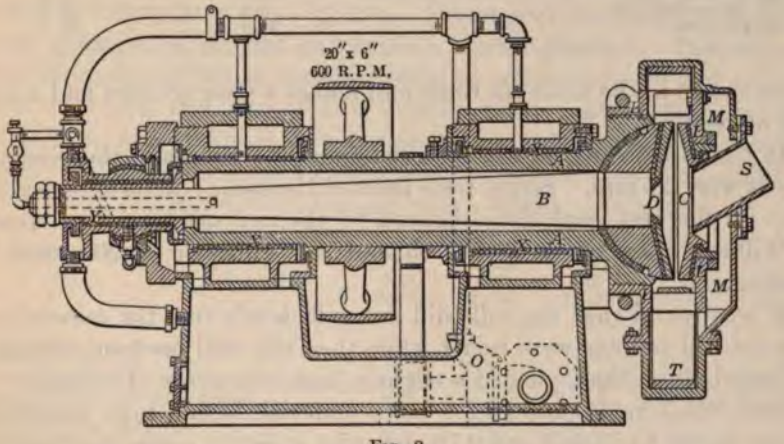

FIG. 2.

reducing as fine as 1/4 in., capacity 20 tons per hour to 3/4 in.; 13 in., taking 2-in. feed and reducing to 1/8 in. and smaller, capacity 7 tons per hour to 1/4 in.

Rolls at the Miami Mill.[1]—In the original Miami mill to treat finely disseminated copper ore, Chile mills were installed for regrinding in two sections and rolls in the third section. The result of the experiment has been to show that Chile mills give the best results because in this case the granular product obtained from the rolls contains too much mineral in the form of included grains. In other words the sliming action of the Chile mills was desirable in order to free the fine mineral.

Stamps and Tube Mills.—The reader is referred to the article on Progress in Gold Milling elsewhere in this volume, pages 358 to 377.

Chile Mills Versus High-speed Rolls at Morenci, Arizona.[2]—Tests to determine the relative amounts of slime produced and copper recovered

[1] *Engineering and Mining Journal*, Vol. XCII (1911), p. 289.
[2] R. Gahl, *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 293; *Mining Magazine*, Vol. V (1911), p. 70; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XII (1911), p. 143.

in regrinding jig tailings have been made at the Detroit mill using a 5-ft. Chile mill, run at 136 r. p. m. and rolls 36 in. diameter, 15 in. face, running at 140 r. p. m. The rolls product was screened by a trommel and the oversize returned continually to the rolls. In the first test the feed was through 7 on 5 mm. round-hole screens and the products of both rolls and Chile mill were passed through 2.5 mm. screens.

	Rolls. Per cent.	Chile Mill. Per cent.
On 20 mesh	23.1	11.0
Through 20 on 30 mesh	16.0	7.0
Through 30 on 100 mesh	23.4	21.2
Through 100 on 200 mesh	6.6	9.3
Through 200 mesh	30.9	51.5

As shown in the table the Chile mill makes a finer product and a great deal more slime.

In test No. 2 the feed was between 5 and 2.5 mm., and the discharge screens were 2.5 mm. Sizing tests showed the same results as in the first test. In this test, each size produced by the two machines was treated on Wilfley tables up to 200 mesh and the undersize of 200 mesh by panning.

It was found that the rolls did not sufficiently free the mineral and that the roll tailings were richer while the Chile mill product, though it contained more slime, yielded a slightly higher recovery of copper.

Test No. 3 made on 5 to 2.5 mm., material, with 1 mm. round-hole screens on the Chile mill and 0.79 mm. wire screen in connection with the rolls showed practically the same recovery of copper by each, the rolls losing more in the sands and the Chile mill in the slimes.

If no further facts in connection with the general concentration problem are taken into consideration, the Chile mill would be the best machine to use because it is simpler and easier to keep in efficient working order. If, however, any subsequent regrinding of the coarse pulp from the Chile mill or rolls is contemplated, the decision in favor of the Chile mill is not necessarily final because the coarse product from the rolls contains more copper that can be freed by further regrinding.

Test No. 4 was made on 5 to 2.5 mm. material, and 0.79 mm. screen for the rolls, but the mill had a 3 mm. slotted screen and the pulp passing through this screen was delivered to a 0.79-mm. screen sending oversize back to the Chile mill. This test showed that with the Chile mill the proportion of slime can be reduced to approximately the same as that produced by rolls and thus the Chile mill would have the same advantage as the rolls from the viewpoint of slime production and would still retain the other advantages already possessed by it.

Chile Mills in Russia.[1]—Gold ores in Russia are largely crushed and amalgamated in slow-speed Chile mills which have been developed in successive steps from the original crude method of spreading the ore over the main thoroughfares for carts and other heavy vehicles to pass over. In the Kotchkar district the general practice is shown in Table 1. The roller tires are of cast steel 5.5 to 6.5 ft. diameter, 7 to 12 in. wide and 9 to 10 in. thick. The annular die is made of 6 to 8 sections 3 to 3.5 in. thick, held in place by wooden wedges. Most of the mills are hand fed with ore broken to any size up to 4-in. ring. The fineness of the product is shown in Table 2. Increased capacity may be obtained by increasing the diameter of the mills and the weight of the rollers or by increasing the number of rollers from two to three. Increase of diameter causes more violent agitation in the pan and hence a coarser product. The mills are splendid amalgamators at the slow speed at which they are run. Inside plates are used and also 2 or 3 outside plates each 4 ft. 8 in. long by 2 ft. 4 in. wide followed by riffles and traps. Six to 8 lb. of mercury are added to each mill per day at intervals, 2 to 3 lb. of this being added an hour before the wash-out or clean-up which comes once a day.

TABLE I.—DETAILS OF CHILE MILLS.

	Mill No. 1.	Mill No. 2.	Mill No. 3.	Mill No. 4.	Mill No. 5.
No. of Chile mills in plant	6	4	6	4	6
No. of rollers per mill	2	2	2	2	2
Diameter of pan across bottom	7 ft. 6 in.	8 ft. 9 in.	8 ft. 2 in.	10 ft. 6 in.	9 ft. 4 in.
Distance between centers of rollers	6 ft.	6 ft. 9 in.	6 ft. 6 in.	8 ft. 3 in.	7 ft. 6 in.
Width of face of roller, inches	10	10	10.5	10	12
Diameter of roller, feet	5.5	5.5	6.5	5.5	6.5
Weight of roller, pounds	5400	5400	5400	5400	10800
Revolutions of mill per minute	12	14	13.5	12	11
Mean travel of roller per minute, feet	226	297	275.5	311	260
Height of discharge, inches	10	10	9.75	9.5	8
Screen used, mm	1 S	1 S	1/4–1/8 P	1 S	1/2 S
Water to ore ratio by weight	10 : 1	6.5 : 1	7 : 1	13 : 1	.....
Duty per 24 hours, tons	16	26	20	19	14
Amalgam from mills	86 %	81 %	83 %	85 %	.....
Amalgam from outside plates	13 %	18 %	16 %	15 %	.....
Amalgam from sundry sources	1 %	1 %	1 %	.....	.....

S = slotted, P = punched.

TABLE II.—SIZING TESTS OF PRODUCTS.

	Mill No. 1. Per cent.	Mill No. 2. Per cent.	Mill No. 3. Per cent.	Mill No. 4. Per cent.
On 30 mesh	0.2	0.4	0.2	0.1
Through 30 on 60 mesh	2.41	14.84	3.2	1.3
Through 60 on 100 mesh	14.72	22.06	12.80	11.0
Through 100 mesh	82.67	62.70	84.80	87.6

Plates are removed, steamed and scraped once a month.

An improved form of Chile mill and automatic belt feeder has been

[1] H. C. Bayldon, *Institution Mining and Metallurgy*, Vol. XX (1911), p. 125; *Mining Science*, Vol. LXII (1911), pp. 154, 181, 613; *Mining World*, Vol. XXXIV (1911), p. 8; *Ibid.*, Vol. XXXV (1911), p. 1023; *Mining Magazine*, Vol. IV (1911), p. 62; *Mines and Minerals*, Vol. XXXII (1911), pp. 108, 177; *American Metallurgical Society*, Vol. I (1911), p. 58.

installed by the author. The head pulley of this feeder is operated by a ratchet or friction drive device similar to that used on the feed plate on the Hendy feeder. Each time the mill revolves a little inclined plate revolving with the mill supported on a framework extending out over one of the roller shafts, strikes and lifts one end of a horizontal lever arm about 3 in., which in turn gives an impulse to the head pulley of the belt feeder. After the inclined plate has passed the lever arm descends by its own weight to its original position. The level of the inclined plate is governed automatically by the amount of ore on the die, or in other words, as the rollers rise they operate through a lever to lower the level of the inclined plate and thus decrease the rate of feed. This feeder would seem to be applicable to slow-speed Chile mills wherever used.

The author's improved Chile mill has a narrow pan at the die, but the width increases rapidly above, owing to the sloping of the inside cone. The dimensions are as follows: diameter of pan, 10 ft. 3 in.; distance between center of rollers, 8.5 ft.; revolutions of mill per minute, 14 and 16; mean roller travel per minute, 340 and 427 ft.; diameter of roller, 6 ft.; face of roller, 1 ft.; weight of each roller, 10,000 lb.; weight of tire alone, 6040 lb.; weight of set of dies, 4880 lb.; size of discharge opening, 44×8 in.; gross weight of mill, 23 tons. The feed is hard quartz considered to be comparable to South African banket ore and the size is 5.7 per cent. over 3 in., 42.5 per cent. between 3 and 1.5 in., 27.6 per cent. between 1.5 and 3/4 in., and 24.2 per cent. finer than 3/4 in. Running at 14 r. p. m., 14.7 horse-power is required for an output of 36 tons in 24 hours, 16.27 horse-power for 39 tons, 17.32 horse-power for 42 tons, and 23.62 horse-power for 51 tons. It is believed that the mills can be run at 20 r. p. m. with advantage. Using a 0.5-mm., slotted screen, the product contains 0.17 per cent. on 30 mesh, 5.20 per cent. on 60 mesh, 16.37 per cent. on 100 mesh, and 78.26 per cent. through 100 mesh. Using a 1.5-mm. slotted screen these figures become 0.70, 8.72, 21.20, and 69.38 per cent. respectively. Running at 14 r. p. m. experience has shown that a height of discharge of 9 to 10 in. and a water ratio of from 4.5 : 1 to 5.5 : 1 with 1-mm. screens gives the best results.

The mechanical efficiency is figured by Stadler's method as 63.97 per cent. Two outside plates each 4×10 ft. are used for each mill. Compared to the No. 2 mill of Table 1 the results are as follows:

	No. 2 Mill	New Plant
Time of clean-up for each mill	20. min.	15. min.
Gold recovered from amalgamation	58.74 per cent.	61.3 per cent.
Mercury losses per ton crushed	0.69 oz.	0.36 oz.
Amalgam recovered from mill	81 per cent.	73.9 per cent.
Amalgam recovered from plates	18 per cent.	24.6 per cent.
Amalgam recovered from sundry sources	1 per cent.	1.5 per cent.

The net wear of tires is from 0.62 to 0.65 lb. per ton crushed and of dies 0.28 lb. When worn out the discarded weight will probably be from 9 to 10 per cent. in the case of the tires and 35 per cent. in that of the dies. The wrought-iron outside liners last 7 months. Slotted screens last about 1200 tons.

The conclusions arrived at from running this plant are as follows: The mill is extremely simple, requiring no skilled supervision and very little attention and is almost fool proof. Temporary failure of the feeder has no other effect than to reduce the capacity and horse-power. The product is uniform and its mechanical efficiency as compared with stamps doing the same work is much higher. The amount of water required is low. The most suitable capacity at 14 r. p. m. is 42 tons per 24 hours. At 51 tons the mill is overtaxed. A throw of 3 in. for the belt feeder is best. For a higher tonnage at 14 r. p. m. a wider belt should be used. At 16 r. p. m. the capacity could be 50 tons and at 20 r. p. m. 60 to 65 tons. Changing from two rollers to three would increase the capacity 50 per cent. The mill compares favorably with stamps on a tonnage basis in respect to space occupied and the initial cost and cost of erection is less.

This paper has led to considerable discussion over the work and field for Chile mills. In Rhodesia sixteen Chile mills running at higher speeds (38 to 45 r. p. m.) and of a type represented by the Akron Chile mill (see Richard's "Ore Dressing," Vol. III, p. 1307), are used for crushing gold ores, being fed with 1 1/4-in. material from stamps and discharging a product which will nearly all pass 40 mesh and 40 to 45 per cent. will pass 150 mesh.

Slow-speed Chile Mill as Applied to Mexican Practice.[1]—The claim is made that the stamp-tube mill combination may be advantageously replaced by rolls and slow-speed Chile mills. Rolls have less cost and require less power and are satisfactory when crushing to only ¼ or ⅜ in. The slow-speed Chile mill is efficient as a fine grinder and the only possible objection to it is its great weight and high first cost.

It probably cannot compete with tube mills where the grinding is comparatively coarse—30 to 40 mesh. An example is given of ten 1250-lb. stamps and two 5×16-ft. tube mills, and requiring 1.19 horse-power for each ton ground to 200 mesh. It is estimated that with slow-speed Chile mills the power consumption would be only 0.87 horse-power per ton ground to 200 mesh.

Advantages claimed for slow-speed Chile mills as compared with stamps are: (1) Less height required; (2) cheaper foundations; (3) fewer repair

[1] J. B. Empson, *Informes y Memorias del Instituto Mexicano de Minas y Metalurgia*, Vol. II (1911), pp. 173, 202.

parts required; (4) less break-downs and delays; (5) overfeeding or underfeeding does not hurt the mill; (6) absence of all screens in some cases; (7) absolutely unskilled labor can operate it; (8) attendance very small if mechanical feeders are used; (9) repairs to nearly every part can be made on the ground; (10) minimum vibration and hence minimum depreciation of mill.

Some improvements in design of slow-speed Chile mills are desirable as follows: (1) Improvement in method of applying power; (2) change in form of pan; (3) trunnion shaft to turn in center bearing where grit cannot get in; (4) device for allowing mullers to lift independently over any unbreakable object; (5) improved method of feeding; (6) more screen discharge area; (7) adjustable chuck blocks; (8) increased use of the Mantey offset.

Crushing with Slow Chile Mill.[1]—At the Susannah mine at Golden, Utah, a 10-ft. slow-speed Chile mill working on hard oxidized quartz ore crushes 20 tons in 16 hours so that 75 per cent. of the product passes a 100-mesh screen. The feed is 1-in. lumps from a Dodge breaker. The gold extraction is 85 per cent. in the mill and on the outside plates. Water is settled and repumped and the loss is only 180 gallons per ton of ore.

At Amboy, California, a hard, close-grained gold quartz ore containing hard hematite and a little copper from the Great Gold Belt mines gave an extraction of 60 per cent. of the gold by amalgamation in laboratory tests. A mill was installed consisting of grizzly with 1-in. spaces, 7×10-in. Dodge breaker, bin, plunger feeder, bucket elevator, 35-ton bin, plunger feeder, 7-ft. Lane slow speed Chile mill, 5×8-ft. amalgamating plates, Pierce amalgamator, and settling tanks to recover water to be used again. A Traylor concentrating table was originally installed but was abandoned as unprofitable on account of the small amount of water available, and low value of the concentrates.

The mill treats 20 to 30 tons in 24 hours and the extraction by amalgamation is 88 per cent. The tailings contain 85.79 per cent. finer than 100 mesh. The cost is $1.50 per ton which is not considered high when the small size of the plant and the distance from the railroad are considered. One man per shift runs the mill.

Huntington Mills in Nicaragua.[2]—For grinding gold ores in Eastern Nicaragua a number of 3.5-ft. sectional and 5-ft. solid Huntington mills are used. The ores are soft and oxidize and contain considerable clay from kaolinized feldspar.

The wear is high, 0.75 to 1 lb. per ton ground. Some of the large

[1] A. McLaren, *Engineering and Mining Journal*, Vol. XCII (1911), p. 305.
[2] C. C. Semple, *Bulletin American Institute Mining Engineers*, No. 58, October (1911), p. 799; *Mining World*, Vol. XXXV (1911), p. 1165.

castings have to be scrapped when worn in certain small areas while the rest of the casting is in good condition. The total costs of milling were 60 cents per ton of which 26 cents was for power and 9 cents for repair parts. Of the cost of renewals, 50 per cent. was for roller tires, 25 per cent. for ring dies, 20 per cent. for parts requiring frequent renewal and 5 per cent. for such parts as were renewed only at long intervals such as housings, shafts, and gears. Ring dies last 3 months and wear evenly down to 0.25 in. at the thinnest part. Roller tires of chrome steel last 5 weeks.

Huntington mills are well adapted for grinding soft clayey ore. For regrinding jig middlings, however, the great wear makes it inferior to Chile mills even though the latter make less slime.

Breaking of Asbestos Bearing Rock.[1]—This operation has two objects; first, the separating of the asbestos veins from the rock; second, the opening up of the asbestos fibers without breaking them, so that they will be sucked away by an air current. The rock particles should not be ground to dust in the operation lest they should be sucked away as well. Any machine using grinding is inapplicable to this work. For coarse crushing, down to between 1.5 and 3 in. jaw breakers are used. Anything finer than this goes to Cyclone impact pulverizers which have been shown by experience to yield the best product although the machines themselves are mechanically defective owing to the high speed and the danger of breakage in case a piece of steel or very hard piece of rock finds its way between the breakers. However, the impact method has the advantage that it makes a clean break and less fines while pressure causes more fines by the attrition of the rock between the jaws of the breaker and also by the movement of the pieces against one another in the mass during crushing. Moreover, in impact machines the smaller the particle becomes the less energy it has and the less liable it is to be broken. This is seen to be true when it is considered how futile it would be to attempt to crush fine dust by any form of impact machine, and it shows how asbestos fibers once they are opened up are not in danger of being broken by impact.

A new form of impact machine has been tried for this work and is claimed to be superior to Cyclone in point of wear, power consumption, less attrition of the rock, less dust, better grade and higher yield of asbestos fibers. The principle of this machine is that of imparting the velocity of impact entirely to the rock stream which is intermittent so that one rock particle is not impacted on the preceding. Multiple impacts are secured by successive mountings on the same shaft.

[1] E. Torrey, *Bulletin Canadian Mining Institute*, No. 13, February, 1911, p. 71.

Theory of Crushing

Comparative Efficiencies of Crushing Plants.[1]—Based on Stadler's figures of mechanical efficiency in crushing, the stamp-tube mill combination has a combined efficiency of 37.3 per cent., but the stamps at the coarse end have 78.3 per cent. efficiency while the tube mill has 20.07 per cent. efficiency. Fine stamps have 33 to 39 per cent. efficiency. Analysis of Stadler's results leads to the question, at what size does the stamp cease to be economical as a fine grinder, and concurrently at what stage does the stamp-tube mill combination become superior to the single stage operation? Furthermore, if account is taken of the extra cost of installation of the stamp-tube mill combination is taken into account, will the cost of this two-stage crushing be any less than that of single stage with fine stamps? The advocates of single stage crushing on the Rand point as an argument to a certain mill using single stage crushing where the cost is below that of the mills using stamps and tube mills and the extraction is at least equal.

If the stamp mill is to be limited to coarse crushing work, is it not likely to be supplanted by some other form of crushing machine? Mr. Denny believes that rolls, Chile mills and ball mills are bound to supplant the stamp to a greater or less extent as time goes on.

Efficiency of Fine Grinding Machinery.[2]—Applying the Stadler method of calculating grinding efficiencies and using published data from various sources the following results have been obtained:

1. East Rand; ten 1634-lb. stamps, one tube mill, 261 tons per 24 hours, 0.284-in. battery screen, fines by-passed the stamps to tube mill, horse-power 150, 99.5 per cent. through 90 mesh; relative mechanical efficiency per horse-power, 46.98 per cent.
2. East Rand; eighty 1680-lb. stamps, eight tube mills, 1600 tons per 24 hours, battery screen 8 meshes per square inch, horse-power 1200, 90 per cent. of product through 90 mesh; relative mechanical efficiency per horse-power, 35.20 per cent.
3. East Rand; same as (2) except tonnage 1840 tons; 40.48 per cent. efficiency.
4. Union mill, Pachuca; slow heavy Chile mills, 16.53 tons per mill per 24 hours, horse-power 13, 80 per cent. through 200 mesh; relative mechanical efficiency per horse-power, 36.40 per cent.
5. Union Mill, Pachuca; like (4) except tonnage 22.04 tons per 24 hours; efficiency 46.13 per cent.
6. Santa Elena mill, San Luis de la Paz, Guanajuato; slow heavy Chile mills, 27.55 tons per 24 hours, horse-power 15, 67.30 per cent. through 200 mesh; efficiency 49.09 per cent.
7. El Bote mill, Zacatecas; slow heavy Chile mills like (4), 18.60 tons per 24 hours, horse-power 13, 70 per cent. through 200 mesh; efficiency 38.77 per cent.
8. South Dakota; fast Chile mills in series with rolls, 105.6 tons per combined unit per 24 hours, horse-power 40 to 45 per combined unit, 39.40 per cent. through 200 mesh; relative mechanical efficiency, 55.73 per cent.

The highest efficiency is that of the fast Chile mill in series with rolls but the ground product is too coarse for cyanide agitation. The slow-speed Chile mills grind fine and have efficiencies which are equalled only by the best Rand practice using stamps and tube mills. Furthermore, the slow Chile mills are simple, easy to run, require little attendance, and have low cost of repairs.

Commenting on the foregoing Mr. Megraw states that the advantages of the slow-speed Chile mill, as a primary crushing machine, are not fully

[1] G. A. Denny, *Mexican Mining Journal*, June (1911), p. 23; *Mines and Methods*, Vol. II (1911), p. 246; *American Metallurgical Society*, Vol. I (1911), p. 29.
[2] W. H. Urbiter, *Engineering and Mining Journal*, Vol. XCII (1911), p. 257.

appreciated.[1] It produces a fine product, can be run steadily for lengths of time that can never be approached by stamps, and the items of power and expense are much less than with either stamps or fast Chile mills. It will not regrind, however, because sand returned for regrinding will pack on the ring like a macadam road. The mill is most efficient when a large quantity of water is used to take away at once the material which is fine enough to get through the screen.

Fine Grinding.[2]—In the present days of fine grinding it becomes a question of cost and extraction just what method to pursue. Mr. Denny believes that for gold ores that the grinding should be carried fine enough so as to allow the same cyanide treatment to be used on all the pulp instead of having separate processes for concentrates, sand and slimes. Depending on the nature of the ore we may have the following processes: (1) Fine grinding of all the ore; (2) fine grinding of only the sands where there are no concentrates; (3) fine grinding of only the concentrates; (4) fine grinding of sands and concentrates. He advocates that amalgamation be discarded and states that coarse gold will be effectually recovered by tube mills crushing in cyanide solution. One scheme suggested for handling an ore containing coarse gold and concentrates which require longer treatment than the silicious material is to discharge the tube mill pulp to concentrating tables which make concentrates to go back to the tube mill and tailings to be cyanided. The concentrates will remain in the circuit until they are slimed so as to pass into the table tailings and go to the cyanide treatment. In this scheme all the concentrating work is done in cyanide solution.

Screening, Classifying and Settling

Screenless Sizing of Ore on McKesson-Rice Table.[3]—This new invention resembles somewhat in appearance a Wilfley table having its table top set with a side slope of 38 degrees, fed at the high corner at the mechanism end and discharging dry sized products on the opposite end and the low side. The table top is 15 ft. long, 6 ft. wide at the back or mechanism end, and 4.5 ft. wide at the discharge end. It is given a longitudinal throw and is supported on four inclined toggles so adjusted as to raise the table top on the forward stroke and lower it on the return stroke. The surface of the table is covered with deflectors extending all the way across from the high side to the low side. These remind one somewhat of the Wilfley table riffles except that they are at right angles to the direc-

[1] H. A. Mcgraw, *Engineering and Mining Journal*, Vol. XCII (1911), p. 387.
[2] H. S. Denny, *Mining Magazine*, Vol. IV (1911), pp. 175, 219; *Mining Magazine*, Vol. V (1911), p. 35; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XI (1911), p. 625; A. W. Allen, *Mining Magazine*, Vol. V (1911), p. 35.
[3] *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 553; B. F. Rice, *Ibid.*, p. 627; C. F Dietz and D. V. Keedy, *Ibid.*, p. 577, *Mines and Methods*, Vol. III (1911), p. 367; B. F. Rice, *Mining Science*, Vol. LXIV (1911), p. 513. C. J. Downey, *Ibid.*, pp. 242, 244, *Mines and Methods*, Vol. III (1911), p. 319.

tion of motion of the table and are of different shape. These deflectors are 6 in. wide and a longitudinal section of the table shows them in the form of saw teeth or steps with a gentle slope of 9 degrees on the side of the tooth toward the mechanism end and a vertical drop on the side toward the discharge end. These deflectors are not arranged in a uniformly parallel manner across the table top, but their side lines correspond to slightly diverging radii extending from an imaginary center above and behind the table top. The deflectors are covered with corrugated rubber with eight corrugations to the inch, each corrugation being 1/8 in. deep. The line of corrugations extends lengthwise of the table top. The mechanism makes 320 r. p. m.

The action of the table is such that the coarsest particles will not progress beyond the first two deflectors before they are discharged over the low side of the table. Finer particles are carried farther and the finest reach the discharge end. Adjustable cutting vanes on the low side and discharge end serve to divide the product into different sizes or grades. The length of the stroke and the inclination of the toggles are adjustable. The capacity is 15 to 20 tons per 24 hours on material crushed to about 20 mesh. One horse-power is required.

SIZING TESTS ON PRODUCTS OF McKESSON TABLE.

Product No.	Screens, Mesh.	Per cent.	Efficiency. Per cent.	Product No.	Screens, Mesh.	Per cent.	Efficiency. Per cent.
1	−24	100.		9	−50	100.	
.....	+30	15.60		.....	+66	6.25	
.....	+40	81.20	81.20	.....	+82	50.00	87.50
.....	−40	3.20		.....	+97	37.50	
2	−28	100.		.....	−97	6.25	
.....	+40	93.75	93.75	10	−66	100.	
.....	−40	6.25		.....	+82	37.50	84.37
3	−30	100.		.....	+97	46.87	
.....	+34	10.92		.....	−97	15.63	
.....	+40	50.00	82.83	11	−66	100.	
.....	+44	32.83		.....	+82	25.00	75.00
.....	−44	6.25		.....	+97	50.00	
4	−34	100.		.....	+116	21.87	
.....	+40	28.13		.....	−116	3.13	
.....	+44	37.50	62.50	12	−66	100.	
.....	+50	25.00		.....	+82	15.63	
.....	−50	9.37		.....	+97	43.75	78.12
5	−40	100.		.....	+116	34.37	
.....	+44	7.82		.....	−116	6.25	
.....	+50	25.00	89.06	13	−66	100.	
.....	+66	64.06		.....	+82	6.25	
.....	−66	3.12		.....	+97	37.50	81.25
6	−44	100.		.....	+116	43.75	
.....	+50	6.25		.....	−116	12.50	
.....	+66	68.75	90.62	14	−82	100.	
.....	+82	21.87		.....	+97	25.00	
.....	−82	3.12		.....	+116	43.75	71.87
7	−50	100.		.....	+139	28.12	
.....	+66	37.50	90.63	.....	−139	3.13	
.....	+82	53.13		15	−82	100.	
.....	−82	9.37		.....	+97	6.25	
8	−50	100.		.....	+116	18.75	
.....	+66	9.37		.....	+139	43.75	68.75
.....	+82	62.50	87.50	.....	+172	25.00	
.....	+97	25.00		.....	−172	6.25	
.....	−97	3.13		Average.....			81.32

The efficiency of the machine is shown in the accompanying table which gives sizing tests on 15 products from ore ground through 24-mesh screen. The table shows that there is some overlapping where sizing tests by diameters are used as a test of efficiency but this is explained by the inventors who claim that the table is a volumetric sizer in which the particles are graded according to bulk and that a flat particle may have the same bulk as a cubical particle but the latter would apparently be smaller when tested by sieves. Specific gravity of minerals is said to play no part in the grading and the machine is not a dry classifier as has been claimed by some. This is shown by the following table of analyses of the sizes in a No. 8 product obtained in treating a mixed ore containing pyrite, blende, and gangue.

ANALYSIS OF SIZES IN NO. 8 PRODUCT.

Size.	Zinc. Per cent.	Iron. Per cent.	Insoluble. Per cent.
On 50 mesh.....	25.10	19.05	2.68
Through 50 on 66 mesh...	32.90	13.30	8.90
Through 66 on 74 mesh..	25.80	7.80	28.64
Through 74 mesh.....	14.80	5.40	48.08
Whole product.....	29.20	12.00	15.56

Laboratory Sizing Tests.[1]—Sizing tests made dry may give misleading results owing to the tendency of very fine material and the slimes to cement together and form lumps. Similarly in wet screening the results may not be satisfactory as shown by the fact that if the oversize be left on the screen and dried, a not inconsiderable number of the dried grains can be made to pass through the sieve. The author's best results have been obtained by first removing all the fine material and slimes in a little laboratory classifier very similar to that shown in Fig. 345, of Richard's "Ore Dressing," Vol. I, page 611. The overflow is removed after about 20 minutes and is dried and weighed as the undersize of 200 mesh. The settled material is dried and sized dry in the usual manner and the undersize of the finest screen may be added to the fines collected in the overflow. This combination process always gives a greater amount of material in the finest size, amounting in some cases to 75 per cent. more than that obtained in ordinary sizing.

Slime Settlement.[2]—At the Sons of Gwalia mine the ore is schistose and makes a large amount of slime. To accelerate its speed of settling in

[1] F. Curtis, *Mines and Minerals*, Vol. XXXII (1911), p. 125; *American Metallurgical Society*, Vol. I (1911), p. 57; *Mining World*, Vol. XXXV (1911), p. 1022.

[2] B. L. Gardiner, *Monthly Journal Chamber of Mines of Western Australia*, May 31, 1911, p. 120; *Journal Chemical, Metallurgical and Mining Society South Africa*, Vol. XII (1911), p. 73; *Mines and Methods*, Vol. II (1911), p. 295; *Engineering and Mining Journal*, Vol. XCII (1911), p. 438, *Mining Magazine*, Vol. V (1911), p. 149; *American Metallurgical Society*, Vol. I, (1911), p. 62; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 467.

cones and spitzkasten they have made use of inclined baffles of corrugated iron which are just submerged below the surface of the water. The principle of the apparatus is the same as that of the De Kalb tank described in Richard's "Ore Dressing," Vol. I, page 459. The details are shown in Fig. 3.

Fig. *A* shows the theory as illustrated in a glass cylinder. When the cylinder is vertical the rate of settling is 40 mm. per minute. When it is

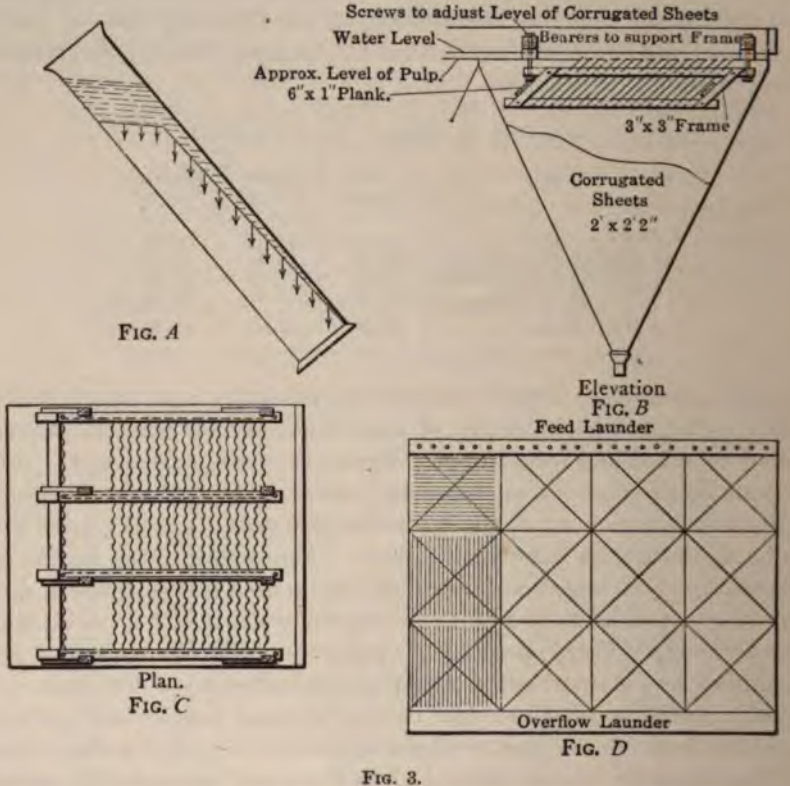

Fig. *A*

Fig. *B*

Plan.
Fig. *C*

Fig. *D*

Fig. 3.

inclined at an angle of 45 degrees the rate is 180 mm. per minute. Figs. *B* and *C* show the details of suspending the baffles; Fig. *D* shows three spitzkasten fitted with the device and nine spitzkasten not fitted. The following table shows comparative results.

The pounds of wet pulp settled per minute is seen to be about proportioned to areas in the two cases. The net result is however to increase the thickness of the pulp from 26 to 41 per cent.

No. of Test.	Spitzkasten 24×8 ft. = 192 sq. ft. with Settling Device.		Spitzkasten 24×24 ft. =576 sq. ft. without Settling Device.	
	Pounds settled per minute.	Percentage of dry slime.	Pounds settled per minute.	Percentage of dry slime.
1	47.3	42.6	168.0	25.0
2	50.4	40.0	139.3	24.4
3	40.6	40.9	144.3	27.2
4	54.4	41.7	144.5	27.2
5	53.0	40.0	182.0	26.6
Average.....	49.1	41.1	155.6	26.1

It has been found best to have the line of the baffles parallel to the direction of flow of the pulp. The baffles extend down about 18 in. and the underflow is so regulated as to maintain 2 or 3 in. of clear water at the surface. The action of this apparatus is not so marked on thin pulp—the thin seeming to mix more easily with water than thick pulp—and the difference in density is not so noticeable.

Hindered Settling Classifier.[1]—Some comparative tests made in the laboratory of the Massachusetts Institute of Technology on free and hindered settling classifiers, both fed with the same mixture of pure galena and quartz, 10 mm. to 0 in size, showed distinctly the superiority of hindered settling. The twelve spigot products, safety spigot, and overflow product of each classifier were each sized on a set of screens. Each product yielded pure quartz in the coarse sizes, mixtures of galena and quartz in the intermediate sizes, and pure galena in the fine sizes.

Comparing the work of the two classifiers it was found that the first spigot product of the hindered settling classifier contained double the galena to that in the first spigot of the free settling classifier. Further the amount of mixed sizes, containing both quartz and galena, was considerably less in the hindered settling products. This was estimated to be somewhere between one-fifth and one-tenth. This reduction in the amount of mixed sizes means that the hindered settling products would be much easier to concentrate on the succeeding jigs and tables.

Richards Janney Classifier.[2]—This apparatus shown in figure 4 consists of a series of hoppers of increasing size and stepped down gradually from the feed end. The lower part of each hopper is lagged to the form of a frustrum of an inverted cone and forms the settling chamber. Below this in succession come the cylindrical sorting column and the hydraulic vortex, with a well-protected glass sight tube between them. The bottom of the hydraulic vortex has a large flanged end tapped to receive the inner valve-seat bushing. Attached to this flanged end is an inverted cone-shaped retarding chamber, the apex of which is

[1] R. H. Richards, *Engineering and Mining Journal*, Vol. XCI (1911), p. 415.
[2] *Engineering and Mining Journal*, Vol. XCI (1911), p. 1198.

tapped to receive the spigot. The retarding chamber is supplied with an air cock. Above each compartment is a cast-iron box with housing for the driving mechanism. A vertical hollow spindle extending downward in each compartment carries a set of sorting blades as shown. A valve rod extending down through the hollow spindle, carries a rubber valve on its lower end which fits into the valve seat bushing previously mentioned. The hollow spindle is given a slow continuous rotary motion by worm gear and the valve rod has an intermittent up-and-down motion.

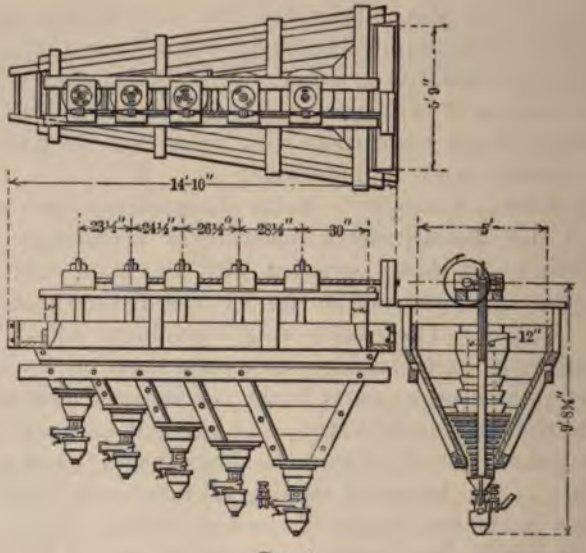

FIG. 4.

CONCENTRATING MACHINES

Ore Sorting at Tonopah.[1]—This operation is carried on at a profit even though the wage scale is $3 to $3.50 per 8-hour shift.

Ore which is to be sorted for shipment is washed before picking, as this is essential for close work. In dry sorting the oversize of 1.5-in. grizzlies passes to a picking table 30 in. high where the pickers remove 10 to 20 per cent. as waste. One man in this case handles 45 to 50 tons of ore per shift and the cost is as low as 25 cents per ton of original ore.

At the Tonopah Belmont ore house the shipping ore goes to Crane washer with 1-in. holes. The undersize is shipped without further treatment and the oversize goes to a 40-in. picking belt. Warm water at a

[1] C. T. Rice, *Engineering and Mining Journal*, Vol. XCII (1911), pp. 761, 836. *Ibid.*, Vol. XCI (1911), p. 1048.

temperature of 90° F., is used. There are three sorters per shift and they sort about 35 tons per shift removing about 25 per cent. as waste.

At the West End the mine ore is dumped into a 240-ton bin which delivers to a pan conveyor feeding a 10×20-in. Blake breaker. The breaker product, 2 3/4 in. maximum size is elevated to 1-in. trommel. The undersize is conveyed to the shipping bins. The oversize goes to Crane washer 46 in. diameter, 12 ft. long and revolving on an axis sloping at about 20 degrees. Fines escape through 3/8-in. holes, and are settled, dried and shipped to smelter. The washed residue has the waste picked out on a picking belt 40 in. wide and 25 ft. long. The plant treats about 85 tons per day working one shift. The crew is one man at the breaker and six pickers all receiving $4 for 8-hour shifts. About 85 per cent. of the original feed to the trommel goes to the pickers and they remove about 32 per cent. of the original feed as waste. The waste assays $2.50 per ton. The entire cost is 43 cents per ton of original feed.

Concrete Jigs at Cananea, Mexico.[1]—Wooden jigs of the Harz type with three compartments made of 3-in. Douglas fir planks with 8×8-in. posts are badly attacked by dry rot after 8 years use and they are being replaced by concrete jigs. The walls and partitions are 3 in. thick. The walls and legs are reinforced with iron screen with 1 1/2-in. holes and 3/16-in. wire. The partitions and legs are also reinforced with a few 3/4-in. bent steel rods. The concrete is poured sufficiently liquid to fill all small crevices and openings and the forms are constantly rapped during pouring to insure proper packing and the expulsion of air. The concrete sets in 36 hours. When the form is removed, the inside of the jig is painted with several coats of a water-proof coating made up of 1 lb. concentrated lye, 5 lb. alum, and 2 gallons of water. To four pints of this mixture, 10 lb. of cement are added and thoroughly stirred. The shafting, brackets, etc. are taken from the old wooden jigs. The brackets rest on the concrete wall of the jig and are anchored with 3/4-in. bolts imbedded in concrete.

The cost of the two first jigs set in place including the form but not the brackets, shafting, etc., was as follows: Cement, $51.20; nuts, bolts, washers and nails, $5.32; reinforcing, $27.84; castings, $19.60; 16-lb. rail, $2.16; total supplies, $106.12; carpenter foremen, $78; carpenters, $185; other labor $59; total for the two jigs $428.12. Since then using the same forms a single three-compartment jig has been built at a cost of $145.05. The original article gives full illustrations of the jigs and forms.

Bar Screens for Jigs.[2]—At Lipine in Upper Silesia investigations have been made on various forms of sieves for jigs. The common practice is to use round punched plate for coarse jigs and wire screens with 1.5 to

[1] F. J. Strachan, *Engineering and Mining Journal*, Vol. XCII (1911), p. 698.
[2] Meyer, *Gluckauf*, Vol. XLVII (1911), p. 1401.

2-mm. holes for the slime jigs. Both of these forms of screens have the following disadvantages: They are not strong and frequently break and have to be replaced causing loss of time and imperfect work of the jig before the break is discovered. The use of stronger material is of no assistance because such material is more brittle. Punched sieves have only from 29 to 40 per cent. of opening and the dead spaces both interfere with the rising water currents through the sieves and affect the jigging action by causing dead sections in the jig bed above the dead spaces of the screen coupled with boiling craters above the screen openings. This effect is accentuated in the case of fine wire sieves which are supported on a coarser wire screen. The sieves blind badly and prevent free passage of hutch material so that they have to be cleaned frequently. The use of thicker plates or heavier wire increases the life of the screen but at the same time increases the tendency to blind. Sieves with square punched holes are not so strong as round punched sieves. Bar sieves were tried which had tapered bars resting on cross supports. They gave larger percentage of opening but the bars spread and the supports broke. Also the edges of the tapered bars wore rapidly thus enlarging the opening. Altogether this form of sieve was found to be unsatisfactory.

The best form of jig sieve was a cast grating with bars which were square above and tapered below, cast in a solid rectangular frame. Two solid cross pieces also serve to reinforce the bars and maintain even spaces between the bars. The tops of the bars may be chilled to prevent wear and prolong the life of the grating. This type of sieve is known as the Schubert patent and it is made successfully with openings as fine as 2 mm. Although it has a higher initial cost this is more than offest by the advantages of greater percentage of opening (50 to 88 per cent.), less running cost, less power, less blinding, much less labor for cleaning, better jigging and increased capacity. Moreover those long-lived gratings allow the use of a jig bed of iron shot which is far more durable than a mineral bed of galena, pyrite, blende, etc., as the case might be. The old style sieves were not strong enough to allow the use of a shot bed.

Patching Vanner Belts.[1]—A common method of failure of vanner belts is by cracking at the edge which crack finally extends into the bottom of the belt and causes leaks. The belt is generally patched by cementing or sewing in an old piece of belt, but the sewed patch results in leaks, and it is hard to make a good patch by cementing. The Tonopah Mining Company has therefore applied vulcanizing. The damaged portion of the belt is cut away and a piece of old belt inserted in the opening. A layer of crude rubber 1/16 in. thick and large enough to extend an inch outside the patch all around is put on as a cover. The vulcanizing is

[1] C. T. Rice, *Engineering and Mining Journal*, Vol. XCII (1911), p. 8.

effected in 45 minutes using hollow brass forms or clamps heated with steam at 60 lb. pressure per square inch. The vulcanized surface is allowed to cool slowly with the forms still clamped in place. Where the patch is large the vulcanizing is performed in sections.

Protecting Riffles on Wilfley Tables.[1]—On the coarse tables in the Steptoe mill at Ely, Nevada, the wooden riffles wear rapidly. Their life is quadrupled by nailing protecting strips of sheet copper on top of the riffles. The copper is 0.025 in. thick and costs $2.05 per table while the wooden strips to cover a table cost $7.

James Diagonal Plane Concentrating Table.[2]—The inventor of this table claims that riffles have been greatly overrated as savers of mineral on jerking tables. The mechanical action of this table resembles somewhat that of the Wilfley table but there are important differences in the table top, which may be considered in two sections: the upper smooth section near the feed launder and the riffled section which includes the remainder of the table. The smooth section is made in the form of a very shallow basin in which the minerals of greatest specific gravities are settled and stratified previous to their discharge upon the riffled portion of the deck.

The riffled surface may be further divided into two sections, one for the reception of concentrates and the other for the tailings. The riffles on the concentrates section are very thin, being but 1/32 in. high. As the concentrates are discharged from the settling basin on this portion of the deck, the low riffles allow them to spread out, which action enables such gangue as may remain in them to become free, forming a thin upper stratum, which is easily washed away by the dressing water. Since the line of table motion is diagonal to the line of settlement there is a tendency for the concentrates to be driven not only forward but upward against the inclination of the deck, thus counteracting the tendency of the dressing water to wash the concentrates into the riffled section for tailings. The tailings in the settling basin do not come in contact with the riffles until they have been washed entirely free of values. The tailings riffles act simply as a retarding influence preventing too rapid discharge of the tailings and wash water from the non-riffled section. They have nothing to do with the stratification of minerals. The table top is not of the same shape as the Wilfley but is considerably extended at the discharge side resembling somewhat the Overstrom table in shape.

Dallemagne Table for Slimes.[3]—This is a table of the Wilfley type

[1] *Engineering and Mining Journal*, Vol. XCII (1911), p. 884.

[2] S. A. Krom, *Bulletin American Institute Mining Engineers*, No. 55, July (1911), p. 569; *Mines and Minerals*, Vol. XXXII (1911), p. 44; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 468; *Mining Science*, Vol. LXIV (1911), p. 86; *Engineering and Mining Journal*, Vol. XCII (1911), p. 11.

[3] Ratel, *Comptes Rendus Mensuels de la Société de l'Industrie Minérale*, 1911, p. 416; *Mining Journal*, Vol. XCIII (1911), p. 552; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XII (1911), p. 34.

having the riffled top, end motion and side slope but differing in the mechanism. There are two or more horizontal cross shafts mounted in boxes on the fixed frame beneath the table which receive a movement of oscillation (not complete rotation) the amount of which may be varied. On each shaft are two eccentrics and the table is mounted on the collars of these eccentrics. Connecting rods join the eccentrics so that they work in harmony and the table thus receives a combined vertical and horizontal movement the relative components of which depend upon the setting of the eccentrics. The speed is high, 600 to 1500 complete strokes per minute, and thus the table descends faster than the ore grains lying on its surface so that these grains are progressed along the table in much the same manner as ore passes over the Ferraris screen.

The table is designed especially for slimes and has done good work on material finer than 240 mesh or 0.07 mm. It can work up to 3 mm. The table is 5 meters long, 1.9 meters wide and treats from 150 to 800 kilos per hour consuming 0.75 horse-power. The water consumption is 17 to 35 liters per minute. Results of tests are given which indicate that the table makes clean products.

Monell Slime Table.[1]—This table belongs to the end-shake, side-slope, endless-belt type. It has a 4-ply rubber belt which runs level longitudinally except for the last few inches at the concentrates end. The ore is fed at the side on the first 12 ft. of the table, the entire table top being 23 ft. long. The width of the table is 5 ft. of which 3 ft. on the feed side is level; the 2 ft. on the opposite side have a downward slant. The motion is longitudinal, at the rate of about 180 vibrations per minute which gives a panning motion to the pulp. The concentrates are taken off over the head of the table while the tailings are washed over the edge. The capacity is claimed to be 10 to 12 tons per 24 hours, with 1 to 1 1/4 horse-power used. The inclination of the tailings side of the belt can be varied at will. The use of the Monell table is said to have raised the saving of tungsten, in the Boulder, Colorado, field by 15 per cent.

Porter Canvas Slime Table.[2]—Among the objections to the usual canvas table plant are the large area occupied, necessitating a large building, and the hand labor required for washing. The Porter slime table aims to overcome these by mounting the tables one above the other and by using mechanical washing jets operating automatically for the purpose of removing the concentrates. There are 12 tables each 12×1 ft. mounted one above the other with 4-in. vertical space between them. The feed pulp is distributed to the 12 tables. A set of traveling jets for each table travels the length of the table in 18 seconds. The four

[1] *Engineering and Mining Journal*, Vol. XCI (1911), p. 1007.
[2] E. C. Porter, *Mining Science*, Vol. LXIII (1911), p. 477; *Mining World*, Vol. XXXIV (911), p. 453.

upper tables are washed as a unit, then the four middle tables and finally the four lower tables.

Magnetic and Electrostatic Concentration

Handling Ore with Electromagnets.[1]—At the Moose Mountain magnetite mine the blasted ore is loaded into cars by a lifting magnet. This has replaced a steam shovel which was worn rapidly by the hard magnetite. An additional advantage is that the magnet automatically discards the non-magnetic waste.

Ullrich Magnetic Separator.[2]—This consists of horizontal revolving iron disc mounted on a vertical shaft. This disc forms one pole of an electromagnet and it has on its underside near the circumference four downward projections extending all the way around the circle to as to form complete rings, the outer ring projecting the least. Set beneath these rings are the radial branches of the opposite magnetic pole which extend out horizontally and are fixed. These branches are so arranged that any point on the rings is first above a branch of the lower pole then over a hopper set between the branches, then over another branch and so on for six or eight times per revolution. Non-magnetic inclined chutes bring the ore, which may be either dry or wet, radially inward over each branch of the lower pole. The non-magnetic particles continue inward and fall from the end of the chutes into a central hopper while the magnetic particles are lifted to the rings and by the revolution of the upper pole are carried into the spaces between the branches where the magnetic force is weak and where they drop off into the concentrates hoppers below.

These machines are working on iron ore at Dunderland, Norway, where on a finely divided hematite and magnetite ore containing 27.6 per cent. iron and 0.164 per cent. phosphorus, the concentrates obtained contained 64.0 per cent. iron and 0.0132 per cent. phosphorus and the extraction of iron was 76.4 per cent.

Murex Magnetic Process.[3]—The Murex plant installed at the Broken Hill Block 14 mine is reported to have made an extraction of 80 per cent. of the metallic content of the ore. The problem now to be solved is whether the de-oiling and de-magnetizing of the pulp can be done cheaply and efficiently.

Wilfley Roaster for Magnetic Work.[4]—The furnace consists of an annular rotating roasting hearth with an auxiliary fire-box, feeding device

[1] *Iron Age*, Vol. LXXXVII (1911), p. 431.
[2] *Stahl und Eisen*, Vol. XXXI (1911), p. 1127.
[3] *Mining and Scientific Press*, Vol. CIII (1911), p. 48.
[4] H. Eggers, *Mining Science*, Vol. LXIII (1911), p. 110; *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 171; *Engineering and Mining Journal*, Vol. XCII (1911), p. 484.

and device for removing the ore from the hearth when roasted. A roller feeder puts the ore on the hearth in a very shallow bed, approximately 1 grain deep. The roasting of pyrite to the magnetic monosulphide is accomplished in from 1 to 3 minutes. The hearth rotates in the opposite direction to the path of the gases so that its temperature is gradually raised while it makes its revolution. The device for removing the ore from the hearth is a specially constructed suction nozzle connected with a suitable receptacle. The speed of the hearth may be adjusted to give the proper time of roast. On an experimental machine 7.5 sq. ft. of hearth area will roast 1 ton in 24 hours.

Huff Electrostatic Plant at Cananea, Mexico.[1]—The ore of the Calumet and Sonora mines contains galena and blende with lesser amounts of pyrite and chalcopyrite in a gangue of calcite, quartz and barite. The wet mill makes a 60 per cent. lead concentrate, middlings with 40 per cent. zinc, 5 per cent. copper, 10.5 per cent. iron and 1.5 per cent lead, and tailings. The middlings are dried and elevated to a 2-mm. trommel. The oversize is crushed in rolls to 2 mm. The material is then sized, 2 mm. to 20 mesh, 20 to 40 mesh, and through 40 mesh. The first two sizes are treated on Huff roll-type separators and the third or finest on Huff toboggan-type separators. The plant treats 40 tons per day and yields a zinc product containing 55 per cent. zinc, 1 per cent. copper, and 4 per cent. iron with a zinc recovery of 85 per cent. The tailings from the Huff separators are treated on tables and yield a copper product containing 9 per cent. zinc, 16 per cent. copper, and 10 oz. of silver per ton. A 20 horse-power electric motor drives the plant and three men are required for shift.

Half of the time the electrostatic machines are run on material from the stock pile. At first this did not give a good separation owing to the fact that some of the copper had been leached and redeposited as a minute film on the blende and thus gave the blende an electric characteristic like the pyrite and chalcopyrite. This difficulty was overcome by subjecting the material from the stock pile to a preliminary treatment with weak cyanide solution which dissolved the copper. This property of blende to precipitate copper has been applied in separating it from a heavy gangue such as barite, rhodonite, etc. Here the material is given a bath in copper sulphate and the blende particles become coated with copper in which condition they may be repelled from the heavy gangue when submitted to the electrostatic treatment.

[1] F. S. MacGregor, *Engineering and Mining Journal*, Vol. XCII (1911), p. 1080; *Mining World*, Vol. XXXV (1911), p. 389; *Mining Magazine*, Vol. V. (1911), p. 308; J. M. Houser, *Mining and Scientific Press*, Vol. CII (1911), p. 760.

Flotation Processes

Experiments on Mineral Flotation.[1]—An exhaustive set of laboratory tests has been made to investigate the buoyancy of minerals under all sorts of conditions, viz., water cold and hot, at normal and at reduced atmospheric pressure, free from air and saturated with gases; acids, strong and dilute; oils, alone and in combination with water; alkaline solutions; minerals bright and tarnished.

Cold Water.—The following minerals will float when dropped carefully on the surface of the water: Galena, blende, sulphur, azurite, malachite, rhodonite, garnet, calcite, mica, telluride of gold, pyrite, pyrrhotite, wolframite, and cassiterite. Some samples of quartz will float and some will not. All metals in the form of foil will easily float. Zinc-blende was heaped up to a depth of 1/4 in. until it amounted to 15 grm. in a 50-c.c. beaker. As under ordinary atmospheric conditions water has the greatest surface tension of all liquids except mercury, the phenomenon is explained on this basis especially as the addition of alcohol or caustic soda, which tend to reduce the surface tension, causes the material to sink. In the case of pulverized particles of almost all minerals a flotation may be produced by covering them with water in a vessel, uncovering them again and then finally bringing the water carefully back. This form of flotation is due to air films on the under side of the particles.

Hot Water.—Most minerals will sink within a few seconds. Usually a bubble of gas leaves each particle as it sinks.

Under Water.—Some minerals submerged in cold water will float when the water is heated. Such are sulphur, crystalline azurite and malachite, some galenas, some blendes, some chalcopyrites and stibnites and some of the metals, copper being the most persistent. Bubbles of air and water vapor appear to be the floating agencies here.

Water under Reduced Pressure.—This will cause many substances to rise which have previously sunk under normal pressure. Sometimes after rising they will sink again and repeat this three or four times. The reduction of pressure causes bubbles to form on the sunken particles perhaps from air included in the particles, but these bubbles will not cause all minerals to float, for example, quartz, garnet, gypsum, hematite, wolframite, cassiterite, and glass.

Cold Dilute Acid.—The results were similar to cold water even with strong sulphuric acid.

Hot Acid Solutions.—When sprinkled on the surface there is very little flotation. When heated in a weak sulphuric acid solution, most sulphides

[1] Proceedings Royal Society of Victoria, Vol. XXIII, Part 2, March, 1911; *Australian Mining Standard*, Apr. 5, 12, 20, 27, (1911), *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 426; *Engineering and Mining Journal*, Vol. XCII (1911), p. 307; *Journal Chemical, Metallurgical and Mining Society*, South Africa, Vol. XII (1911), p. 136.

will float although some galenas and blendes require the addition of calcite to generate carbon dioxide before they will float. Hydrogen sulphide is given off in every case. Hematite, biotite, quartz, wolframite, cassiterite, rhodonite, calcite, siderite, muscovite, serpentine will not float.

Dilute Acids under Reduced Pressure.—Bubbles are formed but they appear to be larger than when acid is absent and they come away from the minerals more readily. Most of the sulphides even, will not float in the cold in dilute acid.

Oils.—None of the minerals will float on the surface of oleic acid, kerosene or alcohol.

Oiled Minerals in Water and Acid Solutions at Normal Pressure.—Nearly all minerals will absorb oil if shaken in water to which oil has been added or if previously oiled with oleic acid, vaseline or kerosene, they will float in water. Some of these float in the cold; others require heating or reduced pressure. The following float well: Calcite, sulphur, galena, garnet, blende, wolframite, cassiterite, siderite, cerrusite, magnesite, azurite, malachite, rhodonite, some quartz, and the metals in the form of foil. The following give no float or a very poor one: muscovite, amorphous malachite, turquoise, gypsum, quartz, and particles of slate. On adding acid to the minerals which have absorbed oils in the presence of water, most minerals, other than sulphides and metals, will not float when heated, the acid apparently causing them to separate from the oil.

Reduced Pressure on Oiled Minerals in Water and Acid Solutions.—In water large bubbles formed as in the case of unoiled minerals but larger and in greater number and the effect of the oil is to make the bubbles more permanent and more tenacious. Addition of acid followed by standing or heating causes many bubbles and apparently the excess of oil to leave the minerals, and in the case of quartz, calcite, garnet and gypsum, the bubbles practically all leave, while sulphides, metals, and garnet still have bubbles on them.

Cold Alkaline Solutions.—Most minerals will float on caustic soda solution, but for a few moments only.

Hot Alkaline Solution.—The flotation effect is either lacking or nonselective unless the solution is acidified.

Saturated Solutions of Gases.—Bubbles form on all minerals and tend to lift them in water containing carbon dioxide, sulphur dioxide, hydrogen peroxide, or air, but when acid is added the bubbles have a tendency to select the sulphides on which to form.

Film Flotation.—Here as in the DeBavay process the sulphides float on the surface as a film one particle deep, each particle being practically submerged, instead of as a scum as in the Potter and other processes.

Tarnishing.—If the surfaces of sulphides are tarnished, as a rule these sulphides will not float.

Summary

All minerals adsorb gases. Sulphides appear to adsorb carbon dioxide naturally. The consequence of this is: (1) Particles are not wholly wetted when immersed in water; (2) particles also tend to float when sprinkled on the surface; (3) when immersed in water and heated, the air or gases dissolved in water will collect on the particles, and float or tend to float them to the surface; (4) saturated solutions of gases in water evolve gas which collects on all particles.

The Addition of Acids.—Metallic particles such as sulphides and metals, when immersed in dilute acid solution, are not wholly wetted, but particles of rock material become more readily wetted and give off their adsorbed gas. When different dry minerals are sprinkled on the surface of dilute cold acid, the tendency for them all is to float as in water. The minerals are more readily wetted by hot acid solutions than cold.

The Addition of Alkalies.—When different mineral particles are immersed in an alkaline solution, they tend to part with their adsorbed gases and will not float even when heated. When sprinkled dry on the surface of the solution they remain for a few seconds only and then break through.

Flotation of Sulphides.—Given a starting-point any gas generated in or introduced into a solution will tend to collect on the surface of particles. If acid is added the gases will leave the gangue particles and collect on the sulphides; the latter, therefore, tend to float and the former to remain under the surface of the liquid. When substances in the solution, such as carbonates, are attacked by acids, the gases evolved saturate the liquid and commence to grow on the sulphides. Sulphuretted hydrogen, steam, air and any other gases either present in the solution, introduced into it or evolved from the minerals present, assist flotation, provided the sulphide particles have dry points on them for the gases to become attached to. Some sulphides naturally adsorb gases and such sulphides readily float. The bulk of Broken Hill ores may be taken as examples. Other ores require preliminary treatment, so as to alter their surface to some extent. This may be done in certain cases by merely heating to 250° C., or immersing in some solution which will attack the sulphides, such as nitric acid, or chlorine solution. Sulphides which condense or adsorb gases well will gather together in a solution as flocculent masses, while those that do not, remain individually distinct like grains of sand.

In certain cases sulphides will float in acid solution at ordinary temperature, but for the most part, with dilute acid, flotation starts at about 80° C. The explanation put forward that flotation at that temperature is due to the fact that certain carbonates, such as rhodochrosite, siderite, smithsonite, are attacked and give off carbon dioxide, does not appear to be correct, for under reduced pressure these minerals are attacked but flotation does not take place. The presence of silicic acid and sulphur, as stated by DeBavay as necessary for a coagulum, is not necessary in many cases of successful flotation. When minerals are treated with dilute acids, and then exposed to the air, certain sulphides adsorb gases on their surface. If they are alternately wetted and exposed to the air, they will become entrapped by the surface film of water with which they are brought in contact. The appearance of the film is the same as that of a bubble of gas under the surface of a solution with sulphides attached. The sulphides in both cases adhere to the surface film of the bubble or free surface of the liquid, but are mainly below the surface itself. The gas obtained in all cases from experimental work, and also from samples obtained from various flotation plants, was carbon dioxide with varying quantities of nitrogen and oxygen. There is no doubt that most of this gas is generated from gangue particles in the ore, and being more soluble in the liquid than the other gases collects readily on the surface of the sulphides.

In all experiments with sulphides, carbon dioxide was obtained on exhaustion. This gas must have been present either in the form of a compound easily decomposed by heating or by reducing the pressure, or as a gas condensed on the surface of the sulphides and there retained with greater force than other gases present. It was found that the carbon dioxide was the last gas to come off.

Effect of Oils.—The effect of oils has long been known in protecting minerals from being wetted by water, owing to the oiling of their surfaces, and also the affinity of oils for metals and many metallic substances. Nearly all the minerals tried could be coated with oil either in the presence or absence of water. This seems to be connected with the adsorption of gases on the particles themselves, for if the particles are deprived of their gas, then, as a rule, they do not become coated with oil. It would, therefore, appear that the oiling of particles is dependent on their gaseous attachments. If the particles are oiled to a minute extent the gases previously adsorbed are retained, and if a solution is heated more gas will become attached to the particles and flotation will ensue. An oil float may be obtained without acids. Nearly all minerals will float after being oiled.

The flotation is apparently due partly to the lesser density of the oil

and the presence of gases adsorbed on the minerals or entangled in the oil. If a large quantity of viscous oil is used, gases become entangled in it, and the product is oily. In course of time the particles will drop away from it, carrying down attached globules of oil.

When a lesser amount of oil is used the product becomes a coherent mass like putty, showing no apparent tendency to float. On further decreasing the quantity of oil, the product becomes firmer and breaks into rounded pellets. With a still less quantity of oil, when shaken up with water in the presence of air the product becomes flocculent and increases in volume. This flocculation is due to the adsorption of gases. With very minute quantities of oil the flocculation becomes more pronounced, and the tendency to float becomes greater. These results may be obtained with gangue and metallic substances. Gases collect more readily on oiled than on unoiled particles.

Effect of Acids on Oiled Particles.—The addition of acid causes a more selective action. If oiled gangue and metallic particles are immersed in water, and acid is added, then the oil will tend to leave the gangue particles and become attached to the sulphides. The action of the oil in this respect is similar to that of the gases attached to the different minerals, and probably is due to the same cause.

If an ore is treated with an acid and the acid is removed, and if the particles are then oiled, the oil tends to become attached to the sulphides only. This is probably due to the removal of the gas from the gangue, which prevents oiling. If acid and oil are added together, the action of the oil is selective as in the previous case. The presence of oil on their surface does not prevent minerals from being attacked by acids.

The amount of acid required for selective flotation depends on the composition of the ore, the temperature and the pressure existing at the time. Generally, it may be stated that an increase of acid will give a purer sulphide product, but flotation will take place at a lower temperature, and with a lesser amount of acid, with oil and acid than with acid alone.

A selective flotation can be obtained in some cases by treatment with a chlorine solution previous to the addition of oil and without the addition of acid.

Some of the light volatile oils will cause flotation in an acid solution at a lower temperature than the heavier and thicker oils. With some Broken Hill tailings a flotation takes place at ordinary temperature with sulphuric acid and a volatile oil.

Effect of Oils and Alkalies.—Flotation can be obtained in an alkaline solution when oil is used but in every case tried the action was non-

selective. On making the solution acid the action becomes selective, only the sulphides floating.

The flotation of mineral particles appears then to depend on the fact that the particles having gas attached to them become entangled or caught by the surface film of a liquid. This film may be (1) the free surface of a liquid, (2) the surface of a bubble of gas under the surface of a liquid, or (3) the surface of a bubble of gas above the surface of a liquid. Once the particles are entangled in this film they are sustained there. That a considerable weight may be supported on this surface film is shown by the experiment of floating fine-grained mineral on the surface film of the water.

Eucalyptus Oil for Mineral Flotation.[1]—Experiments at the Central mine at Broken Hill have shown that eucalyptus oil gave the best results, only 6 cents worth per ton being required to give the desired separation of sulphides from gangue.

Flotation Patents.[2]—In connection with the mass of litigation over flotation patents it is interesting to note that a patent covering the use of a fat or an oil or a constituent thereof, an acid or soluble neutral or acid salt, and water, in ore separation, was given to Miss Carrie J. Everson in U. S. Patent 348157, Aug. 24, 1886. This patent long ago expired in all the countries where it was taken out.

Zinc Corporation and Flotation Processes.[3]—Since changing from the Elmore to the Minerals Separation process the Zinc Corporation at Broken Hill has reduced the tenor of the tailings to 2 per cent. zinc. Formerly they contained 4 per cent. zinc. The working costs will probably be lessened by between 24 and 36 cents per ton. The credit of this is not due entirely to the change in process but partly to mechanical improvement by the Zinc Corporation staff.

Elmore Process at Butte.—A new installation is reported to be treating successfully the zinc ores of the Butte and Superior Company at Butte. No details of the plant or results have yet appeared in the technical press. To the best of the writer's knowledge this marks the advent of the Elmore vacuum process in America although the old Elmore process was used some years ago at the Le Roi mill in British Columbia.

Macquisten Tubes in Idaho.[4]—An installation of 120 tubes has been made at the Morning mill at Mullan, Idaho, to treat 40-mesh Wilfley table middlings containing about 10 per cent. zinc, 35 per cent. siderite, and some barite. The middlings are first dewatered to reduce the moisture content to 10 or 15 per cent. and before passing to the distributors

[1] *Mining and Scientific Press*, Vol. CIII (1911), p. 556.
[2] *Engineering and Mining Journal*, Vol. XCI (1911), p. 745.
[3] *Engineering and Mining Journal*, Vol. XCI (1911), p. 1198.
[4] *Metallurgical and Chemical Engineering*, Vol. IX (1911), pp. 236, 567; *Mining and Scientific Press*, Vol. CII, (1911), p. 693; W. A. Scott, *Ibid.*, Vol. CIII (1911), p. 809.

they are passed through an acid bath containing 5 per cent. sulphuric acid. In addition to this the water in the circuit contains 0.5 per cent. sulphuric acid. The purpose of the acid is to cleanse the blende particles to expose their surfaces so as to better induce flotation. As originally installed, the pulp was divided so that each portion passed through four tubes in series, each tube removing some concentrates from the pulp passing through. The tailings from the last tube of the series were pumped to Callow tanks and the settled slime was put through other tubes for the recovery of the finer particles of mineral. During October, 1911, the capacity of the plant was 61 tons per day and the concentrates amounted to 200 tons carrying 48 to 49 per cent. zinc. It was demonstrated that by keeping the flow surface in the tubes crowded or fully covered, the grade of concentrate produced was higher than when running with lighter feed. The original practice has been changed so that all the tubes receive direct feed instead of running four tubes in series. This has increased the capacity to 95 tons per day giving a concentrate product of close to 300 tons per month that is estimated to contain 49 per cent. zinc. The results are so satisfactory that another unit is planned. The cost of acid does not exceed 2 cents per ton treated. The Macquisten tubes were fully described in Richard's "Ore Dressing," Vol. III, p. 1563.

Accesory Apparatus

Automatic Pulp Distributor.[1]—At the Sons of Gwalia mill the distributor for 10 grinding pans consists of a fixed cylindrical receiver divided by radial webs into 10 sections. Above this receiver is a feed hopper in the form of a vertical cylinder revolving on ball bearings. Pulp fed into this cylinder flows out through four curved discharge pipes which discharge consecutively into the ten compartments of the receiver beneath. The rotation of the cylinder is brought about by the reaction effect of the four discharge pipes. In the event of one grinding pan being temporarily out of commission a hinged launder deflects the material from that compartment of the receiver back to the tailings wheel which elevates the pulp to the distributor. If the outlet to the compartment were simply blocked those compartments on either side would receive more than their fair share of the pulp.

Capacities of Sand Launders.[2]—An extended investigation has been made on the effects of size, shape and material of launders, and also on the effects of the size and shape of the particles carried therein. From

[1] W. R. Degenhardt, *Journal Chemical Metallurgical and Mining Society, South Africa*, Vol. XI (1911, p. 575; From *Journal of Chamber of Mines of Western Australia*, Jan. 31, 1911, p. 42.
[2] G. C. Bring, *Jern Kontorets Annaler* (1911), p. 702.

the results it is possible to design the best launder under the conditions existing in any special case.

Prevention of Foaming in Launders.[1]—A small wooden paddle wheel is placed across the slime launder at the Last Chance cyanide plant at Mogollon, New Mexico. This wheel is revolved by the force of the slime stream, the paddles serving to knock down the foam.

Air Lift for Raising Sand.[2]—For raising and transporting sand from cyanide vats at Burbank's Main Lode in West Australia an air lift of 23 ft. 3 in. in two stages is used. The two lifts use 47 cu. ft. of free air per minute under a pressure of about 70 lb., discharge 320 lb. of sand per minute and require 800 lb. of water per minute. A vat holds 30 long tons and with one man shoveling to the discharge opening in the bottom which leads by launder to the air lift, the tank is emptied in 3.5 hours. The vertical discharge pipes of the lifts are 3 in. diameter and the air pipe is 3/4 in. diameter. The nozzle ends of the air pipes are closed and twelve hack saw slits are cut across each of the ends, the cuts extending around about one-quarter of the circumference of the 3/4-in. pipe and cut at an angle of 45 degrees thus giving the air an upward direction. The consumption of air may be taken as about 50 per cent. more than the best pump would use when in perfect condition.

Principles, Theory and General Ideas.

Value of Coarse Concentration or Ore Sorting[3]—Many mills put through considerable waste rock which can be removed by hand picking. The removal of waste not only raises the value of the ore going to the mill but also effects a saving in the mill in time, labor and work that would otherwise be expended on the waste. A further saving is made because this crushed waste is not present in the tailings and thus operating to carry off with it some of the fines or slimes produced in milling.

In one special case an old dump which averaged too low to be of value was raised to commercial grade by treatment in a Crane washer followed by picking tables. The crew consisted of one washer man and three pickers per shift at a total cost of $12 per 8 hours. Sometimes one or two extra pickers were added. Details of costs and results are given in the table on page 839.

In another case an ore containing auriferous quartz and barren clay would barely pay for milling. The clay coated the amalgamated

[1] V. H. Wilhelm, *Engineering and Mining Journal*, Vol. XCI (1911), p. 1095.
[2] A. R. Mill, *Monthly Journal Chamber of Mines of W. Australia*, Nov., 1910, p. 367; *Journal Chemical, Metallurgical and Mining Society, South Africa*, Vol. XI (1911), p. 398; *Engineering and Mining Journal*, Vol. XCI (1911), p. 706.
[3] H. M. Adkinson and A. L. Stewart, *Engineering and Mining Journal*, Vol. XCII (1911), pp. 145, 161, 243.

plates and caused high tailings losses. By installing a log washer to remove the clay the mill feed was raised to $6 per ton where before it has assayed only $4 per ton. A better recovery was obtained and the profit increased.

Comparative Efficiencies of Screening and Classifying Fine Material and Comparison of Screened with Classified Feed on Wilfley Table.[1]—The development of the Wilfley table and Callow traveling belt screen in recent years has produced a fruitful field for investigation. An investigation has been made at McGill University, Montreal, the primary object of which was to determine the relative merits of screened and classified feed for the Wilfley table. Incidentally the efficiencies of Callow screens and hydraulic classifiers were studied. The ore used in the greater part of the work was a low-grade chalcopyrite ore from the Bruce Mines, Ontario. The Callow screen used had a belt 1 ft. wide, and thus had one-fourth the capacity of the standard screen which has two belts each 2 ft. wide. The classifier used was a specially designed Bell laboratory classifier. The Wilfley table was a standard laboratory table 7 ft. long. All of the ore was prepared in either stamps or Huntington mill so that it would pass an 18-mesh screen.

The Callow screen tests showed an efficiency of from 40 to 75 per cent. The efficiency is measured as the quantity of undersized material passed through the screen, expressed as a percentage of the amount of undersized material in the screen feed. The rate

[1] J. R. Cox, G. G. Gibbons, and J. B. Porter, *Bulletin Canadian Mining Institute*, No. 16, October (1911), p. 79.

	1908.			1909.			1910.			
	July.	August.	September.	July.	August.	September.	June.	July.	August.	September.
Tons material into washer	2665	2666	2448	3837	4323	3660	4122	4822	5014	4240
Waste sorted out	818	962	632	1458	1517	1200	2279	2358	2422	2036
Tons ore after sorting	1847	1704	1816	2379	2806	2460	1843	2264	2592	2204
Per cent. waste rejected	30.7	36.1	25.8	37.8	35.1	32.8	55.2	53.0	48.3	48.0
Cost of sorting per ton of material—cents	11.9	12.9	14.3	20.7	19.5	21.0	19.3	17.7	17.3	18.7
Cost of sorting per ton produced—cents	17.2	20.2	19.3	33.4	30.1	31.3	43.1	37.7	33.5	36.0
Tons handled per man in 8 hours[1]	26.1	24.1	21.0	16.7	17.7	16.5	15.5	17.0	17.3	16.0
Tons handled per picker in 8 hours[2]	34.8	32.2	28.0	22.4	23.7	22.0	20.1	21.3	22.0	20.5
Tonnage through washer in 8 hours	104.5	96.6	84.1	66.1	72.0	65.3	72.3	83.1	80.8	73.1
Average value of picked ore	$6.26	$7.56	$7.34	$8.02	$8.87	$7.68	$8.36	$7.05	$7.39	$7.78
Original value of material	$4.35	$4.83	$5.45	$4.97	$5.76	$5.16	$3.73	$3.31	$3.82	$4.04
Percentage of increase	43.9	56.5	34.7	61.3	54.0	48.8	124.1	113.0	93.4	92.5

[1] Based on total crew. [2] Based on pickers only.

of feed was about one-third the proportional rated capacity given by the makers of the screen. The efficiency is affected by the rate of feed, by the percentage of undersize in the feed and by the amount of water used. The classifier tests yielded products which showed some overlapping when sizing tests were made of them. There appears to be no simple method of calculating the efficiency of the classifier work but on the whole it appears to be better than that of the Callow screen. Previous work by Professor Richards[1] on mixtures of pure minerals led him to conclude that both sized and classified feeds for the Wilfley table were better than natural feed and further that where good classification is obtainable the classified feed is better than the sized feed for mill work. These conclusions were reached after a full consideration of the quality and amount of the Wilfley table concentrates, middlings and tailings. Messrs. Cox and Gibbons and Dr. Porter believe that Richards' results from the viewpoint of the quality of the tailings lead to the conclusion that sized feed is better than classified feed.

SUMMARY OF RESULTS.

Feed.	Copper Assay in Feed. Per Cent.	Percentage of Products.			Percentage of Total Copper.			
		Concentrates.	Middlings.	Tailings.	Concentrates.	Middlings.	Tailings.	Concentrates Plus Middlings.
Natural, 18 mesh–0	2.57	6.3	15.0	78.7	69.1	16.6	14.3	85.7
Sized, 18–20 mesh	1.51	4.1	4.5	91.4	61.7	6.8	31.5	68.5
Sized, 20–30 mesh	2.03	5.1	6.6	88.3	71.5	11.4	17.1	82.9
Sized, 30–40 mesh	2.48	6.9	4.9	88.2	75.1	9.9	15.0	85.0
Sized, 40–60 mesh	2.98	9.1	6.0	84.9	83.6	5.9	10.5	89.5
Sized, 60–100 mesh	4.42	12.9	6.6	78.5	81.4	7.9	10.7	89.3
Sized, 100 mesh–0	6.27	13.6	33.4	53.0	62.3	23.6	14.1	85.9
Classified, coarse end	4.68	14.1	9.5	76.4	76.8	11.8	11.4	88.6
Classified	2.62	7.5	8.5	84.0	73.0	12.3	14.7	85.3
Classified	2.03	5.3	7.0	87.7	67.0	16.0	17.0	83.0
Classified	1.88	5.3	6.8	87.9	69.9	15.1	15.0	85.0
Classified	2.13	5.9	7.0	87.1	72.0	12.3	15.7	84.3
Classified, fine end	3.22	4.4	20.7	74.9	52.6	23.7	23.7	76.3

The present investigation, a summary of the results of which are given in the table, seems to show that for the case in hand the classified feed is inferior to screened feed. The quantity of middling produced with sized feed is smaller except on the finest size and the percentage of total copper in the middlings is less. The following table has been prepared to show the disposition of the total copper on a milling basis assuming that the natural feed represents one mill process, the six sized feed tests represent a second process and the six classified feed tests, a third process.

[1] *American Institute of Mining Engineers*, Vol. XXXVIII, p. 556, Vol. XXXIX, p. 303.

Process.	Percentage of total copper.		
	Concentrates.	Middlings.	Tailings.
Natural feed.....	68.9	16.6	14.4
Sized feed.....	72.0	12.4	15.6
Classified feed	69.1	15.0	15.9

These figures also indicate the superiority of sized feed.

Note.—The writers of this abstract wish to express their gratitude to Messrs. Cox and Gibbons and Dr. Porter for having thrown further light on the problem of Wilfley table work, and to acknowledge that their results appear to be in favor of sized feed. The whole problem needs to be studied further and recent investigations by the abstractors on the size of fine free mineral which goes into Wilfley table tailings are expected to throw further light on the question. The standard 7-ft. laboratory table is not considered to be a good type of Wilfley table.

From the point of view of the mill man there are other points aside from the relative recoveries by sized and classified feeds, viz: the inefficiency of fine screens, the cost and wear of fine screens, the efficiency of mill classifiers. It is to be hoped that some large mill may be able to equip two sections so as to try out the two processes under exactly similar conditions in practical work.

Hydraulic Ore Classification for Wilfley Table Feed.[1]—Based on the theory of stratification on Wilfley table as effected by both free and hindered settling of large and small particles of heavy and light minerals, and also on the experimental results from a large number of tests on the transporting action of water films, the conclusion is drawn that sized feed to Wilfley tables will not give as good results as good classified feed.

One noteworthy point brought out by the experiments is that the transporting power of water on particles not in suspension is less for fine grains than for coarse. See typical table showing material transported on different grades and for different sizes. Other tables in the original article show similar results with increased water quantities.

TRANSPORTING POWER USING 2.5 POUNDS OF WATER PER UNIT OF WIDTH.

Per cent. Grade.	Proportion of Quartz.			Proportion of Pyrite.		
	8 to 16 Mesh.	16 to 30 Mesh.	40 to 150 Mesh.	8 to 16 Mesh.	16 to 30 Mesh.	40 to 150 Mesh.
1.25	0.0444	0.0580	0.0640	0.0308	0.0267	0.0100
1.5	0.0640	0.0720	0.1600	0.0444	0.0428	0.0200
2.0	0.1332	.....	0.3428	0.0940	0.0923	0.0780
3.0	0.3692	.....	0.8000	0.2370	0.2286	0.2460

[1] G. A. Overstrom, *Mines and Methods*, Vol. II (1911), pp. 159, 179; *Revue de Metallurgie Extraits*, Vol. VIII (1911), p. 746; *Mining World*, Vol. XXXIV (1911), pp. 451, 833.

The results lead to the conclusion that with classified feed the particles of concentrates being smaller than the gangue particles sink to the lowest stratum and are protected from the action of the water current and further that until they get so fine as to be held in suspension the finer the particle the closer it clings to the table and the better it resists the action of the film of water. In the case of sized feed the particles of heavy mineral are the same size as the light and therefore are more liable to be washed off into the middling or tailings.

The work of the McKesson sizing table discussed elsewhere in these abstracts bears out Mr. Overstrom's theory.

To prepare the best classified feed for the Wilfley tables and avoid the excess of water with the slimes which occurs in ordinary classification, an inverted classifier has been designed in which the slimes are taken out first and the products graded from fine to coarse up to the last which is the coarsest.

Development of Economical Ore Dressing Systems.[1]—The tendency of the times is to continually treat a lower and lower grade of ore. Extractions in gold and silver mills run up to over 90 per cent. by concentration and cyanide. Successful results are obtained in treating mixed sulphide ores and in treating copper ores containing less than 2 per cent. copper.

The improvements are due to better planning of mills, to improvements in machines and in wearing parts, to new processes, and last but not least to greater attention and thought on the part of the mill operators.

Recent Problems in Copper Milling.[2]—The development of the so-called porphyry copper mines has led to large capital expenditures for large mills and for the perfection of the milling operation to yield increased saving. This expenditure is justified by the fact that where the margin of profit on these low-grade ores is so small, a comparatively small increase in extraction may double the profit. Careful study has increased the extraction at the Arthur plant of the Utah Copper Company until it averages well above 70 per cent. and has reached as high as 77 per cent.—a figure previously considered to be impossible on this low-grade ore.

The process used has to be simple and involves the crushing of the ore quickly and cheaply to a size which will permit the separation of mineral and gangue, and the making of this separation as completely as possible with the fewest operations and machines and with the least possible production of slime. The exact flow sheets have been published

[1] F. E. Shepard, *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 412; *Mining Science*, Vol. LXIII (1911), p. 421; *Mining and Scientific Press*, Vol. CII (1911), p. 553; *Mining World*, Vol. XXXIV (1911), p. 781.

[2] *Mining and Scientific Press*, Vol. CIII (1911), p. 438.

in past volumes of MINERAL INDUSTRY and that of the Ray Consolidated mill is given elsewhere in this article (page 848–9).

An important problem in milling is the loss of copper in material finer than 200 mesh. Two methods of attack are open: the first being to still further reduce the slime produced in crushing, and the second to develop some method of concentrating this fine material. It does not appear that much advance can be made in lessening slime production, but the mills are making a careful study of possible improvements in the methods of settling and concentrating slime. A significant point is that screen analyses and chemical analyses of the mill tailings show that the material between 100 and 200 mesh is the lowest in copper.

Calculations of Percentage of Recovery.[1]—Previous formulas have not considered the middling product. The first of the following formulas gives the recovery and the second gives the weight of the material treated, the other factors indicated being known in both cases.

$$R=\frac{Cc(a-b)}{Ca(c-b)+Da(d-b)}$$

$$A=\frac{C(c-b)+D(d-b)}{R(a-b)}$$

Where A = weight of feed, B = weight of tailings, C = weight of concentrates, D = weight of middlings, a = tenor of feed, b = tenor of tailings, c = tenor of concentrates, d = tenor of middlings, R = percentage of recovery.

EXAMPLES OF MILLING PRACTICE

Calumet and Hecla Mill.[2]—This mill was described in Richards' "Ore Dressing," Vol. IV, pp. 1891–1895. Recently a regrinding plant has been installed to regrind the richer conglomerate tailings from the Wilfley tables and jigs. This material is brought by two distributing launders to (1).

1. Forty-eight Chile mills, 6 ft. diameter, 20-mesh needle slot screens. To (2).
2. Twenty-four hydraulic classifiers, 3 1/2×9 ft., with three compartments; first spigot to (3); second to (4); third to (5); overflow to (6).
3. Twenty-four Wilfley tables. Concentrates to smelter; middlings to (7); tailings to waste.
4. Twenty-four Wilfley tables. Products like (3).
5. Twenty-four Wilfley tables. Products like (3).
6. Forty-eight Wilfley tables. Products like (3).
7. Sixteen Wilfley tables. Concentrates to smelter; tailings to waste.

This regrinding plant treats 1400 tons in 24 hours. The mill building is 123×346 ft. with concrete floors and foundations throughout. The plant is divided into eight sections with one 250 horse-power induction

[1] C. W. Wright, *Mining Magazine*, Vol. V (1911), p. 38.
[2] H. C. Wright, *Engineering and Mining Journal*, Vol. XCI (1911), p. 1161.

motor for each section. An Allis-Chalmers tube mill and a Hardinge conical pebble mill are being tested as substitutes for the Chile mills.

Concentration in Joplin District, Missouri.[1]—Too little attention is believed to have been paid to the regrinding of middlings and to the proper dewatering of the feed to Wilfley tables. At the new mill of the Petersburg Land and Mining Company the ore is crushed in breakers and rolls to 7/8 in., and is then classified in Richards' classifiers and jigged in three sets of jigs. Jig middlings are reground to 3 mm. and again classified and jigged. All the fine material from both sets of classifiers is settled, classified in Richards' classifiers and treated on Wilfley tables. The tables also have dewatering boxes preceding them to eliminate unnecessary water from their feed.

Zinc Ore Dressing in Colorado.[2]—Roasting and magnetic separation followed by wet concentration is the scheme at the Wilson mill near Robinson, Colorado. The ore is a mixed sulphide with some siderite in a silicious gangue. Its average composition is 11 per cent. silica, 23.6 per cent. iron, 10.4 per cent. lead, 14.8 per cent. zinc, 4 per cent. lime, 35.6 per cent. sulphur, 7 oz. silver and 0.07 oz. gold per ton.

The ore from the mine is dumped to (1).
1. Grizzly with 1 1/2-inch spaces. Oversize to (2); undersize to (3).
2. Symons No. 5C gyratory breaker crushing to 2.5 in. To (3).
3. Belt conveyor with electromagnet in the end pulley to remove bits of iron and steel. To (4).
4. Shaking launder 12×3 ft., with 1-in. round holes at the end. Oversize to (5); undersize to (6).
5. Symons No. 3C gyratory breaker crushing to 1 in. To (6).
6. Belt and bucket elevator. To (7).
7. Trommel with 1/4-in. holes. Oversize to (8); undersize to (9).
8. Rolls, 14×27 in., set 3/8 in. apart. To (6).
9. Bin with 1000 tons capacity and having distributing conveyor above it. By eight discharge gates and plunger feeders to (10).
10. Belt conveyor and bucket elevator to (11).
11. Fairchild tower dryer, 5×8 ft. and 20 ft. high. In the tower are 14 pans in series one above the other. These pans are normally horizontal but when each is filled with about 75 lb. of ore it dumps automatically into the next pan below. The ore passes through in 3 1/2 minutes. The capacity is about 9 tons per hour and the coal consumption in 7 to 8 lb. per ton of ore dried. By plow conveyor to (12).
12. Bucket elevator. To (13).
13. Trommel with 6-mesh screen. Oversize to (14); undersize to (15).
14. Rolls, 14×27 in. To (12).
15. Trommel with 10-mesh screen. Oversize to (16); undersize to (17).
16. Rolls, 14×27 in. To (12).
17. Belt conveyor delivering to dry ore bins of 275 tons capacity. By elevator to (18).
18. Oscillating plow conveyor to feed hoppers and plunger feeders to (19).
19. Five circular multiple-hearth Dewey roasting furnaces. Six hearths, 14 ft. diameter, one above another. Rabbles make revolution in 20 seconds. Ore passes through in about 2 hours. Capacity per furnace is 1 ton per hour. Temperature of three lower hearths is 1100° to 1150° F., maintained by Bristol recording pyrometer. Dust sucked off to dust catcher whence it is withdrawn and mixed with roasted ore. To (19).
19. Revolving cooling drum, 30×3 ft., inclined 1 in. per foot and making 4 r. p. m. Cooling water is sprayed over the drum. To (20).
20. Elevator. To (21).
21. Edison 10-mesh screen. Oversize (fritted ore) to (22); undersize to (23).
22. Rolls. To (20).
23. Traveling plow conveyor delivering to 500-ton bin. To (24).
24. Five Fairchild magnetic separators of the belt type. Each has three sets of magnets in series using 1.5, 3, and 6 amperes respectively. Each set has 10 magnets of alternate polarity. Magnetic iron product to smelter; residue (zinc-lead-silica) by belt conveyors and elevators to (25).
25. Pulp mixer. To (26).
26. Dewatering box. Settlings to (27); overflow to (30).
27. Classifier. First spigots to (28); second and third to (30); overflow to waste.

[1] J. H. Dietz, *Engineering and Mining Journal*, Vol. XCI (1911), p. 868.
[2] H. C. Parmelee, *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 91; *Mining Magazine*, Vol. IV (1911), p. 230.

28. Eight Wilfley tables. Lead concentrates; zinc concentrates; middlings to (29); tailings to waste.
29. One Wilfley table. Lead concentrates; zinc concentrates; tailings to waste.
30. Card table. Lead concentrates, zinc concentrates; middlings to (31); tailings to waste.
31. Card table. Products like (29).

The iron product contains 6.5 per cent. silica, 46 per cent. iron, 3 per cent. zinc, 2.2 per cent. lead, 0.05 oz. gold and 4.5 oz. silver per ton. The zinc product contains 7 per cent. silica, 41 to 42 per cent. zinc, 11 per cent. iron, 6 per cent. lead and 8 oz. silver per ton. The lead product contains 67 per cent. lead, 6 per cent. zinc, and 34 to 38 oz. silver per ton. The tailings contain about 9 per cent. zinc, and 1.2 per cent. lead and amount to 15 to 18 per cent. of the ore treated.

Provision is made for automatic sampling of the ore just after the crude ore bins. Electric power is used. Water is pumped from Ten Mile Creek. The labor is five men per shift when crushing, and four men when the crushing department is idle.

Colorado-Toledo Mill.[1]—This mill in Summit County, Colorado, treats a mixed sulphide ore containing lead, zinc, gold and silver.

The mill bins feed a No. 4 Austin gyratory breaker and the crushed product is elevated to No. 1 trommel 1 1/4 in., making oversize back to breaker and undersize to mill bins holding 270 tons. From the bin the ore goes by belt conveyor and No. 2 bucket elevator to the trommel line as follows: No. 2 trommel, 8 mm., feeding coarse rolls delivering back to No. 2 elevator; No. 3 trommel, 5 mm., feeding No. 1 jig; No. 4 trommel, 3.5 mm., feeding No. 2 jig; No. 5 trommel, 2 mm., feeding No. 3 jig; No. 6 trommel, 1 mm., feeding No. 4 jig; four-spigot hydraulic classifier of which first spigot goes to No. 5 jig and the second, third and fourth spigots and overflow go to Card tables. No. 1 and No. 2 jigs are two-compartment and make galena on the first sieve, impure galena on the second, and tailings to No. 2 rolls and back to No. 2 elevator. No. 3, 4 and 5 jigs are four-compartment and make galena on the first compartment, galena and pyrite on the second, pyrite and zinc on the third and zinc on the fourth; the tailings of No. 3 go to No. 2 rolls and the rest of the tailings to waste. The zinc-pyrite middling is reground and goes to the hydraulic classifier. The jig sieves are 3 1/4 mm. on Nos. 1 and 2, 2 mm. on Nos. 3, 4 and 5. The No. 5 jig is bedded with concentrates from No. 3 jig. The tables all make galena-pyrite product, pyrite-zinc middling, zinc product, and tailings to waste. All the mill products are dewatered and the muddy water settled in a large tank. The settled slime is delivered to a Card slimer which makes the same products as the other tables.

Frontenac Mill in Gilpin County, Colorado.[2]—This mill represents

[1] W. B. Le Wald, *Mining and Scientific Press*, Vol. CII (1911), p. 261.
[2] H. C. Parmelee, *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 241; C. J. Downey, *Mining Science*, Vol. LXIV (1911), p. 52; *Mining Magazine*, Vol. V (1911), p. 228.

a wide departure from the old style Gilpin County mill which had stamps, plates, and Gilpin County bumping tables. The ore is mainly pyrite with some galena and tetrahedrite in a quartz and granite gangue. The gold is with the pyrite and the silver with tetrahedrite. The gold is 0.30 to 0.35 oz. per ton and the silver 4 to 6 oz. The ore comes from the mine to (1).

1. Receiving bin holding 250 tons. To (2).
2. Shaking grizzly. Oversize to (3); undersize to (4).
3. Blake breaker, 13×24 in., run only 8 hours out of 24, crushing to 2 in. By 16-in., belt and bucket elevator to (4).
4. Bin 250 tons. By plunger feeder to (5).
5. No. 1 rolls, 16×36 in., 60 r. p. m., set 1/4 in. apart. By 16-in. belt elevator to (6).
6. Flood automatic sampler. Sample worked down. Rejects to (7).
7. Two trommels, 4×6 ft., with 9-mm. slotted holes. Oversize to (8); undersize to (9).
8. Two No. 2 rolls, 16×36 in., 80 r. p. m., set close together. By elevator to (7).
9. Two No. 2 trommels, 4×6 ft., with 3-mm. slotted holes. Oversize to (11); undersize to (10).
10. Two No. 3 trommels, 4×9 ft., with 1.5-mm. slotted holes. Oversize to (12); undersize to (14).
11. One double two-compartment jig. Concentrates to (13); tailings to (8).
12. Two double two-compartment jigs. Concentrates to (13); tailings to (19).
13. Shaking launder with perforated screen bottom for a portion of its length. Slope 1 in. per foot. Concentrates to smelter.
14. One 5-ft. cone thickener followed by three Flood hydraulic tubular classifiers, each with three spigots. Overflow of cone to (21); first spigots to (15); second to (16); third to (17); overflow of classifiers to (18).
15. Two Card tables. Lead concentrates to smelter; pyrite concentrates to smelter; middlings to (19); tailings to waste.
16. Two Card tables. Products like (15).
17. Two Card tables. Products like (15) except middlings to (21).
18. One Card table. Products like (17).
19. One 5-ft. cone thickener. Thick pulp to (20); overflow to (21).
20. Two 6×12-ft. tube mills, 21 r. p. m., silex lining, loaded with 7000 lb. of Norway pebbles. Only one tube in operation grinding 60 tons in 24 hours, to pass 40 mesh. To (21).
21. Belt elevator, 16 in. wide. To (22).
22. Two 5-ft. cone thickeners. Thick pulp to (23); overflow to (29).
23. Six Flood classifiers each with three spigots. First spigot to (24); second to (25); third to (26); overflow to (27).
24. Three Card tables. Pyrite concentrates to smelter; middlings to (28); tailings to waste.
25. Three Card tables. Products like (24).
26. Three Card tables. Products like (24).
27. One Card table. Products like (24).
28. Three Card tables, one for (24), one for (25) and one for (26) and (27) together. Concentrates to smelter; tailings to waste.
29. Three 8-ft. Callow cones. Spigots to (30); overflow to (32).
30. Three No. 3 Deister tables. Pyrite concentrates to smelter; middlings to (31); tailings to waste.
31. One No. 3 Deister table. Pyrite concentrates to smelter; tailings to waste.
32. Dewatering tanks. Water used as wash water on tables.

The mill tailings are sampled by a Flood automatic tailings sampler. The concentration is 4 tons into one and the recovery is 85 per cent. of the gold, 78 per cent. of the silver and 75 per cent. of the lead and copper. The following table shows the tenor of the products:

	Ounces Gold per ton.	Ounces Silver per Ton.	Per cent. Lead.	Per cent. Copper.	Per cent. Silica.
Jig concentrate	0.70	12	10	1.20	10
Card pyrite concentrate	0.60	11	....	2.00	9
Card lead concentrate	5.00	12	37	....	....
First slime department	0.45	12	....	3.00	20
Second slime department	0.40	12	....	3 to 4	30

The mill is run by electric power. Water is pumped from Clear creek after passing several settling basins for removal of sand and slime.

New Hercules Mill, Wallace, Idaho.[1]—The ore contains lead, silver, iron and a small percentage of zinc in a silicious gangue. This mill was put in operation in April, 1911. Ore is delivered through an adit level to the sorting plant where the waste is picked out and the high-grade ore and mill stuff separated. The mill ore is then crushed in a breaker to 2 1/2 in. and transported 3500 ft. by aerial tram to the railroad which then takes it to the 1500-ton mill ore bin.

The mill is built in two sections, each being housed in a separate structure, the two being 80 ft. apart. The first section just below the ore bins contains the crushing machinery, jigs, screens, and classifiers. The second section or slime plant contains settlers and classifiers, traveling screens, tables and vanners.

The ore is discharged from the bins through plunger feeders upon a 70-ft. belt conveyor that extends lengthwise of the bins; this conveyor discharges upon another at right angles delivering the ore to four-section trommel system—two 12-mm., one 20-mm., and one 36-mm. The oversize of 36 mm. goes to fine breaker; the undersizes of the 20- and 36-mm. trommels go to the jigs while that of the 12-mm. passes to three Franz traveling belt screens of 4, 8, and 20 mesh respectively. The oversizes of the Franz screens go to jigs while the undersize of 20 mesh flows to the slime plant. The tailings from the coarse jigs are returned to middling trommels; the jig middlings are dewatered by shaking screen and go thence to a set of rolls following which it is passed over traveling screens, the oversize from which is reground to 20 mesh in Huntington mills and joining the undersize flows to the slime plants.

The most important product of the mill is made in the 16 two-compartment jigs and consists of the oversizes of 4, 8 and 20-mesh screens.

In the slime plant, material from all the 20-mesh screens and from the three 6-ft. Huntington mills is sized and classified by traveling screens, 4-spigot classifiers and Callow tanks, the products being treated on eight Wilfley tables, eight Franz tables and six vanners. The Franz table has on its deck a system of parallel riffles or grooves directed toward outlet slots through which the coarser product is discharged; also a pair of pans and deflector ribs for directing the stream of finer mineral to the pans where the finer concentrates and gangue are separated. The tail end of the deck may be adjusted with respect to the other part of the deck to regulate the speed of concentration of the material.

The present mill capacity is 400 tons per 24 hours but this is soon to be increased to 600 tons. The concentration is 8 tons into one.

The shipping bins have two lines of conveyors extending on an incline from the ground to the top of the building. The structure is 20×40

[1] W. A. Scott, *Mining and Scientific Press*, Vol. CII (1911), p. 724.

ft., and 40 ft. high, divided into three compartments—one each for jig concentrates, slime concentrates, and tailings. Each conveyor has a length of 110 ft., one carrying the two classes of concentrates, and the other the tailings, the latter to be hauled or laundered away and not deposited in the stream.

Concentrating Platiniferous Copper Ores at Rambler Mine, Wyoming.[1]—The Rambler ore is a sulphide associated with diorite and talc. Besides the copper sulphides there are the various secondary oxides, carbonates, etc. The platinum mineral sperrylite has been isolated. In the experimental mill the ore is crushed in 7×10-in. Blake breaker, and 14×20-in. rolls and elevated to 20-mesh trommels. The oversize is recrushed in a second set of 14×20-in. rolls and elevated back to the trommels. The undersize passes to a three-spigot classifier sending spigots to three Wilfley tables and overflow to two canvas tables. The Wilfley tables are run so as to take off, first a precious metal streak, second a copper concentrate carrying the bulk of the sulphides, third a middling of oxides and carbonate and fourth a tailing. The following table shows some of the assays of the various Wilfley products.

Product.	Ounces per Ton.					Per cent. Copper.
	Gold.	Platinum.	Silver and Palladium.	Iridium.	Total.	
Mill feed					1.27	2.8
Table No. 1.						
Precious metal streak	4.01	11.21	1.76	3.21	20.19	19.7
Copper concentrate		0.36	1.21		1.57	7.3
Middling			1.01		1.15	2.7
Table No. 2.						
Precious metal streak	2.06	3.24	1.63	1.03	7.96	15.4
Copper concentrate		0.12	1.12		1.24	7.9
Middling			0.91		1.07	1.8
Mill tailings					0.90	2.3

Boston Consolidated Mill, Utah.[2]—This mill which was originally fitted with Nissen stamps, Wilfley tables, and suspended vanners, as described in Richards' "Ore Dressing," Vol. IV, page 1850, has been dismantled and refitted to correspond to the practice of the mill of the Utah Copper Company. Mr. Rice has written an obituary in which he gives a detailed account of the mill and its work and brings out its good points. Incidentally he makes the claim that the present race for capacity at a sacrifice of extraction is a poor policy, and that in the end there will be a reversion to the opinion that a close saving is better than high tonnage.

Ray Consolidated Mill.[3]—This mill at Hayden, Arizona, is to treat ore containing chalcocite and cupriferous pyrite finely disseminated in

[1] *Metallurgical and Chemical Engineering*, Vol. IX (1911), p. 75.
[2] C. T. Rice, *Engineering and Mining Journal*, Vol. XCI (1911), pp. 816, 862.
[3] C. A. Dinsmore, *Mining and Scientific Press*, Vol. CIII (1911), p. 282.

schist. It is estimated that the ore already developed amounts to over 80 million tons averaging 2.17 per cent. copper.

There are two shafts and at each shaft the ore is dumped from the skip to the receiving bin. It passes from the bin through steel gates and over grizzlies to No. 8 gyratory breakers, and thence through two sets of Garfield rolls, 72×20 in. The final product, 1-in. maximum size, is distributed by belts to bins having a total capacity of 20,000 tons. From these bins the ore goes by railroad cars to the mill.

The mill is now operating three units of what is to be a 10,000 ton plant of eight units. The receiving bins hold 25,000 tons. The description of each unit is as follows: From bin the ore goes to a conveyor belt bringing it to a 30-in. 15-bucket elevator, which takes it to six impact screens of 7 and 8 mesh. The oversize goes to two 16×42-in. Garfield rolls (dry crushing) regrinding it until it passes through the screens. The undersize goes to six Garfield roughing tables. Concentrates from the tables go back to an elevator and are discharged to a four-compartment Janney classifier, the primary, where it is classified and returned to Garfield Chile mills, grinding through two No. 77 screens.

The overflow from the primary classifier goes to 10-cone tanks which feed Isbell vanners. The second compartment of the classifiers discharges to second classifiers, four-compartment, and these feed Isbell vanners with corrugated belts. The Chile mill product goes to three-compartment improved Janney classifiers which feed to four Garfield roughing tables and to Wilfley tables. On the Wilfleys the first finished concentrates and tailings are made. The overflow from secondary classifiers goes to 10-cone tanks, is siphoned off, and the pulp taken for the different vanners. The mill tailings go to the tailrace, being automatically sampled every 30 minutes. The concentrates go to receiving tanks under the mill, are elevated by centrifugal pumps, pass through automatic samplers and go to concentrates filter bins. The water is sucked off by a vacuum pump and put back into circulation again. The overflow of all the cone tanks is settled and used again in the classifiers and roughing tables. Water is pumped 1 mile from wells at the confluence of the San Pedro and Gila rivers. The mill is run by electricity from a central steam plant. The ratio of concentration is about 20 tons of ore into one.

The mill is patterned after the practice of the Utah Copper Company at Garfield, Utah, treating a similar ore.

Use of Canvas Plant in Concentration.[1]—California has been the home of the canvas plants for saving the fine rich concentrates which pass into the tailings of vanners in gold mills. The following plants are

[1] W. H. Storms, *Mining World*, Vol. XXXIV (1911), pp. 581, 681, 783.

described in detail: Golden Gate, Keystone, Gold Bank, Utica-Stickles, Gates, Zeila, Argonaut, Darrow, and White. The last named is in Australia; all the others are in California.

Washing Brown Iron Ores.[1]—There are three methods of obtaining commercial brown iron ore: hand screening; washing; and washing and concentrating.

Hand screening is falling into disuse owing to the fact that the oversize going to the furnace may contain considerable foreign matter and the undersize which is waste may contain considerable iron.

In washing, the ore comes first to a grizzly with 3-in. spaces. The undersize goes to the log washer; the oversize may be broken by hammers until it passes through, or it may go over the end of the grizzly to a breaker and thence to log washers, or it may go to screen making undersize to log washer and oversize to loading bin. A nozzle, delivering water at about 100 lb. pressure, is of great service in washing the oversize on the grizzly. The log washers usually have a trommel preceding them, to take out material too coarse for the log washers and also to give the ore further washing. The oversize of the trommel goes to picking belt, from which it is delivered to breaker and thence to another picking belt and finally to loading bin. Often the breaker is omitted and only the picking belt used. The undersize goes to the log washers. These eliminate the clay, loam and sand, but will not take out the pyrite, quartz, and limestone. The iron ore passing out the head of the washer goes to a sand screen or trommel with 1/8-in. holes. The oversize passes on to a picking belt for delivery to loading bins. The undersize is either sluiced to jigs or sent to waste. Where there is considerable silicious material in the fine ore it is advisable to use a 1-in. sand screen. Up to the present time jigs are the only concentrating machines that have been used on the undersize, but it is believed that there is a field for Wilfley tables, and possibly also for magnetic or electrostatic separation.

Portland Canal Mining Company's Mill.[2]—This new mill treats an ore containing galena and pyrite, both carrying gold and silver with a small amount of blende and occasionally copper pyrites, in a gangue of quartz mixed with fragments of argillite. The ore is brought by aerial tram to (1).

1. Ore chute holding 75 tons and bin holding 175 tons. By Challenge gate to (2).
2. Sturtevant breaker, 6×16 in., crushing to 1 in. By belt conveyor to (3).
3. Mill ore bin, holding 175 tons. By roll-feeder to (4).
4. One set rolls, 12×24 in. By No. 1 elevator to (5).
5. No. 1 trommel 5 mm. Oversize to (16); undersize to (6).
6. No. 2 trommel with 2- and 3-mm. holes. Over 3 mm. to (10); 3 to 2 mm. to (11); under 2 mm. to (7).

[1] H. S. Geismer. *Bulletin* No. 56, American Institute Mining Engineers, August (1911), p. 643; *Iron Age*, Vol. LXXXVIII (1911), p. 528.

[2] W. F. Robertson, *Canadian Mining Journal*, Vol. XXXII (1911), p. 17; *Mining Science*, Vol. LXIII (1911), p. 64.

7. Hydraulic classifier with two spigots. First spigot to (12); second to (13); overflow to (8).
8. Four conical iron settling tanks in series. First and second spigots to (14); third and fourth to (15); overflow to (9).
9. Large rectangular slime tanks. Overflow to waste.
10. No. 1 jig, four compartments. First discharge to (20); second and third to (21); fourth to (16); first hutch to (20); second, third and fourth to (21); tailings to waste.
11. No. 2 jig. Products like (10).
12. No. 3 jig. Products like (10).
13. No. 4 jig. Products like (10).
14. Two Wilfley tables. Galena concentrates to (20); pyrite concentrates to (21); middlings to (17); tailings to waste.
15. Two Frue vanners. Concentrates to (21); tailings to waste.
16. No. 2 rolls. To (18).
17. Lane 10-ft. Chile mill. To (18).
18. No. 2 Elevator. To (19).
19. No. 3 trommel, 2 mm. Oversize to (16); undersize to (7).
20. Lead concentrates bin. To smelter.
21. Iron concentrates bin. To smelter.

The crushing department of the mill will handle 100 tons per day but the jigs and tables will handle only 50 tons. The ore contains 3.7 per cent. lead, 0.15 oz. gold and 14 oz. silver per ton. The iron concentrates contain 0.4 to 0.5 oz. gold and 15 to 20 oz. silver per ton. The lead concentrates contain 65 to 75 per cent. lead, 0.4 to 0.5 oz. gold and 35 to 40 oz. silver per ton. No zinc concentrates are saved as they assay only 1.5 oz. silver per ton. The extraction is about 80 per cent. The mill has a wooden frame sheathed with double boarding with paper between. Water-power is used under 105-ft. head. One 3-ft. Pelton wheel drives the jigs and tables and one 6-ft. wheel drives the rest of the mill.

Braden Copper Mill, Chile.[1]—This mill to treat low-grade copper ore is designed somewhat after the lines of the Utah and Nevada porphyry mills but with coarser preliminary crushing and with the introduction of jigs to catch the coarse mineral. The ore is brought to (1).

1. Coarse ore bins holding about 1400 tons. By belt conveyors to (2).
2. Two No. 7 1/2 gyratory breakers, two sets of 48-in. rolls, and six sets of rolls 36-in. diameter and 15 in. faces. Crushed product by belt conveyors to (3).
3. Fine ore bins holding about 4000 tons. By belt conveyors over automatic weigher to (4).
There are four units in the mill with a total capacity of 3000 tons. Only one unit will be described.
4. Two trommels. Oversize to (5); undersize to (6).
5. Rolls. By elevator to (6).
6. Two sets of Woodbury two-compartment jigs. Concentrates to (22); tailings to (7); slimes to (9).
7. Five 2.5-mm. trommels. Oversize to (12); undersize to (8).
8. Classifier with three spigots. First spigots to (12); second and third to (10); overflow to (9).
9. Two conical settling tanks. Settlings to (10); slimes to (20).
10. Eight Wilfley tables. Concentrates to (22); middlings to (11); coarse tailings to (12); fine tailings to (23).
11. Four Wilfley tables. Concentrates to (22); tailings to (12).
12. Sloughing off tank. Spigots to (13); overflow to (14).
13. Two Chile mills with 10-mesh screens. To (14).
14. Three Yeatman classifiers. First spigots by Frenier pump to (16); second and third to (17); overflow to (15).
15. Eight conical settling tanks. Spigots to (17); overflow to (20).
16. One Chile mill with 14-mesh screens. To (14).
17. Eight Wilfley tables. Concentrates to (22); middlings of coarse tables to (18); middlings of fine tables to (19); tailings to (23); slimes to (20).
18. Four conical settling tanks. Spigots to (21); overflow to waste.
19. Six conical settling tanks. Spigots to (21); overflow to waste.
20. Six conical settling tanks. Spigots to (21); overflow to waste.
21. Twelve vanners. Concentrates to (22); tailings to (23).
22. Automatic concentrates sampler. Concentrates to bins.
23. Automatic tailings sampler. Tailings to waste.

The ore assays 2.70 per cent. copper. It concentrates 9 tons into one and the extraction is 65 per cent.

[1] P. Yeatman, *Engineering and Mining Journal*, Vol. XCII (1911), p. 1186.

Treatment of Coal

Testing Apparatus in Washing Coal.[1]—A little glass cylinder with screen in one end is partly filled with coal and then jigged up and down in water. When the proper stratification has taken place the cylinder is removed and its contents taken out in slices and each slice analyzed. From the results it is possible to plot curves which will compare the yield and tenor of the products under varying conditions of washing.

Washing Coal.[2]—A study has been made on the theory of coal washing as revealed by investigations of different layers in the jig beds of coals of various sizes, composition and character. At the outset the total ash content of the coal is divided into the combined ash which is a constitutional part of the coal and the free ash which is present in the form of slate or other impurities in sufficiently large sizes to be freed by crushing. A large number of curves has been made showing how the recovery and the purity of the products are affected by the relative amounts of the two forms of ash present in the unwashed coal.

Washing and Sorting Small Coal.[3]—The necessity of treating small coals may be shortly put as follows: (*a*) The ability to use the smaller dry coals when separated into suitable sizes and freed from dirt; (*b*) The necessity for increasing the value of the commodity. Washed coal yields a superior quality of coke and the washing removes waste on which freight would have to be paid in transporting to market.

For screening use is made of shaking screens, vibratory screens of the Zimmer and Caracole types, concentric trommels and gyratory screens of the Coxe type.

For washing the machines used are feldspar jigs for fine coal, Sheppard or Luhrig jigs for nut coal, Stewart jig for fine and nut coal, Baum jig for all coal up to 3 1/8 in., Blacketts barrel washer. The washed coal is drained on conveyors and in settling tanks and stored in bins. For breaking coal either rolls of the fluted or toothed type or disintegrators are used. The original article contains detailed descriptions and comparisons of the above mentioned machines.

Washing and Sorting of Small Coal.[4]—The mining of less pure coal, the demand for clean coke for smelting, and of clean coal with low ash for steam purposes, together with the utilization of fine coal in briquettes have all led to the increased use of coal washing. The cost in England ranges between 2 and 12 cents per ton, being done more cheaply in the larger plants.

[1] R. A. Henry, *Revue des Mines*, Series IV, Vol. XXXIV (1911), p. 68; *Revue de Metallurgie Memoires*, Vol. VIII (1911), p. 804.
[2] S. Hanappe, *Revue des Mines*, Series IV, Vol. XXXIV (1911), p. 75.
[3] G. Roblings, *South Wales Institute of Engineers*, Vol. XXVII (1911), p. 313; *Colliery Guardian*, Vol. CI (1911), pp. 119, 169.
[4] N. T. Williams, *Cassier's Magazine*, Vol. XL (1911), p. 606.

In general the practice in South Wales is to wash only the coal that passes through 1 or 1 1/4-in. grizzlies. This undersize is classified into different sizes by screens and each size is washed separately. The washed coal is drained and sometimes sprayed with clean water before shipment. The waste discharged from the washers is drained and sent to waste dump. The finer sized coal from the washers is settled in tanks and shipped to market. All the dirty water from screens, settling tanks, etc., is settled in large spitzkasten, and the settlings, if pure enough, are added to the fine washed coal or, if dirty, are treated by fine coal washers. If the coal is to be coked then the coarser sizes are pulverized.

Crushing may also be used on the raw coal to free the shale and pyrite before washing. Among the machines used are rolls, both fluted and toothed, the latter being applicable when the coal is very hard or associated with clayey material; Carr's disintegrator; Devil disintegrator. The moving of the coal is generally effected by chain and bucket elevators, shoots, screw and various other forms of conveyors.

For screening the coal use is made of trommels, usually of the concentric type, Vibrometor screens and Callow traveling belt screens. Below 1/4-in. size screens are replaced by spitzkasten for classifying the coal.

The removal of dust is effected by suction fan and dust chamber; the suction being applied to the coal when it falls on the screens.

The washing machines may be divided into (1) jigs with fixed sieve and movable plunger, or with movable sieve, used for washing nut coal; (2) jigs of these same types but with a feldspar bed, used for washing fine coal; (3) trough washers; (4) bumping tables. The ordinary fixed sieve jig has the plunger at one side of the sieve. The Diescher jig has the plunger placed beneath the sieve. The Stewart, Christ and Primus jigs all have movable or reciprocating sieves. The Shepard jig is a movable sieve jig having a feldspar bed.

The trough washing type of machines is represented by the Elliot trough washer.

The Craig bumping table consists of a Y-shaped tray mounted on wheels running on inclined rails and receiving a bumping motion by cams and springs.

The drying of washed coal is effected by draining bins, perforated bucket elevators, perforated pan conveyors and cylindrical driers heated by furnaces.

Costs of plants for the given capacities in 10 hours are as follows: 200-ton plant, $20,000; 500-ton plant, $60,000; 1600-ton Humboldt plant, $200,000; 100-ton Elliot plant, $3750; 400-ton Elliot plant,

$8000; 1000-ton Elliot plant, including buildings, $25,000 to $50,000; 400-ton Craig plant, $3500.

Preparation of Anthracite.[1]—This subject is treated exhaustively by Mr. Sterling. Owing to lack of space it is impossible to give a complete abstract and only the following outline can be given of the ground covered so fully in the original article. Commercial sizes of anthracite. Loading. Typical scheme of dry preparation. Typical scheme of dry and wet preparation. Typical scheme of wet preparation. Proportion of different commercial sizes produced. General outline of process. Disposal of clean coal and waste. Standard allowable percentages of bone, slate, etc., permitted in each size of coal. Machinery in general. Fixed screens of bars or plate. Trommels. Shaking screens. Comparison of plate and wire screens. Gyrating screens. Oscillating or movable-bar screens. Rolls. Roll teeth. Roll speeds and roll sizes. Hand picking. Jigs. Handling jig products by elevators and conveyors. Gravity pickers. Mechanical picking table. Other mechanical pickers. Conveyors and elevators of various types. Chutes. Automatic feeders. Power. Labor. Loading pockets. Water used. Costs of installation. Cost of operation.

Peck's Shaft Anthracite Breaker.[2]—In this breaker located at Peckville, Pennsylvania, the coal is dumped at the surface to (1).

1. Hopper. By roller feeder to (2).
2. Scraper line 228 ft. long, inclined 30 degrees. To (3).
3. Lump coal shaking screens. Oversize to (5); undersize to (4).
4. Grate coal shakers. Oversize to (7); undersize to (12).
5. Picking belt. Slate to waste; boney coal to (16); coal to (6).
6. Rolls, 27×36 in. To (8).
7. Rolls, 32×36 in. To (8).
8. Elevators. To (9).
9. Dry coal shakers making egg, stove, nut and pea sizes to (10), and final undersize to (15).
10. Dry spiral slate pickers, one for each size. Coal to (11); boney coal to (16).
11. Picking chutes. Slate to waste; boney coal to (16); coal to bins.
12. Mud screens sprayed with water, making egg, stove, nut, and pea sizes to (13) and undersize to (15).
13. Wet spiral slate pickers, one for each size. Coal to (14); boney coal to (16).
14. Picking chutes. Slate to waste; boney coal to (16); coal to bins.
15. Fine screens making pea, and Nos. 1, 2, and 3 buckwheat sizes to bins.
16. Hazelton jigs, one each for egg, stove, nut and pea sizes. Coal to bins; slate to waste.

All the screens in the breaker are of the shaking type. The toothed rolls are made of manganese steel and in segments, not individual teeth. These are more economical than individual teeth of ordinary steel. The dry spiral slate pickers are differently designed from the wet pickers owing to the difference in sliding properties of wet and dry material. Wet treatment avoids dust, allows mechanical washing and has less trouble from clogging of screens, but it more expensive on account of pumping, the breaker is more sloppy and the refuse is not so simply disposed of. The breaker is of wood but is fully equipped with auto-

[1] P. Sterling, *Bulletin American Institute Mining Engineers*, No. 58, October (1911), p. 749; *Colliery Guardian*, Vol. CII (1911), pp. 918, 969, 1021, 1123.
[2] E. B. Wilson, *Mines and Minerals*, Vol. XXXI (1911), p. 513.

matic sprinklers and fire extinguishing apparatus. The rolls are run at 125 r. p. m. or about one-half the usual speed. This low speed gives better results. The lip screens on the shipping bins deliver any undersize by chutes to a pocket feeding into elevator (8). Any rejected car of coal is also delivered into this pocket.

Taylor Anthracite Breaker near Scranton, Pennsylvania.[1]—This new reinforced concrete breaker may be considered to have three streams of coal passing through it which may be identified by the name of the screens over which they are subsequently sized: the main screen coal, the mud screen coal and the pony-screen coal or recrushed bone coal. All of the screens are of the shaking type, all of the crushing is done in rolls and the cleaning is done by hand picking, Emery pickers, spiral pickers, and jigs. The complete scheme of the breaking and washing will be found in the original article.

Luhrig Washer at Marianna Coal Mine.[2]—The coal is dumped upon (1).

1. Bar screens with 1 1/4-in. holes, and dumping weigh basket. Oversize to (2); undersize to (4).
2. Steel hoppers holding about 4 tons each. By bottom plate feeders to (3).
3. Four plate conveyors 6 in. wide, 45 ft. long. Each pan is 6 ft. long and has vertical sides about 8 in. high and open ends. Speed 40 ft. per minute. Three men per conveyor pick out slate which goes by hopper and conveyors to waste bins, and bone which goes by hoppers and conveyors to crusher and thence to (11). Coal to loading bins.
4. Two 12×36-in. scraper conveyors and two 50-in. continuous bucket elevators. To (5).
5. Three 8×24-ft. hexagonal trommels with 3/4-in. wire screens. Oversize (nut coal) to (6); undersize (slack) to (7).
6. Hoppers delivering either to railroad cars or to (8).
7. Hoppers delivering either to railroad or to (8).
8. Four 10×20-in. scraper conveyors, two delivering nut coal to (9) and two delivering slack either to (9) or to (10).
9. Hoppers over the discharge ends of the conveyors (3). These serve to deliver the proper proportion of slack and nut to the lump coal so as to make run-of-mine for shipments.
10. A scraper conveyor, 12×36 in. The top strand receives slack from (8) and delivers to raw coal bin (11); the bottom strand receives washed coal from (17), and delivers to (11).
11. Bins 98 ft. long and 32 ft. wide, total capacity 1500 tons of washed slack, divided by transverse partitions into compartments about 16 ft. long. One of these is for raw coal from (10) to (12); one is for crushed bone from (3) and for washed bone from (18) an one is for slate from (19). These bins deliver to larries running to coke plant, boiler house for bone, and dump for refuse.
12. Williams No. 3 crusher. By continuous bucket elevator to (13).
13. One 7×14-ft. trommel with 3/16-in. round holes. Oversize to (12); undersize by hopper and dividing launders to (14).
14. Sixteen primary Luhrig jigs. Coal to (17); refuse to (15).
15. Tank and elevator to (16).
16. Four secondary Luhrig jigs. Boney coal to (18); slate to (19).
17 Bin and elevator to (10) or to railroad cars.
18. Bin and elevator to (11).
19. Bin and elevator to (11).

The washer has a capacity of 150 tons per hour. The following grades may be produced: (1) Mine run; (2) 3-in. lump; (3) 1 1/4-in. lump; (4) 3/4-in. lump; (5) 4-in. mine run; (6) 4-in nut; (7) 1 1/4-in. nut; (8) nut and slack; (9) slack; (10) washed slack. The products analyze as follows:

[1] M. A. Walker, *Coal Age*, Vol. I (1911), p. 334.
[2] A. C. Beeson and F. W. Parsons, *Engineering and Mining Journal*, Vol. XCI (1911), p. 228.

Grade.	Volatile Per cent.	Fixed Carbon Per cent.	Ash Per cent.	Moisture Per cent.	Sulphur Per cent.
Raw coal run of mine	36.11	56.19	6.78	0.92	0.99
Raw coal, slack	33.25	53.55	12.00	1.20	1.50
Primary washed	35.46	58.16	5.28	1.10	1.18
Secondary washed	34.60	56.18	8.10	1.12	1.46
Final refuse, slack	16.24	25.20	56.56	2.00	6.12

Coal Washing at Wencelaus Mine at Molke in Lower Silesia.[1]—The process used includes the following operations: (1) Preliminary screening; (2) picking on belts to remove waste from coarse sizes; (3) removal of dust by suction fans; (4) jigging coarser sizes; (5) jigging finer sizes; (6) sizing of washed coal for market; (7) settling tanks and filter presses for sludge. The original article gives a complete story of the various operations and description of the apparatus.

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For want of space the following contributions to the subject are referred to by title only. This list contains only articles which are not referred to in the text.

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[1] Westermann, *Gluckauf*, Vol. XLVII (1911), p. 1281.

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A REVIEW OF MINING DECISIONS IN 1911

By R. S. Morrison[1]

Mineral Waters.—The most interesting case decided in 1911, although of no materiality to western miners, and only important by analogy to certain legislative control of gas and oil wells, is the decision of the Federal Supreme Court in Lindsley *v.* National Carbonic Gas Co., 220 U. S. 61. The New York Legislature in 1908 had passed an Act prohibiting the pumping of the mineral waters of Sarotoga Springs under the pretense of preventing useless waste of water, one pumping plant drawing not only its own water but water from neighboring land. The New York courts (194 N. Y. 326) held void those provisions of the Act which went so far as to prohibit the pumping of water as clearly a deprivation of a man's property without compensation or process of law, but upheld those provisions which merely tended to prevent the wasting of the waters from the common source of supply. The Federal Court reviewed this case and came to the same conclusions.

Patent After Forfeiture.—When a patent was applied for in 1880 in the names of the then owners, and many years afterward their grantees perfected the application and received a patent in the name of such original applicants, it was held that the patent gave title not to the persons therein named, but to their grantees or heirs, as the case might be, and that a forfeiture of a co-tenant's interest for non-performance of annual labor enforced when the application was pending, made the forfeiting party one of such grantees. Van Sice *v.* Ibex M. Co., 183 Fed. 895.

The same case holds that there is nothing in the point that the advertising out by publication is not due process of law and that the policy of the law is so strongly against the non-contributing co-tenant that a Court of Equity will enforce the forfeiture, contrary to the general rule that a Court of Equity does not favor forfeitures.

Placers.—Zeiger *v.* Dowdy, 114 Pac. 565 (Ariz.), was a case between the owner of a placer claim and a town lot claimant. The Court held that there can be no valid possession of ground on the public domain, except where a party connects himself with some one of the various methods by which title is initiated, and that a squatter is a mere licensee

[1] Author of "Morrison's Mining Rights," Denver, Colo.; abstract from *Mining Science*, Jan. 11, 1912.

until he makes a location or claim of some sort. It also holds that a placer claimant must show a discovery upon which to initiate a location, and further, that while a purported relocation admits the validity of the original location, yet one who has made such relocation may disaffirm it and proceed de novo to procure a new title.

Murray *v.* White (Mont.), 113 Pac. 754, was a controversy between an agricultural claim and a placer. The bedrock lay deep, and it was in a mineral locality. The Court discussed the question as to what constitutes a valid discovery on a placer claim in such a situation. It is sufficient that gold is found in such quantity and under such conditions as would justify further prospecting in the expectation of finding pay.

Unequal Interests in Placer Claim.—Nome Co. *v.* Snyder, 187 Fed. 385, is a peculiar case and decides a point entirely novel, so far as I am informed. The facts were these: Five persons located a placer claim of 100 acres in their individual names. But contemporaneously they formed a joint stock company and distributed the shares unequally, two of the locators receiving only a nominal amount of stock, two more a little more or less than one-fifth and more than one-half going to the fifth locator.

The Court held that because no locator can take up more than 20 acres this was device by which one locator attempted to acquire more than 50 acres for himself and that the entire location was void.

So far as the case holds that the two who received only a nominal amount of stock were not bona fide locators, it is in accord with previous decisions, but that the law requires each locator not only to have a real interest but an interest exactly equal to each co-locator, it would seem to be a very debatable question.

Suppose that an outfitter agrees to supply a prospector with money necessary to place a couple of sluice boxes to work a surface placer on a small scale and a prospector agrees to furnish his entire time in labor, it is obvious that the prospector's time is worth much more than the money necessary to launch a scheme of that kind. Suppose on the other hand that the outfitter agrees to place a hydraulic plant and the prospector to furnish labor, certainly in such a case the outfitter is entitled to more than the prospector. We do not see how a fair although unequal division of a 40-acre location in either such case would contravene the law.

Known Lode in Placer.—Washoe Copper Co., *v.* Junila 115 Pac. 917 was a controversy between a placer patent and a known lode in which the Supreme Court of Montana reviewed at considerable length the evidence admissible in such cases and the rights of the respective parties, although they decide no material point not already passed on in other cases except

the point that a lode location certificate recorded before the issue of the placer patent, but void for want of compliance with the Montana statute, was no evidence whatever of the existence of the lode.

Injunction.—The lessors of a placer claim obtained an injunction against their own lessees and drove them to a lawsuit to recover possession. The Court most justly held that the lessors could not forfeit the lease for failure to keep its covenants when such failure was caused by the landlord's own acts; and further, that the lessees were entitled to an extension of time to enable them to sluice the gold which they had in sight. Halla *v.* Rogers, 187 Fed. 778.

Fraud.—Where a plaintiff bought an interest in a coal mine on false statements as to its output and capacity, he is entitled to a rescission; but a co-plaintiff who was a party to the fraud, or presumably at least cognizant of the real facts of the case, is not entitled to relief. One who makes representations on which another relies, which he does not know to be either false or true, is responsible if they turn out false; but if the buyer has ample facilities to ascertain whether true or false, he can have no relief if he fails to avail himself of such opportunities. King *v.* Lamborn, 186 Fed. 21.

Prospector Betrays His Outfitter.—In a New Mexico case 117 Pac. 833, Lockart *v.* Washington M. Co. the plaintiff had hired a prospector on a grubstake contract. The prospector found a valuable lode and began to perfect his location when the defendants appeared and bought off the prospector to allow his time to expire so that they could locate the lode, which they did.

The Court held that the defendants were guilty of fraud and conspiracy and gave a decree in favor of the outfitter. The report contains a full copy of the complaint upon which the plaintiff recovered and the form of the final decree.

Wilson Coal Co. *v.* U. S., 188 Fed. 545, holds that when coal land has been fraudulently entered and conveyed to a corporation whose incorporators were connected with the fraud, the patent may be set aside, although stock had been sold to stockholders who were ignorant of any attempt to defraud the Government.

Conveyance.—When the grantee in a mining deed of an undivided interest agrees to do certain development work in the mine it is not a condition upon breach of which the deed may be vacated, but a covenant, for breach of which damages may be claimed. Womble *v.* Womble (Cal.), 113 Pac. 353.

In Shaw *v.* Caldwell, 115 Pac. 941, the Supreme Court of California construes a peculiar deed in which the grantees of an undivided interest were permitted to work the mine, dividing the proceeds with the grantors.

The Court held that it was not a covenant to work, but only a permission and was not compulsory.

Annual Labor.—The work may be done on any one of a group of contiguous claims for the entire group, but work so done is not available for a claim so situated that it could not possibly tend to develop such claim. Morgan *v.* Myers (Cal.), 113 Pac. 153.

The holders of a lien or indirect interest of any kind may perform the annual labor on a mining claim to prevent forfeiture under a New Mexico statute to that effect. McVeigh *v.* Veig, 117 Pac. 857. And we apprehend that without the aid of any such statute the same could be done anywhere by any party having any interest to protect in the claim.

In Evalina M. Co. *v.* Yosemite M. Co., 115 Pac. 946, certain co-owners failed to contribute their share of the annual labor. They had parted with their interest before the forfeiture notice was served, but this fact was not known to the party that served the notice and the deed was not on record. The grantee in the deed was immediately informed of such demand or notice, but failed to redeem. The Court sustained the forfeiture. In the same case there was evidence that the assessment work had been done in full in the way of unwatering the mine to show an intending purchaser. But the parties who did the draining admitted that it was not done for purpose of assessment work. Upon such facts the Court held that it did not avail for the annual labor. Considered in connection with such admission it cannot be said that the Court held that draining a mine is not good to protect the requirement of annual labor.

Mineral in Railroad Lands.—A patent to the Central Pacific Railroad Co. does not cover lode claims located before the issue of the patent. While the patent is conclusive proof that the land was of the character allowed to the railroad company, it does not preclude proof that claimant comes within the exception contained in the patent. Van Ness *v.* Rooney, 116 Pac. 392 (Cal.).

But in Roberts *v.* S. Pac. R. Co., 186 Fed. 934, the grant to the Southern Pacific Railroad Co. received an entirely different construction.

Eminent Domain.—In a condemnation proceeding the owner is not to be compensated for mineral value where the alleged existence of the ore is entirely speculative; otherwise if valuable and accessible mineral is known to exist. Edwards *v.* City of Cheyenne, 114 Pac. 689.

Interstate Commerce.—The Oklahoma statute forbidding pipe lines to cross any state road for the purpose of carrying oil or gas beyond the state is an interference with interstate commerce and therefore void. Harkel *v.* Cowham, 187 Fed. 403.

Time.—The familiar clause recited in every well-drawn lease and

option: "Time being of the essence of this agreement," applies to every material provision of the agreement. Snider *v.* Yorbrough (Mont.), 115 Pac. 411.

Discovery.—Evidence that there were "seams of mineral" within a certain claim not followed by proof of what sort of mineral, or that it had any value, is not sufficient proof of discovery. Harper *v.* Hill, 113 Pac. 162.

Excess Width.—The Lookout mine was staked in 1889 as a claim 600 ft. in width. Years later, and at the date of trial, it was shown to be 323 ft. from the center line of the vein and the excess 23 ft. had been covered by a later location.

The Court held that the staking having been made in good faith, the center line being the center of the vein as it was then supposed to run, the error was not one which left the subsequently known excess feet open to location.

All previous decisions are to the contrary of this doctrine, notably Patterson *v.* Hitchcock, 3 Colo. 533, and McElligott *v.* Krogh, 151 Cal. 132. The latter case is cited in the opinion and a distinction attempted to be drawn, but we are unable to perceive that such a distinction exists and consider that the old construction is the better law. Harper *v.* Hill (Cal.), 113 Pac. 163.

Inspection Without Suit.—National Mines *v.* District Court upholds the Nevada statute allowing the Court upon proper showing to order a survey and inspection of a mine, although no suit has been brought to recover or concerning such mine. 116 Pac. 996. But the Colorado statute does not allow such survey or inspection without a supporting suit, as held in the Colorado case cited in the opinion.

Electricity.—Grants Pass Co., *v.* Enterprise Co. decides that electricity for power purposes is "supplies" to the mine under the mechanics' lien law of Oregon. 113 Pac. 859.

Sale of Knowledge.—In McCrimmon *v.* Murray, 117 Pac., 73 (Mont.), the defendant was alleged to have agreed to pay 10 per cent. of the selling price of a lead lode, which plantiff was to show him, provided such information would be "satisfactory" to the defendant. The case turned on the difference between "valuable" and "satisfactory," which words the opinion distinguishes and gives the rule for construing contracts where a party's judgment is to be convinced by the facts laid before him.

By far the most interesting case on the value of "Knowledge" as a legal consideration is that of Edelman *v.* Latshaw, 180 Pa. 419; 36 Atl. 926. This case was decided long before 1911.

The East Tintic Decision.—The above Summary of Decisions in 1911 made no reference to the East Tintic case because it was not intended

to cover rulings of the Land Department, which are not authoritative when cited before the judicial tribunals, and although respect is paid to them they are not considered to have been made by minds trained to the adjudication of legal principles.

This decision is reported at length in the *Engineering and Mining Journal* of June 15, 1912, p. 1167, with editorial comment on p. 1163.

It is difficult to say in terms that the two syllabi of the case are substantially wrong, but if we consider the language of the affidavit proving the discovery:

"That each and every one of said claims is based upon a discovery of mineral bearing rock in place," this language seems to show "a discovery of the vein or lode," which is the wording of the U. S. statute requiring a discovery as the basis of the location, and what is contained in the syllabi is to a great extent extraneous to the facts of the matter as reported.

Assays as high as $10 had been taken from the surface from the "small quartz seams," presumably found in the discovery shafts or cuts.

The decision seems to have been the result of an *ex parte* inquisition of the Land Department, impeding the hopes of the prospector and discouraging to the main idea of development, as if the policy of the Land Office were to deny the right of the miner to patent before he had a pay mine and losing to the goverment the purchase price of land worthless for timber or any purpose other than mining; and it fully justified the Editorial comment above referred to and is directly in conflict with the spirit of the decision in the leading case of *Erhardt vs. Boaro*, 113 U. S. 537.

CYANIDE

(By WILLIAM J. DWYER[1]).—A dull and monotonous year characterized the cyanide industry in 1911, but so far as ascertainable, the consumption of both the sodium and potassium salt was about equal to that of the previous year. The imports for the fiscal year were about 48½ tons less than those of the preceding year. A better inquiry was noted in the fall. While cyanides are manufactured in this country by several firms, the foreign markets are still the chief source of supply and the imports are generally regarded as a safe criterion of market conditions. The average value of the imports, weights and package, ex-vessel, port of shipment, in 1911, was 14 cents per pound; in 1910, 13½ cents per pound and in 1909, 14 cents per pound. The duty is 12½ per cent. The imports of potassium cyanide in short tons since 1903, the earliest figures obtainable, to date are shown in the accompanying table.

POTASSIUM CYANIDE IMPORTS.

Year	Tons.
1903	1,509.7
1904	756.9
1905	812.1
1906	1,054.3
1907	1,535.1
1908	1,664.1
1909	1,376.4
1910	1,211.3
1911	1,163.6

Statistics covering the imports of sodium cyanide are not tabulated as most of this salt is sold as high-percentage potassium cyanide. Of the 1911 imports of potassium cyanide 1087.9 short tons valued at $304,359, or about 93 per cent. of the total imports, were brought in through the port of New York. The trade is in the hands of two firms representing German and English interests. One of the importers is likewise the largest domestic producer. Prices of both sodium and potassium salts remained unchanged throughout the year at 18 cents per pound in carload lots.

Bromo-cyanide.—Hans Foersterling of the Roessler & Hasslacher Chemical Co., the largest domestic producer of cyanides, was granted patent No. 999,215 on Aug. 1, covering a composition of matter which will easily generate halogen cyanides in connection with an oxidizing agent. The process calls for the melting of molecular proportions of sodium cyanide and sodium bromide, which are allowed to solidify. This

[1]*E. and M. J.*, Jan. 13, 1912.

double salt is used instead of the mixture heretofore used consisting of sulphuric acid, sodium bromide, sodium bromate and cyanides. As an oxidizing agent a 25 per cent. hydrogen-peroxide acid solution is added. Bromocyanide is rapidly formed and the bromine is taken up by the hydrocyanic acid. It is proposed to furnish the mines with a ready-made, correctly proportioned mixture of alkali halogen with cyanide, "all that is necessary being to dissolve a certain amount of double salt and add an oxidizing agent." The inventor claims that the largest part of the alkali halogen is not lost in general mill work and it is only necessary to add so much alkali halogen to the cyanide as has been actually lost.

The discussion as to purity of commercial cyanide arose from the fact that pure potassium cyanide does not contain so much cyanogen as pure sodium cyanide. At some of the largest mines much careful analyzing has been done on the product as received from the dealer, and as it is generally conceded that the cyanogen is the valuable constituent as far as solution for gold is concerned, the mill men objected to a mixture of sodium and potassium salts adulterating the product; further there was discussion of the question as to whether the consumer is entitled to know that he is receiving potassium or sodium cyanide, and, if a mixture, in what proportion. Large consumers are in a position to insist on the proper specification of their cyanide. Where a group of consumers can maintain a common laboratory as in some mines in South Africa, this object could easily be accomplished. The percentage of cyanogen in chemically pure potassium cyanide is 39.95.

A method of determining potassium in sodium cyanide is given by Clennell.[1]

The usual silver nitrate titration, with potassium iodide indicator in one case gave cyanide strength higher than the truth on account of the presence of hypo (thiosulphate) as an impurity. This difficulty is overcome by titrating with iodine, after neutralizing with acid till methyl orange added has a pink tint and which just remains when the iodine color has disappeared.

The wide range of commercial product has been known for a number of years. New manufacturing methods and sources of supply have been encouraged by the increased demand for cyanide, so that one-half the price with the product containing two or three times the cyanogen has resulted. The commercial product was at one time only 15 per cent. cyanogen.

Pure sodium cyanide contains 53.05 per cent. The absurdity of having 125 per cent. sodium cyanide, or even a higher percentage as advertised by the dealers (highest possible 132.8), would be avoided by

[1] *E. and M. J.*, June 25, 1910.

stating the cyanogen. Commercial cyanide is not strictly pure potassium cyanide but always a mixture with more or less carbonates, chlorides and cyanates, and also smaller amounts of sulphites, sulphates, thiocyanates, ferrocyanides, formates, etc. These come from the process or the original raw materials. In most methods of making cyanide, a ferrocyanide is produced as an intermediate product. This is treated with sodium carbonate or, in more modern practice with metallic sodium. With the former method a certain amount of cyanate is formed, while with sodium a mixture of the cyanides of potassium and sodium results. The use of sodium compounds in either case is due to there not being an abundance of potassium compounds. There is also more difficulty in preparing the metallic potassium. The sodium ferrocyanide crystallizes with more molecules of water (12), whereas the one used has only three molecules of water. The carbonate impurity in commercial cyanide may act as a protective alkali to neutralize cyanicides. The small amounts of sulphides usually do no harm as they are generally oxidized or precipitated as zinc sulphide. Large amounts could not be tolerated. The ammonium compounds frequently given off may carry cyanogen, especially after being in solution and reacting to form ammonium cyanide. Ammonium cyanide is volatilized and partly decomposed at 36° C.

It is believed that under a selling agreement the dealers in cyanide did substitute sodium cyanide mixed in varying quantities in the commercial potassium cyanide sold. The United States appraisers were the first to determine this by actual assay, on account of the difference in duty on sodium and potassium cyanides.

It is estimated that one-quarter of the output of gold in the world is produced by cyanidation. Wm. Neill[1] estimates that research on improvements for cyanide manufacture and for factories exceeds $8,000,000 since 1891, when the price of cyanide was 48 cents a pound. The price continued to fall, until in 1903 it reached 14 cents, several producers going under. The 20 per cent. cyanide made on the continent is not a fused product but is obtained by wet process and finished in small briquetts.

The briquetted cyanide derives its nitrogen from the gases given off by the residue of sugar produced in the beet sugar refinery. The heating of this residue gives off ammonia and hydrocyanic gases, collected respectively by sulphuric acid and caustic soda. The 98 per cent. and 130 per cent. products are fused on the same plan as Castner's patents. In Great Britain and elsewhere much fused cyanide is produced from ferrocyanide and metallic sodium.

[1] *E. and M. J.*, May 6, 1911.

The cyanide manufacture in the United States is now carried out by only a few firms. The process used is probably that which was patented a number of years ago. This consists in melting in an iron pot metallic sodium, the temperature being in the neighborhood of 400° C. Dry ammonia gas is then passed through the metal thus converting it into sodamide at the same time liberating hydrogen. The reaction is as follows:

$$2NH_3+Na_2=2NaNH_2+H_2$$

Pure charcoal which has been heated to a definite high temperature is then dropped into the molten mixture. The resulting reaction is the formation of sodium cyanide and again the evolution of hydrogen. It is as follows:

$$NaNH_2+C=NaCN+H_2$$

The source of the nitrogen in the production of cyanides was formerly from organic matter that was mixed to produce the ferrocyanide. It has been suggested that the nitrogen compounds obtained in distillation of coal or oil could be utilized for this purpose. The fixation of nitrogen (see p. 554) has opened a wide field, although originally designed for the making of fertilizer.

One of the chief sources of nitrogen for various purposes is calcium cyanamide which was produced during the year by the American Cyanamide Co. at its plant in Niagara Falls, Ont., just across the Canadian border. The use of the product, almost exclusively for fertilizer purposes, revealed a marked increase. The consumption of calcium cyanamide in the United States up to the end of the fiscal year was 5617 short tons valued at $295,545 compared with 764 tons valued at $40,091 consumed from Aug. 6, 1909 to June 30, 1910. This is mentioned on account of the perfecting of the Clancy process of treating gold ores and promises a new outlet for the product.

The cyanides ordinarily used are potassium and sodium salts, KCN and NaCN. In recent years, however, there has been a development in the cheap production of cyanides of the alkaline earths, such as calcium cyanide, $Ca(CN)_2$, and barium cyanide, $Ba(CN)_2$, by means of the electric furnace. These electric-furnace products, even impure as they are, if they contain 80 to 90 per cent. of the theoretical quantity of cyanogen, are available for and may be used in ammonia solutions as gold and silver solvents. D. Mosher has described its application in U. S. Pat. 911,254, 1909.

Halogen Cyanide.—The use of cyanamide with electrolysis is a success on some ores such as tellurides. The Moore-Clancy process as used at the Colburn Ajax mill at Victor, Colorado, and installed in June, 1911, consists in generating in the solution of cyanamide to which was added crude iodine,

the cyanogen iodide by means of a low voltage current. As this halogen cyanogen compound is not stable in strong alkali solution, the first stage of the process is started in a nearly neutral solution. The prolonged electrolysis then regenerates the used-up cyanides by the reaction between cyanamide, sulphocyanides and cyanates. The alkalinity of this stage is then neutralized with the next ore. The cyanamide is about 56 per cent. soluble in water, the clear solution only being used. D. Mosher states the following:[1]

The theoretical per cent. of nitrogen in calcium cyanamide, Ca(CN)N, is 34.22, while the theoretical per cent. of nitrogen in calcium cyanide, $Ca(CN)_2$, is 31.5. In calcium cyanamide, Ca(CN)N, there is one atom of nitrogen uncombined with carbon and one molecule of cyanogen. The atom of nitrogen uncombined with carbon forms, on slaking the product in water, NH_2 or amide, of no known value as a gold and silver solvent. If this nitrogen atom, however, be united with carbon, which is readily feasible, another molecule of cyanogen is gained. That is, calcium cyanamide is only of half the value in gold and silver extraction of that of calcium cyanide, due to double the quantity of cyanogen $(CN)_2$ existing in the latter.

A typical analysis of commerical calcium cyanamide is as follows: $CaCN_2$, 57 per cent.; C, 14; CaO, 21; SiO_2, 2.5; Fe_2O_3, 4; S, P and $CaCO_3$, 1.5 per cent. The percentage of nitrogen is 19.5. Only half of this is present as cyanogen, of value in gold and silver extraction, and is equivalent to 9.75 per cent. available nitrogen, and as this corresponds to 19 per cent. cyanogen, at 3 cents per pound, calcium cyanamide is by no means a cheap cyanogen product. On an average of a 15 per cent. basis of cyanogen contents, calcium cyanamide, in comparison with sodium cyanide, works out as follows: One pound sodium cyanide, 48 to 52 per cent. cyanogen, is equivalent to 3 1/3 lb. calcium cyanamide of 15 per cent. cyanogen. At 3 cents per pound 3 1/3 lb. calcium cyanamide cost 10 cents, additional freight on 2 1/3 lb., probably 2 cents, total 12 cents per pound, as against 19 cents for sodium cyanide. With this comparison and the possibility of cheaper sodium cyanide, cyanamide offers little inducement for the cyanide operator, when the impurities in the calcium cyanamide are taken into account.

Calcium cyanide has an available nitrogen content of 19.5 per cent., equivalent to 36.6 per cent. cyanogen and is produced at a cost of 4 cents per pound.

The following are the cyanogen percentages (on a chemically pure basis) in the various cyanide salts: sodium, 53.07; potassium, 39.94; calcium, 56.47; barium, 32.06; and sodium sulphocyanide, 32.07. The

[1] *E. and M. J.*, April 1, 1911.

Badische Anilin and Soda Fabrick manufactures barium cyanide by passing unsaturated hydrocarbons over the cyanamides at a temperature below 1200° C.

A description is given of the possibilities of electrolytic cyanide regeneration by J. E. Clennell.[1] From the tests made, the conclusions arrived at were, that under ordinary circumstances, the regeneration is imperfect and, in many cases, may be reversed. This is especially true in silver cyanidation. These conclusions apply to tests on refractory gold-silver ore from 90 to 95 per cent. gold and 30 to 50 per cent. silver. See page 351.

Cyanide Poisoning.—The best known antidote for cyanide poisoning is an alkaline solution of ferrous sulphate. If the antidote is not administered within 5 minutes of the fatal dose of poison, there is no hope for the patient. When this antidote is kept in the most convenient form, it takes 2 to 3 minutes to prepare it for administration so that the chances of recovery for the patient would be very slim. J. L. Jond of Melbourne, Australia, experimented with andrenalin to see whether it would delay the work of the poison so as to give time for the application of the usual measures and antidotes. He found that if given within 5 minutes—12 c.c. of the 1:1000 solution—it helped the matter materially. See page 354.

Potassium cyanide of itself is not believed to be poisonous, but the hydrocyanic acid liberated from it by acids of the stomach, or hydrocyanic or prussic acid acting on the blood, are very poisonous. Wounds have been freely exposed to alkaline solutions of cyanide without fatal effects.

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H. SCHEIDEL. (*Bull.* 5, p. 30, Cal. State Mining Bureau, 1894.)

[1] *Eng. and Min. Journ.*, May 27, 1911.

SAMPLING AND ASSAYING IN 1911

By Donald M. Liddell

In reviewing the literature of sampling and assaying, or any kindred line, one is constantly struck with the amount of vain repetition which exists, and is soon led to the belief that if many writers spent half the time on reading that is spent on working out "new" processes, that the general analytical standard would be higher. However, this repetition is, after all, not without its uses, for the young chemist is thereby enabled to do better without a large library than he could otherwise hope to. And so, although there will be much repetition in this article, it is perhaps somewhat pardonable, in that it will at least give later and perhaps for that reason more accessible references than would otherwise be available.

That there were not many remarkable articles published on sampling in 1911 is scarcely strange, since the subject of ore sampling seems to have been pretty thoroughly exhausted, both from the theoretical and practical sides by Mandy's, Little's, and Wright's work, taken up at some length last year in the Mineral Industry.[1]

Cobalt Ore Sampling.—On this subject there was an unsigned description appeared of the Campbell-Deyell mill at Cobalt.[2] The ore is first crushed to pass a 1-in. ring. It is then put through a ball mill, giving an 8-mesh product. From here the screened metallics go to a bullion furnace, the slag going with the next shipment of ore. The crushed and screened ore then goes to a quartering machine, where each quarter is treated as a separate sample and each reduced three times in 1/20 cuts. Moisture samples are taken on the last cut. The ore is then bagged and weighed and is ready for shipment. The 1/20 cuts are taken by twos on Vezin samplers, giving 1/10 samples, the feed to the Vezins being by shaking plates.

A controversy over cobalt ore sampling was carried on by F. W. Pugsley, who described the ridging method of sampling[3] and A. A. Nason,[4] who described some of the difficulties inherent in the process. Mandy in some of his earlier papers seems to sum the matter up best: that the hand sampler is good only insofar as he becomes mechanical, hence he believes mechanical sampling preferable. Those who must sample by hand are, however, recommended to read these references, as following Pugsley's

[1] Mineral Industry, Vol. XIX, p. 792.
[2] *Canadian Mining Journal*, Oct. 1, 1911.
[3] *Eng. and Min. Journ.*, April 15, 1911; July 1, 1911.
[4] *Eng. and Min. Journ.*, June 10, 1911; July 8, 1911.

directions, and avoiding the errors pointed out by Nason, should certainly give good results.

A device[1] for sampling screenings is shown in the accompanying figure 1. To construct the sampler, take a piece of ordinary 4-in. galvanized pipe, as used for ventilating mines and cut as shown in the illustration. Place this diagonally across and about 6 in. beneath the screen. The square holes in the pipe should be cut a little larger than the mesh of the screen used. If a 1/2-in. screen in used, the holes should be cut about 1 in. square and proportionately. Allow the end of the pipe to

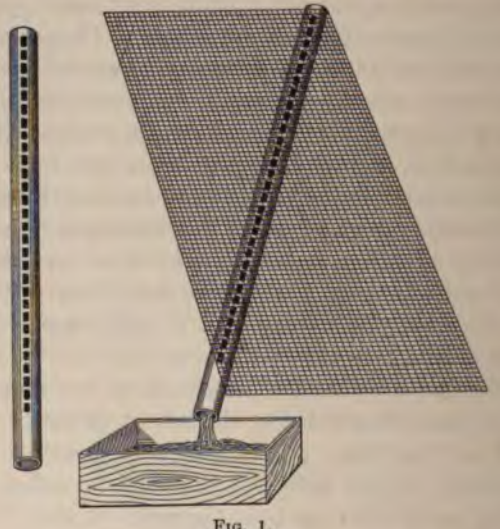

FIG. 1.

extend about 18 in. from the side of the screen and place a box underneath the spout to receive the sample. This may be emptied at the end of each shift, and a fairly accurate sample of the day's run will be obtained. From the theoretical standpoint the apparatus would seem liable to take a little more of the finer portion than is just, for the reason that there would be a greater portion of coarse strike part in and part out, and bound from the edges.

Mine Sampling.—An interesting study of mine sampling is given by A. J. Sale.[2] His method is to plot his assays on section paper, using the assays as ordinates and distances as abscissas, using the same scale as used on the mine maps. His claim is that it can be deduced from such maps whether the work in progress is really a true drift or is crosscutting the vein. He also gives the application of graphic methods to the

[1] *Min. and Sci. Press*, Nov. 4, 1911.
[2] *Mines and Minerals*, June, 1911.

study of churn-drilling results, and to the study of the problem of where the maximum efficiency of concentration is found.

J. A. Herdy[1] calls attention to the desirability of a multiplicity of samples of an ore body cut by a tunnel, say one cut for each 5 ft., so that one may judge whether the body is workable as a whole, or only in small high-grade shoots.

Walter R. Hodge shows a method for sampling old dumps by means of a set of 4×6-in. timber, about 1-in. inside measure, spiles of 4-in. diameter being driven down around it as the hole deepens.[2] When the first set is only about 1 1/2 in. above the collar, a new frame is driven in place at the bottom of the pit, and the driving of another set of spiles commenced.

Glenn Anderson[3] describes the tools and the method of sampling deposits exposed in underground workings, as well as some of the things to be guarded against for fear of salting. Some of the good ideas recommended are: moils with handles to avoid danger of the moil holder being hit, and a sample collector made of heavy iron wire with a canvas funnel suspended from it. In referring to the original, note the errors spoken of on p. 1143, *Engineering and Mining Journ.*, June 10, 1911.

Grab Sampling in Mines.—The question of grab sampling in mines is taken up by Claude T. Rice,[4] who speaks of the difficulty of getting accurate groove samples, when the ore has hard and soft streaks, and instances the use of grab samples for settlement at the Calumet & Arizona smeltery, and draws the conclusion that under some conditions, a well chosen grab sample is about as good as one with pretensions to accuracy.

Placer Ground Sampling.—In the sampling of placer ground, T. Lane Carter believes that nothing is equal to test pits, in case the bedrock lies not over 12 to 15 ft. below the surface.[5] The residues from all the test pans are dried and combined and the black sand blown away, then the gold is collected and weighed. Charles Janin is of the impression that this last is not good practice, and that amalgamation of the black sands is the only correct method.[6] He also calls attention to the fact that it is the nature of the ground, and not the depth which determines how deep the pits may be sunk. He says in his own work he has put down pits 35 ft. He also calls attention to the folly of combining samples from different pits. R. F. Lafferty[7] also takes up this matter, in full agreement with Janin. Lafferty also recommends,[8] the use of the rotary

[1] *Eng. and Min. Journ.*, April 22, 1911.
[2] *Eng. and Min. Journ.*, Nov. 25, 1912.
[3] *Eng. and Min. Journ.*, March 4, 1911.
[4] *Eng. and Min. Journ.*, Oct. 14, 1911.
[5] *Eng. and Min. Journ.*, Mar. 25, 1911.
[6] *Eng. and Min. Journ.*, April 11, 1911.
[7] *Eng. and Min. Journ.*, April 29, 1911.
[8] *Eng. and Min. Journ.*, Nov. 25, 1912.

hand drill, and thinks the churn drill entirely inaccurate. What seems to be a good suggestion for sampling placer ground is that of J. Edgerton Wood,[1] who advises carrying small squares of test lead into the field. A sample of the ground is weighed and panned, and the gold washed into a sheet of the foil, which has its corners turned up to make a small basin. The foil is then folded, a number scratched on it, and in that shape it can be carried until an assay office is reached.

Assaying Diamond Drill Samples.—In assaying the samples obtained by diamond drilling, W. J. Mead[2] gives as the various methods the following: (1) To assay core and cuttings as one large sample. This is correct only if all the cuttings are recovered. (2) Assay cuttings alone. As hard layers usually give more core, and soft more cuttings, this it incorrect. (3) Assay core alone. Also incorrect. (4) Assay cuttings and core separately and take a numerical average of the results. (5) To calculate the total volume of material cut out by the drill, and the amount recovered as core, then the difference is cuttings and a weighted average can be taken between the assays of core and cuttings samples. The only error here, outside of those of measurement, seems to be the assumption of equal specific gravities for the hard and soft portions, which is, however, a small one. The author gives a diagram, by means of which, knowing the size of bit used, if of Longyear & Hodge's manufacture, and the inches of core recovered per 5 ft. of drill hole, the proportions of core and of cuttings can be read off directly.

Of interest in connection with drilling is Guy C. Stoltz's description[3] of his aziclinometer. Needles for reading the points of the compass and dip are so arranged in this instrument, that they will be clamped by the release of springs through the melting of candy, or salt, or other soluble substance. In a magnetic formation the instrument is useless, but in non-magnetic rocks there is no difficulty.

Concerning drill-exploration work, E. E. White gives elaborate directions concerning taking and saving sludge samples as practised on the Michigan iron ranges.[4] The chief points seem to be careful flushing out of the holes before taking a sludge sample; care in stopping one sample and beginning another whenever the drill runs in or out of ore; and taking samples of sludge for at least 20 ft. after running out of ore, in order to see whether or not the ore is caving. Albert E. Hall advises duplicate sludge boxes, so that one can be used while the deposit is settling in the other,[5] and the use of a small filter bag for filtering the samples.

[1] *Bull.* 73, Inst. Min. and Met.
[2] *Eng. and Min. Journ.*, May 6, 1911.
[3] *Eng. and Min. Journ.*, Mar. 25, 1911.
[4] *Eng. and Min. Journ.*, Oct. 7, 1912.
[5] *Eng. and Min. Journ.*, Nov. 18, 1912.

Copper Bullion Sampling.—Edward Keller discusses the paper of William Wraith.[1] Mr. Wraith had concluded that shot sampling by batting the running stream of metal was the best method of sampling copper bullion. Dr. Keller shows that there can be no question that the template gives results equally reliable.[2] Of interest in showing the accuracy of the work in Dr. Keller's own laboratory is a table showing 40 samples assayed by two men independently. The bullion worked on was about 80-oz., and the maximum variation was 0.85 oz., the second largest 0.44 oz. per ton, which seems to me a striking answer to the critics of commercial assay methods. I discussed the errors of the weight-volume ratio of drill holes on the bevel of copper bars, recommending a thin bar with a small bevel.[3] Dr. Keller points out that the ordinary geometrical weight-volume ratio is subsidiary to a weight-volume ratio determined by a segregation-geometry, and again urges his one-hole templet method. As stated in my later reply to him, while his method is correct theoretically, practically it seems impossible of adoption, owing to the chance of intentional dishonesty by the shippers.

Gas Analysis.—The U. S. Steel Co. published, through J. M. Camp, an exhaustive study of the methods of gas analysis and sample collecting. While there appears to be nothing new, the papers[4] form a good treatise on the present-day approved practice, and as he says "the delineation of the methods is in considerable detail, so as to be intelligible to amateurs, as well as to those better versed in the subject." The paper is a mine of information for those not specially versed in the subject, and should be read in full.

Contamination of Samples by Grinding Machinery.—George A. James calls attention to the contamination of samples by iron from the grinding machinery.[5] In reducing ore to 80-mesh in a disk grinder, he finds 0.25 to 3.0 per cent. of iron in the sample. This results in many errors. In panning samples a heavy black substance is often found which results immediately in the chemist's being set to work on the search for iridium, etc. In smeltery work, of course this results in erroneously mixed furnace charges, but it also results in reducing the valuable metals by an appreciable percentage. Evolution methods for CO_2, etc., are also thrown out by the metallic iron.

A small wooden filter press for dewatering slimes samples is described by C. C. Semple.[6] The device is admirable. The same is true of the sam-

[1] MINERAL INDUSTRY, Vol. XIX, p. 797. *Eng. and Min. Journ.*, March. 26, 1910.
[2] *Bull.* A. I. M. E., June, 1911, p. 500.
[3] *Eng. and Min. Journ.*, Dec. 16, 1911
[4] *Met. and Chem. Eng.*, June, 1911; July, 1911.
[5] Trans. Am. Metal Soc., May, 1911.
[6] *Eng. and Min. Journ.*, Feb. 25, 1911.

pling apparatus for sands and slimes described by W. O. Borcherdt.[1] This device derives its power from the flow of the pulp through it.

At the Tonopah-Belmont mill, boxes with false bottoms (10-oz. canvas over a coarse wire screen), are connected with a vacuum pump to dewater slime samples.[2]

Assaying and Analysis

Portable Assay Outfit.—The question of assaying in the field is gone into by S. K. Bradford.[3] His outfit can be carried in a 26-in. valise. It consists of a small hammer; a steel mortar, 2.5 in. wide and 3 in. high with stand sawed off flat, so that it can be used for an anvil; small steel pestle; blow-pipe and blow-pipe cupel holders; small and large spatula; button pliers; light 1-in. cupel mould; two 3.5×8-in. muffles; 18 "*F*" crucibles, and six covers; short crucible tongs; 25-c.c. burette; six 1/2-oz. Denver parting bottles; six 1/2 oz. Berlin annealing cups; two or three beakers; hand balance such as is used for weighing gold dust, and set of gram weights; a 6-in., 60-mesh screen; six small glass stirring rods; tea spoon and table spoon; and a micrometer for measuring the diameter of buttons. This may be a short fixed microscope stand, and several extra eye pieces. The reagents are: 1/2 lb. potassium cyanide; 1 lb. c.p. nitric acid; 1 lb. c.p. hydrochloric acid; 2 lb. aqua ammonia; these should be in bottles enclosed in metallic cases. Other reagents are: a few ounces of sheet copper; 1 oz. c.p. sheet silver; 2 lb. ten-penny nails; 5 lb. bone ash; 5 lb. sodium bicarbonate; 5 lb. litharge; a few pieces of blow-pipe charcoal; a half dozen candles and some old newspapers. The author calls attention to some substitutions which can be made on a long trip if supplies run out, such as red lead for litharge, broken beer bottles for beakers, etc.

Assay Apparatus.—Arthur O. Christensen describes rough-and-ready assaying in a blacksmith's forge. The results, while not accurate, will at least tell whether or not a prospector has anything.[4] C. C. Semple discusses one of Mr. Christensen's points, calling attention to the fact that direct as well as radiated heat plays an important part in such work,[5] and giving some good advice on making rude furnaces.

A suggestion by Norman Roberts and F. A. McDermott, for more easily heating crucible furnaces and thus obtaining higher temperature,[6] is to replace the greater part of the usual solid clay or earthenware wall

[1] *Eng. and Min. Journ.*, March 4, 1911.
[2] *Eng. and Min. Journ.*, Nov. 11, 1911.
[3] *Bull.* A. I. M. E., Jan., 1911.
[4] *Eng. and Min. Journ.*, Dec. 23, 1911.
[5] *Eng. and Min. Journ.*, Dec. 30, 1911.
[6] *Journ. Am. Chem. Soc.*, p. 507, 1911.

by an equal or greater thickness of a non-conducting refractory powder such as light magnesia, kaolin, or even sifted ashes. The construction is especially indicated for small furnaces.

The Gold Assay.—That an excess of borax in an assay gives decidedly low results is urged by Keyes and Riddell.[1]

Louis D. Huntoon took a sample which was being prepared for the assayer,[2] and screened it through various sizes of screens. The differences in assays on the various portions was of course great, as is to be expected, since this would be a function of the differences in friability between the mineral and the gangue. He points out the great care necessary in mixing samples after each screening, and also says that, as a rule, pulverization below 60-mesh, while an aid to furnace work, is unnecessary for accuracy. That is, most samplers are putting in an excess of work on the wrong end of the crushing process.

For the niter assay, P. D. Wilson[3] advises the use of a plotted diagram, from which, having determined one's niter oxidation-equivalent, one can immediately read off the niter required per A. T. of ore, as soon as the sulphur percentage of the ore, or size of the preliminary button is known.

Cupels.—An investigation of the effect of various sorts of cupels was undertaken by Benner and Hartmann.[4] Their results were that the cupellation losses in various cupels are: Morganite, 1.99%; bone ash, 2.36; equal parts of cement and bone ash, 2.95; brownite, 2.89; casseite, 3.09; cement, 3.38. The losses look as though the cupels had been run hotter than in commercial work, and as the purity of the residual silver bead seems to have been tested in no case, I question whether these results are worth very much. They also report that the size of the bone ash used is without effect on the degree of absorption. In my own work it appeared that between 40- and 50-mesh the absorption became excessive; and, while this is the negative of absorption, above 120-mesh, I often had cupels which refused to cupel; while between 100- and 120-mesh, the cupellation had to be carried on so hot that volatilization became excessive.

C. Crichton, before the Chemical, Metallurgical, and Mining Society of South Africa, recommends an electrolytic cabinet for the assay of cyanide solutions.[5] The anodes are carbon, and it is important that they should not be in contact with the solution when no current is passing through, in order to avoid deposition of the gold on them. The cathodes are lead foil. After deposition is complete, the cathode is

[1] Wilkes-Barre meeting, A. I. M. E.
[2] *Eng. and Min. Journ.*, June 24, 1911.
[3] *Eng. and Min. Journ.*, Dec. 30, 1911.
[4] *Journ. Ind. and Eng. Chem.*, Nov., 1911.
[5] *Eng. and Min. Journ.*, Nov. 18, 1912.

dried, cupelled with a little silver, and the button parted and weighed.

In testing samples of ore for completeness of cyanide extraction, a cyanicide should be used immediately upon removal of the sample, otherwise extraction will continue. H. A. White recommends[1] the use of an excess of alkaline permanganate, or an alkaline solution of bleaching powder. Addition of potassium permanganate to permanent pink color is the easiest method.

Copper.—A new, sharp qualitative reaction for copper is that of 1,2-diamidoanthrochinon-3-sulphonic acid in dilute alkalis, which takes on an intense blue color with copper.[2] The color is easily noticeable with 0.0000019 grams of copper in 1 c.c. of solution. Another organic reagent finding application in inorganic analysis is "cupferron," nitrosophenylhydroxylamine, $C_6H_5(NO)ONH_4$, as a precipitant for copper and iron ions.[3] The precipitate settles rapidly, washes easily, and is quantitative in strong acid solutions. The copper is easily soluble in NH_4OH, and the ferric salt stays on the filter, while the iron salt is readily soluble in ether, chloroform, acetone, and similar solvents. Directions are given for preparation of the reagent.

A new kink in the iodide assay is produced by C. E. Kendall,[4] in which the nitrous-acid fumes caused by dissolving the substance in nitric acid, are destroyed by addition of sodium hypochlorite. The volume of the solution should be 50 to 60 c.c., and the acidity equal to 4 or 5 c.c. of nitric acid. The temperature should not be over 77° F. About 5 c.c. of hypochlorite solution formed by boiling 112 grm. of calcium hypochlorite, and 100 grm. of sodium carbonate in 1200 c.c. of water. As soon as the color changes from a clear blue to greenish, enough hypochlorite has been added. Allow to stand about 2 minutes and add 10 c.c. of a 5% solution of phenol to destroy any free chlorine. Add the phenol quickly by blowing through a pipette from which the tip has been removed. Blow out the chlorine gas. Wash down the sides of the flask containing the solution and add sodium hydroxide until a slight precipitate forms. Clear the solution with a few cubic centimeters of acetic acid, and titrate as usual.

W. N. Iwanow determines copper in pyrites or blue billy by heating the sample with an equal portion of powdered iron (*ferrum hydrogenio reductum*), dissolving the result in hydrochloric acid, and precipitating the copper with H_2S.[5]

Tin.—J. A. Sanches gives a method for determining small amounts

[1] Chem. Met. and Min. Soc. of S. A., Sept. Meeting, 1911.
[2] *Chem. Engineer*, Jan., 1911.
[3] *Journ. Ind. and Eng. Chem.*, *Sept.*, 1911.
[4] *Journ.* Am. Chem. Soc., Dec. 1911, p. 1947
[5] *Chem. Zeit.*, May 18, 1911; *Eng. and Min. Journ.*, June 17, 1911.

of tin in the presence of antimony by titration with ferric chloride.[1] This oxidizes stannous chloride, and does not affect antimonious chloride. If the metals have been already separated as sulphides they may be dissolved in hydrochloric acid and potassium chlorate, and after all free chlorine is expelled, the antimony thrown down by aluminum wire, the tin being reduced only to stannous chloride. Other articles on tin-ore and tin-alloy analysis, which are too involved or too specialized for brief abstracting, are noted in the bibliography at the end of this article.

Nickel.—A cyanometric method of determining nickel in nickel steel was worked out by G. Raulin,[2] but as it involves the determination of constant correcting factors, which must be determined for each grade of steel, its republication seems out of place in a general review, although it seems worth while for an iron chemist.

Manganese.—A method for manganese, unaffected by iron or nitric acid is given by Metzger and Marks,[3] which depends on the fact that in the presence of sufficient hydrofluoric acid, "ous" manganese and permanganic acid react quantitatively to give "ic" manganese. Titration is performed in beakers made by cutting the tops off ceresine bottles, and the application to the anaylsis of steel is thus described.

Weigh out about 1 grm. of sample; dissolve in 10 c.c. of nitric acid (1 : 1) in a covered casserole. Cool slightly; add 1 gram of ammonium persulphate and let stand until effervescence ceases; boil briskly for a few seconds, remove cover and evaporate to dryness (do not bake); take up with 20 c.c. of sulphuric acid (1 : 2) and 30 c.c. of water and boil until the solution is clear. Cool, transfer to a wax beaker, add 5 grams of ammonium fluoride, 25 c.c. hydrofluoric acid, dilute with water to a volume of 100-150 c.c. and titrate. The solution is colorless and the pink endpoint is very easily seen. As much as 2 c.c. of concentrated nitric acid has no effect on the titration; hence, if the residue has become baked add 1 or 2 c.c. of dilute nitric acid to make complete solution take place readily.

The solutions used are a solution of $KMnO_4$, approximately $N/30$; standarized against specially prepared Mohr's salt; 1 c.c. = 0.001721 grm. Fe or 0.001354 grm. Mn; and a solution of $MnSO_4$, standardized by precipitation as manganous ammonium phosphate, and weighed as $Mn_2P_2O_7$ after ignition; 1 c.c. = 0.002727 grm. of manganese.

A modification of the Volhard-Wolff method for manganese in iron or steel consists in dissolving 1.0 grm. of the iron in 25 c.c. of HCl, sp. gr. 1.12, and adding compressed chlorate of potash lozenges (1grm. each),

[1] *Eng. and Min. Journ.*, June 3, 1911.
[2] *Monit. Scien.*, 1911, p. 84–86. *Cf. Journ. Soc. Chem. Ind.*, Feb. 28, 1911
[3] *Journ. Ind. and Eng. Chem.*, May, 1911.

and boiling until all the carbon is gone, boil until the free chlorine has disappeared, neutralize with zinc oxide, avoiding an excess, and titrate with permanganate.[1] While one lozenge is enough with steel, even six may be necessary with pig irons. A determination may be made in 15 to 20 minutes.

Analysis for the Rare Metals.—W. B. Hicks tried some experiments on the use of sulphur monochloride in the analysis of the rare-earth metals.[2] The substance to be analyzed was placed in a porcelain boat in a glass tube and heated while sulphur monochloride was passed over the sample. The acid-metal oxides pass over with the sulphur monochloride, the basic-metal oxides remain behind. As can be seen, columbium, titanium, tantalum, etc., occurring as acids can readily be separated in this way from any bases whatever.

A. L. Sweetser gives the following scheme for analysis for some of the rare earths and rare-earth metals.[3] Cerium, didymium, lanthanum and thorium are precipitated with alumina by ammonium hydrate. The precipitate is dissolved in hydrochloric acid, this solution oxidized with chlorine water, and oxalic acid added. Cerium, didymium and lanthanum are precipitated as oxalates, which on ignition, are converted into oxides. These are soluble in acids. Their solution in hydrochloric acid is nearly neutralized; acetate of soda is then added, and an excess of sodium hypochlorite. On boiling, the cerium is precipitated as a dioxide, which is filtered off, ignited and weighed.

Didymium and lanthanum are determined by converting into oxalates, then into oxides and dissolving in dilute nitric acid. This solution is then evaporated to dryness and ignited, for a few minutes, just below redness. A subnitrate of didymium is formed, and remains as an insoluble residue on the addition of hot water. Lanthanum is now in the solution, which is evaporated to dryness. Both salts are treated with ammonia and ignited and weighed as oxides. (Di_2O_3 and La_2O_3.)

Thorium is separated from alumina and other earths by dissolving it in hydrochloric acid, neutralizing and boiling with sodium hyposulphite. The thorium will be in the precipitate. It is dissolved with water, and an excess of ammonium oxalate is added and heated.

The precipitate will be thorium oxalate, which is washed with hot water, dried and ignited. It is then weighed as thoria (ThO_2). In this condition it is not readily soluble in acids. (It is not clear to me that there will be a good separation if Ce, Di, La, and Th come in one sample, if one works by this scheme.—D. M. L.)

[1] *Chem. Zeit.*, 1910, p. 1225.
[2] *Journ. Am. Chem. Soc.*, Sept., 1911.
[3] *Mining Science*, May 11, 1911.

Zirconium is also precipitated by ammonium hydrate, and is then soluble in hydrochloric acid. From the solution it can then be precipitated from neutral solution (neutralized with sodium carbonate), by boiling with sodium hyposulphite. Then the precipitate is dissolved with water and on the addition of ammonium carbonate, any titanium that may be present will be precipitated. The zirconium will be in the solution and is recovered by the addition of potassium sulphate which causes a precipitate. The percentage of the zirconium is found by fusing this precipitate with potassium bisulphate and extracting with dilute sulphuric acid. The residue is fused with caustic soda and extracted with distilled water. The portion not dissolved, consisting of zirconate of soda, is dissolved in hydrochloric acid. The solution is then diluted, filtered, if necessary, and treated with ammonia in excess. The precipitate is filtered off, washed with hot water, dried, ignited and weighed as zirconia (ZrO_2), which is a white powder insoluble in acids.

On boiling a solution containing tantalum with hydrochloric acid, tantalum is precipitated. To this precipitate ammonium hydrate is added to remove any tungstic acid, if present. It is then fused with potassium bisulphate and dissolved with hydrofluoric acid and extracted with cold water, gradually adding a boiling solution of acid potassium fluoride. Potassium fluotantalate separates, and, after heating with sulphuric acid and washing out the potassium sulphate formed, the precipitate is ignited with ammonium carbonate and weighed as tantalic oxide (Ta_2O_5).

To determine titanium in ores, add hydrochloric acid and nitric acid, and from the solution the hydrate of titanium will be thrown down by the addition of ammonium chloride and ammonia. It is separated from other hydrates, in the precipitate, by fusing it with potassium bisulphate which renders the oxide of titanium soluble in cold water. On extraction with water, the solution is boiled, which causes a precipitate. This is collected, washed, dried and ignited with the addition of a little ammonium carbonate and weighed as titanic oxide (TiO_2).

To determine tungsten, take 2 grm. of the sample and boil with 50 c.c. of hydrochloric acid for half an hour, adding 5 c.c. of nitric acid toward the end. Allow the solution to stand over night and boil again for 20 minutes; then dilute with an equal volume of water and filter. The precipitate must then be washed with dilute hydrochloric acid and dissolved in a few cubic centimeters of warm, dilute ammonia. Then add a little distilled water and allow to settle and filter. In a dish of

known weight, evaporate, ignite and weigh the precipitate, which is tungstic acid (WO_3).

The precipitation of tungsten as W_2O_5 was recommended by B. Mdivani.[1] This is done by adding a freshly prepared solution of stannous chloride to the solution containing tungsten as WO_3. This throws down a precipitate of W_2O_5, which is ignited, and weighed as WO_3. Additional important bibliography for tungsten is appended, partly under steel analysis.

The Glow Reaction for Platinum Metals.—An investigation of the application of the well-known power of platinum and allied metals to condense gases, was made by Curtman and Rothberg.[2] The method of making the tests for the platinum metals was to dissolve the material to be tested and absorb about 0.2 c.c. into a piece of thin asbestos paper by alternately dipping the paper into the solution and heating it. The moist paper is finally heated to redness and after redness has ceased, but while the paper is still hot, it is brought into a stream of the mixed air and illuminating gas from a Bunsen burner. If platinum (or some of the allied metals) is present, it will begin to glow. The glow may last some time, and can be brought back, usually, by reheating, and again holding in the stream of air and gas. One part of platinum in 50,000 gives a test (the amount of Pt present on the asbestos was 0.002 mgrm.). With palladium 0.0005 mgrm. gave a glow; with rhodium, 0.0009 mgrm., but osmium and ruthenium give negative tests, probably due to volatilization of the oxides. Other metals interfere surprisingly little with this reaction.

Considering the difficulties encountered in the lithium determination, the following seems worthy of a trial.[3] Lithium, says E. Murmann, may be determined in a mixture containing its chloride, with those of sodium and potassium, by extraction with pyridine, in which the latter salts are insoluble. The pyridine extract is evaporated to dryness, and a slight excess of dilute sulphuric acid is added to the residue, after which the mixture is again evaporated, and the sulphate finally ignited and weighed. A trace of lithium remaining with the chloride residue may be recovered by dissolving this in water, evaporating, and again extracting with pyridine. The alkali chloride mixture must not be heated to redness, or very appreciable loss of lithium occurs.

Laboratory Apparatus.—One of the new ideas in laboratory apparatus which has been originated during the year, which seems helpful, is J. E. Spurrier's method of grooving the sides of a funnel with hydro-

[1] Trans. from *Bull.* Soc. chim. de Belg., in *Mining World*, Apr. 22, 1911.
[2] *Journ. Am. Chem Soc.*, 1912, p. 718.
[3] *Zeit. anal. Chem.*, 1911, pp. 171–174.

fluoric acid, to promote fast filtering.[1] The use of an alloy furnished by the General Electric Co. is recommended by Samuel A. Tucker for wire-wound resistance furnaces.[2] The alloy, known as calorite, is composed of nickel, iron, chromium and manganese, and has a melting-point of 1550° C. In many respects it is considered superior to platinum, or nichrome.

A support for crucible triangles made of soapstone is recommended by W. M. Hornton, Jr.,[3] of the Virginia Geological Survey, as being superior to the ordinary ringstand in cleanliness and convenience.

For titrations where hydrofluoric acid is present Metzger and L. E. Marrs advise the use of beakers made by cutting the tops off ceresine bottles.[3]

The use of chrome steel bucking boards and mullers at the Tooele smeltery, copied from the U. S. Steel Corporation, is favorably regarded,[4] as preventing undue contamination of the sample by extraneous iron.

A steam-jacketed drying pan for samples is described by C. C. Semple.[5] The arrangement consists of a cast-iron jacket with a rim around the top, setting down over steam coils. Rush samples are placed directly on the jacket top, but ordinarily pans are used which fit neatly within the rim.

Of interest to all chemists was the preliminary report of the committee on quality of platinum laboratory apparatus, of the American Chemical Society.[6] The objections to inferior ware are: Undue loss of weight on ignition; undue loss of weight on acid treatment, especially after strong ignition; unsightly appearance of the surface during and after ignition; adherence of dishes to triangles after heating; alkalinity of the surface of the ware after ignition; blistering and development of fire cracks. The loss of weight on ignition seems to be due partly to base metals, and partly to the iridium, added presumably to stiffen the metal. Of prime importance to chemists is the conclusion that with platinum-iridium ware, the loss on heating is almost negligible up to 1100° C., and over that increases sharply, even with pure platinum, the moral being to use the blast lamp as little as possible. The sticking of platinum ware to platinum triangles was found great, even with the best platinum. It is recommended that the triangles be of the same grade as the dishes, and that considering the heavy cost of platinum dishes, no experiments in false economy by using nichrome triangles,[7] be made. But platinum triangles show little advantage over clay,

[1] *Journ. Am. Chem. Soc.*, 1911, p. 1585.
[2] *Journ. Ind. and Eng. Chem.*, Aug., 1911.
[3] *Journ. Ind. and Eng. Chem.*, May, 1911.
[4] *Eng. and Min. Journ.*, Oct. 7, 1911.
[5] *Eng. and Min. Journ.*, Apr. 22, 1911.
[6] *Journ. Ind. and Eng. Chem.*, Sept., 1911.
[7] THE MINERAL INDUSTRY, 1910, p. 799.

especially clay triangles with projecting points, or one-piece quartz triangles, or nickel or nichrome covered with quartz. The alkalinity of platinum ware, which may be shown by laying moist red-litmus paper on the dish after heating, is an interesting property, being due to lime. This, it is supposed arises from absorption of lime from the crucibles in which the platinum is melted. The other defects are in the main due to this lime, or to other base impurities, chiefly iron. (For instance, one worker obtained 6 mgrm. of Fe_2O_3 off one 22-grm. dish.) These troubles will, it is feared, multiply as more and more scrap comes back on the market to be reworked.

In closing this article, I must apologize for many omissions. Many articles are long in space, and equally so in ideas, and brief abstracts are impossible. Certain of these I have noted in the bibliography following. Many equally good articles I have probably not seen, "so broad is learning." Many others, I have intentionally omitted giving mention, because of what seemed to me over-specialization, or needless refinements (as in atomic-weight work), some few, but I hope no one will jump to class his article, if I have omitted it, in this category, because I thought the subject too often treated even for this review, or because of what appeared to me to be intrinsic demerits.

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For Spanish readers, a good résumé of rapid smeltery methods was published in the *Mexican Mining Journal* of March, 1911, taken from the *Boletin* de la Soc. Nac. de Minera, Santiago, Peru. In English, see the Hawley paper, *Mex. Min. Journ.*, March, 1911.

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MCCABE. "Modification of Escka's Method for Sulphur in Coal and Coke." (*Chem. Engineer*, Jan., 1911.)

DATA OF THE WORLD'S PRINCIPAL MINES.
By S. F. Shaw.

Name of Mine.	Loc.	Year.	Tonnage Produced.	Profits.	Dividends.	Cost per Ton.	Ore Reserves Tons.
Abangarez	*j*	1911	45,514	$206,590	.....	$16.51	340,200
Abosso Gold	*z''*	1910	77,623	£30,991	£60,000	9.02	284,028
Ahmeek	*z'*	1911	598,549	870,272	.....	.....	.....
Alaska-Mexican	*a*	1910	222,698	380,793	342,000	1.75	1,093,028
Alaska-Treadwell	*a*	10–11	770,084	710,311	600,000	1.55	6,637,370
Alaska-United	*a*	1910	420,659	252,446	135,150	1.36	2,619,573
Allouez	*z'*	1910	.....	71,096	.....	1.80	.....
Amalgamated Zinc	*c*	10–11	319,256	£98,301	£75,000	.....	.....
Amparo	*o*	1910	68,217	281,527	240,418	6.03	.....
Anaconda	*z'*	1910	3,253,343	5,720,902	6,930,000	.....	.....
Anchor Tin	*y*	10–11	102,944	.....	.....	.....	.....
Arizona Copper	*z'*	10–11	744,746	.....	£214,518	0.93	.....
Ashanti Goldfields	*z''*	10–11	111,209	£144,565	£152,951	12.38	453,730
Associated Gold Mg	*c*	1910	128,181	141,000	.....	.....	.....
Assoc. Gold Mines of W. Aust.	*c*	10–11	105,238	— £5,846	.....	.....	.....
Assoc. No. Blocks	*c*	10–11	18,158	£16,325	.....	7.04	.....
Aurora West	*c*	1910	128,163	767,939	1,000,000	4.57	339,000
Baltic	*z'*	1910	781,419	.....	.....	1.67	.....
Batopilas	*o*	1910	43,612	— 106,258	.....	.....	417,877
Bibiana	*z''*	1910	67,565	£2,052	.....	6.37	397,759
Bingham-N. Hav	*z'*	1911	285,426	.....	228,689	.....	.....
Blackwater	*r*	1910	39,192	£39,231	£37,498	6.70	69,000
Boleo	*o*	1910	365,000	.....	.....	.....	.....
Boston and Montana	*z'*	1909	1,374,504	.....	.....	.....	.....
Brakpan	*w*	1910	.....	.....	.....	.....	1,720,181
Brilliant Extended	*c*	½–11	27,580	£10,002	£12,500	8.84	40,000
British Broken Hill	*c*	½–11	63,370	£15,142	£26,400	6.43	.....
British Col. Cop. Co	*f*	1911	384,588	.....	147,931	.....	.....
Broken Hill Bl. 10	*c*	½–11	48,754	.....	£10,000	.....	.....
Broken Hill Bl. 10	*c*	½–11	33,831	— £16,523	.....	.....	.....
Broken Hill Bl. 14	*c*	½–11	10,753	.....	11,500	.....	220,000
Broken Hill Bl. 14	*c*	½–11	11,064	.....	.....	8.79	.....
Broked Hill North	*c*	½–11	128,270	£66,808	£52,500	4.02	1,500,000
Broken Hill Prop	*c*	10–11	52,441	£167,814	£48,000	.....	.....
Broken Hill South	*c*	½–11	175,044	£72,083	£60,000	4.97	.....
Buffalo	*s*	10–11	41,484	.....	370,000	10.09	.....
Bunker Hill and Sullivan	*z'*	10–11	483,290	870,986	1,062,750	5.53	.....
Burma Mines	.....	1910	28,772	£9,034	.....	25.62	.....
Butte and Boston	*z'*	1910	372,965	751,712	399,680	6.98	.....
Butte Coalition	*z'*	1909	591,737	.....	250,000	.....	.....
Butters Salvador	.....	1909	21,664	£68,129	£103,125	18.37	100,000
Calumet and Ariz	*z'*	1911	.....	2,572,241	.....	.....	.....
Calumet and Hecla	*z'*	10–11	2,795,514	.....	.....	1.92	.....
Camp Bird	*z'*	10–11	79,186	£219,434	£220,000	9.39	38,415
Cananea Cons	*o*	1910	795,313	459,202	.....	5.76	.....
Cape Copper	*w*	10–11	82,925	.....	.....	.....	180,200
Carn Brea and Tincroft	*i*	½–11	38,253	— £1,936	.....	5.46	.....
Carn Brea and Tincroft	*i*	½–11	41,670	.....	.....	5.43	.....
Caucasus Copper	.....	1909	30,100	.....	.....	.....	.....
Centennial Eureka	*z'*	1911	108,275	.....	.....	.....	.....
Centennial	*z'*	1910	106,095	— 20,742	.....	.....	.....
Chaffers	*c*	1910	46,584	23,000	.....	.....	.....
Champion	*z'*	1910	722,051	939,205	$900,000	1.90	.....
Champion Reef	*m*	10–11	205,169	£133,808	£112,688	7.92	365,296
Chillagoe	*c*	10–11	60,943	£21,000	.....	.....	.....
Cinderella Deep	*w*	1910	181,100	£43,225	.....	5.59	421,884
City and Suburban	*w*	1910	308,266	£89,291	£68,000	5.02	828,936
Coniagas	*s*	10–11	32,849	.....	$1,440,000	.....	10
Cons. Goldfields of New Zealand.	*r*	1910	21,363	.....	£24,273	4.48	25,919
Con. Main Reef	*w*	10–11	252,485	£87,974	£80,881	5.22	516,844
Con. Min. and Smelt	*f*	10–11	388,785	202,278	.....	.....	.....
Copiapo	*k*	1910	12,752	— £16,595	.....	26.18	56,425
Copper Queen	*z'*	1911	.....	4,155,011	5,200,000	.....	.....
Corocoro United Cop	*d*	10–11	47,481	£3,646	.....	.....	.....
Creston	*o*	1910	107,500	.....	.....	4.74	284,310
Crown Mines, Ltd	*w*	1910	1,510,070	£1,182,224	£1,128,127	4.41	6,282,719
Crown Reef	*w*	1909	172,387	£118,822	.....	.....	.....
Crown Reserve	*s*	1910	2,749	1,368,124	1,050,000	.....	.....
Daly Judge	*z'*	1911	59,461	.....	$180,000	.....	.....
Daly West	*z'*	1911	88,141	278,259	162,000	.....	.....
DeLamar	*z'*	1910	44,961	£4,142	£4,000	9.90	.....

DATA OF THE WORLD'S PRINCIPAL MINES.—*Continued.*

By S F Shaw.

ne of Mine.	Loc.	Year.	Tonnage Produced.	Profits.	Dividends.	Cost per Ton.	Ore Reserves Tons.
per	z′	1911	517,087	$930,496	$800,000	$	
efs	m	1909	21,574				
	i	½–11	40,315	£36,370	£34,000	7.67	
	o	1910			270,000	12.60	64,759
as	o	1909	348,580	1,893,270		3.54	2,018,000
Sulphur, Copper	z′	1910		£38,376	£24,458		
odepoort	w	1910	240,530	£61,721	£33,000	6.14	867,551
	z′	10–11	85,876	268,651			
da	t	1909	91,400	— £34,884		6.46	
Prop	w	1909	1,830,280	£1,259,653		3.82	
anket	t	10–11	82,802	£92,742	£90,000	6.61	179,343
	o	10–11	360,294		£172,125	3.90	484,139
	o	1910			62,400	6.23	53,100
	o	1910	188,108	858,669	200,313	5.60	367,363
	b	1911	188,570				
	z′	10–11	759,108	1,241,115		5.88	863,340
	w	1910	346,150	£291,756	£285,000	3.84	598,671
ep	z	10–11	373,196	£327,732	£409,500	5.94	1,837,808
ldfields	z′	1911	48,874	— 42,008			
	z′	1909	170,546			2.06	1,250,000
rnace	z′	1910	375,677				
d Bolivia	d	1910	13,140	£10,137		18.61	
	t	10–11	29,550	11,777	£13,024	$6.52	30,000
Deep	w	1910	827,065	£251,305		5.32	2,521,481
	t	10–11	123,789	£97,860	£49,360	4.10	308,146
	w	1909	260,845	£121,739	£120,000	4.67	803,113
Phoenix	t	½–11	36,619				179,140
denburg	w	10–11	33,870	£44,022	£42,500	6.13	73,280
ge	c	1910	28,999	£167,500	£135,000		
seshoe	c	1910	257,333	£75,294	£75,000	5.86	803,151
ons	z′	1911	330,549	7,379,651	7,118,296	8.52	600,000
	f	1910	1,175,548	564,946	270,000	3.08	6,250,000
der Prop	c	1910	195,407	£266,610	£262,500	8.18	760,177
der Perseverance	c	1910	91,852	— £20,913		4.40	772,214
r	c	10–11	475,988				3,088,000
d	c	1910	125,871	£21,642	£25,000	6.39	
oy	c	10–11	69,791	— £9,836		10.26	590,522
Con	i	1911	17,059	£6,956		6.65	
Red. and Mines	o	1909	86,580			4.16	
s	o	1910	228,344	376,837		2.44	48,690
	c	10–11		— £10,712			
	c	10–11	62,322	£4,607	£3,750	6.21	100,000
	z′	1910	129,480		200,000		
d Mining	f	1911	57,815	318,152	300,000		
	z′	10–11	25,765		418,542		
	z′	10–11	1,468,263		1,310,400	2.69	
	z′	½–11	888,057		764,400	2.53	
	z′	1911		— 9,397			
Mine	m	1910	38,706	£9,236	£3,480	6.44	40,000
ce (Stratton's)	z′	10–11	109,800	£21,000	£12,500		
m	z′	1910	35,933	299,242	280,000		
Mg. Co	z′	1911	42,673	187,600	$100,000		
	z′	1911	457,440	156,708		1.67	
	c	1910	232,842	236,785	220,000	5.36	985,036
	w	1910	55,011	£2,428			
l	t	10–11	40,490			8.48	30,803
	w	10–11	120,802	£43,865	£40,000	7.25	122,314
ep	w	1909	92,960	£64,366	£10,000		60,000
	w	1910	267,398	£46,879		4.92	1,074,009
	c	10–11	127,010	£116,326	£136,500	7.01	
	s	10–11	1,810	$937,379	1,200,000	162.46	
ep	w	10–11	695,670	£252,833	£225,234	2.99	1,650,000
ntral	w	1910	302,228	£79,011	£45,000	4.25	700,000
efs	r	1909	13,390	2,371		8.61	20,900
orporation	v	1910	89,509	£64,335			1,500,000
	o	10–11	67,354				
r	z′	1909	10,124				
Estate	w	1910	629,614	£300,134	£260,850	$ 4.22	1,267,769
	s				$500,450		
	f	1909	26,534	— £9,701		15.20	
2	f	10–11	44,619	£30,176	£36,000	6.68	

DATA OF THE WORLD'S PRINCIPAL MINES.—*Continued.*

By S. F. Shaw.

Name of Mine.	Loc.	Year.	Tonnage Produced.	Profits.	Dividends.	Cost per Ton.	Ore Reserves Tons.
Lena Goldfields	*v*	10–11	.....	£333,643	£336,664	.....	.....
Liberty Bell	*z'*	1910	133,899	$282,147	£143,606	$5.27	.....
Lucky Tiger	*o*	1910	40,690	$485,304	$429,000	10.06	218,280
Luipaardsvlei	*z*	.....	.....	.....	.....	.....	398,000
Main Reef West	*z*	10–11	196,391	£139,873	£101,250	4.92	628,200
Mammoth	*z'*	1909	401,244	.....	.....	.....	.....
Mason and Barry	*x*	1910	360,406	£65,458	£55,551	.....	.....
May Consolidated	*w*	1909	178,845	.....	£115,500	4.21	400,416
McKinley Darragh	*s*	1910	43,582	$976,640	.....	10.36	126,410
Mercur Con	*z'*	10–11	239,190	$496	.....	2.32	.....
Mexico Mines of El Oro	*o*	10–11	136,408	.....	£144,000	.....	318,720
Meyer and Charlton	*w*	1910	153,717	.....	£80,000	4.61	277,029
Miami	*z'*	1911	450,036	.....	.....	3.02	18,232,000
Midas Deep	*w*	1909	20,776	.....	.....	.....	.....
Minerals Separation	*c*	1910	179,218	.....	.....	1.82	.....
Moctezuma	*o*	1911	517,352	$1,206,183	$754,000	.....	.....
Mohawk	*z'*	1910	802,537	$188,083	$200,000	1.60	.....
Montana-Tonapah	*z'*	10–11	52,092	$172,602	$120,000	.....	.....
Montgomery-Shoshone	*z'*	1909	68,632	$86,703	.....	4.32	.....
Morning	*z'*	1910	333,900	.....	.....	.....	.....
Mt. Bischoff	*y*	½–10	110,664	£35,311	£21,000	0.94	.....
Mt. Boppy	*q*	1910	78,690	.....	£42,350	5.37	234,684
Mt. Elliott	*c*	10–11	42,616	£130,907	£73,759	.....	215,000
Mt. Lyell	*y*	½–11	183,094	£85,775	£75,000	4.87	.....
Mt. Lyell	*y*	½–11	160,965	£69,033	.....	.....	3,519,670
Mt. Morgan	*c*	10–11	354,137	£136,563	£200,000	11.21	3,613,500
Mungana	*c*	1909	26,052	£1,096	.....	9.06	.....
Mysore	*m*	1911	291,477	£424,274	£366,600	6.18	1,300,500
Nababeep	*w*	1910	54,050	.....	.....	.....	219,800
Napa Consolidated	*z'*	1909	18,583	61,899	.....	.....	.....
Namaqua Copper	*w*	1910	30,539	£34,794	£23,583	.....	94,583
Natomas Consolidated	*x'*	1909	.....	$870,410	.....	.....	.....
Nevada Consolidated	*z'*	1909	2,237,028	$2,345,382	.....	2.76	40,360,523
New Goch	*w*	1910	338,616	£140,626	£82,500	4.71	741,688
New Heriot	*w*	1910	149,990	£107,504	£92,000	4.93	485,585
New Idria	*z'*	1909	67,247	$109,639	$120,000	.....	.....
New Kleinfontein	*w*	1910	466,882	£288,577	£218,250	3.89	1,397,412
New Modderfontein	*w*	10–11	538,400	£294,010	£297,500	.....	3,452,195
New Primrose	*w*	1910	257,168	£210,278	£185,000	4.00	490,800
N. Y. & Honduras-Rosario	*l*	10–11	35,813	.....	.....	14.79	121,088
Nigel	*w*	1910	208,880	.....	£66,931	5.21	227,304
Nipissing	*s*	1910	6,699	$2,167,103	2,100,000	.....	.....
North Butte	*z'*	1910	408,528	.....	$446,000	.....	958,680
North Star, Calif.	*z'*	1911	95,401	$419,528	$425,000	5.86	.....
Nourse	*z*	10–11	643,675	£220,319	£165,564	5.40	2,050,700
Nundyroog	*m*	1911	102,873	£140,573	£127,350	9.38	.....
Old Dominion	*z'*	1911	.....	$621,964	.....	.....	.....
Oregum	*m*	1910	133,407	£148,406	£129,250	7.60	164,534
Oriental Consolidated	*h*	10–11	342,956	$672,720	$644,085	2.53	392,280
Original Con	*z'*	1909	226,492	$181,359	.....	7.29	.....
Oroyo-Black Range	*c*	1910	54,180	£24,516	£24,986	8.39	.....
Oroya-Links	*c*	1910	100,177	$125,000	$71,000	.....	.....
Oroville Dredging	*z'*	1910	566,612	$275,452	.....	.....	.....
Osceola	*z'*	1911	1,246,596	$664,628	$673,050	1.37	.....
Otavi	*z''*	10–11	42,000	.....	£224,500	5.59	.....
Ouro Preto	*e*	10–11	69,680	£4,946	£6,163	6.57	170,834
Pahang Cons	*n*	10–11	78,448	£21,232	.....	.....	.....
Panuco	*o*	1909	13,411	$238,419	$212,500	4.15	.....
Parrot	*z'*	1909	122,349	.....	$68,954	.....	.....
Penhalonga	*t*	10–11	49,900	£966	.....	5.19	.....
Peñoles	*o*	1909	176,616	$298,570	$499,200	5.58	.....
Pittsburg Silver Peak	*z'*	1909	150,359	$337,004	.....	.....	.....
Poderosa	*k*	1910	23,341	£38,471	.....	.....	35,000
Portland	*z'*	10–11	67,515	$367,380	$240,000	14.62	.....
Princess Estate	*w*	1910	120,414	£58,431	£53,000	5.53	.....
Progress	*r*	1910	50,202	— £13,220	£10,312	8.39	.....
Prestea Block A	*z''*	1910	.....	.....	.....	.....	678,859
Quicksilver Mng	*z'*	10–11	24,976	$22,182	.....	4.99	.....
Quincy	*z'*	1911	1,382,524	$507,597	.....	1.72	.....
Randfontein, Central	*w*	1910	.....	.....	.....	.....	2,822,713
Randfontein, South	*w*	1910	1,168,641	$484,011	$300,000	4.64	2,788,000
Rand Mines, Ltd	*w*	1909	.....	.....	.....	4.46	4,000,000

DATA OF THE WORLD'S PRINCIPAL MINES.—*Continued.*
By S. F. Shaw.

Name of Mine.	Loc.	Year.	Tonnage Produced.	Profits.	Dividends.	Cost per Ton.	Ore Reserves, Tons.
Real Del Monte	*o*	1909	215,209	.....	.....	.....	.....
Republica	*o*	½–10	3,870	.....	.....	$19.08	.....
Rezende	*t*	1910	43,800	£16,624	.....	5.21	112,980
Rio Plato	*o*	1909	14,580	$20,046	$140,037	8.18	.....
Rio Tinto	*x*	1911	2,185,605	.....	£1,065,625	.....	.....
Robinson	*w*	1909	550,176	£940,696	£825,000	3.05	2,788,362
Robinson Deep	*z*	10–11	533,850	£334,379	.....	4.59	1,140,000
Rooiberg Min. Dev	*w*	10–11	17,781	£52,674	£13,500	.....	21,650
Rose Deep	*w*	1910	656,954	£289,813	£280,000	4.75	3,553,474
Round Mountain	*z'*	1910	36,252	$84,313	$69,322	6.17	111,928
Salisbury Gold	*w*	10–11	93,396	£15,306	£13,500	3.61	82,896
San Rafael y Anexas	*o*	½–11	71,587	$273,057	$217,200	.....	.....
Santa Gertrudis	*o*	.....	.....	.....	.....	.....	630,000
San Toy	*o*	1910	20,181	$157,556	.....	25.70	.....
Selukwe-Columbia	*t*	10–11	31,535	£14,965	£12,500	7.94	40,000
Seoul Mining	*h*	1910	32,793	$215,151	$100,000	4.66	190,100
Shannon	*z'*	10–11	263,975	.....	.....	.....	1,000,000
Shattuck Ariz	*z'*	10–11	32,029	$396,687	.....	.....	.....
Sheba Gold Mining	*w*	10–11	49,455	£20,573	£13,486	7.75	104,250
Simmer and Jack	*w*	10–11	811,448	£526,866	£600,000	3.11	2,520,000
Simmer & Jack East	*w*	1910	365,620	£63,989	.....	4.33	950,000
Simmer Deep	*w*	1910	480,803	£60,712	.....	4.03	1,255,000
Skidoo	*z'*	1909	14,726	92,617	50,000	6.55	.....
Snowstorm	*z'*	10–11	34,463	£25,691	$44,985	4.23	.....
Sons of Gwalia	*c*	1910	162,082	£112,360	£97,500	.....	.....
South Crofty	*i*	1911	63,882	£31,981	£17,500	5.24	.....
South Kalgurli	*c*	10–11	112,170	£20,465	£20,000	5.35	173,200
Spassky Copper	*v*	1910	10,794	£9,522	.....	.....	170,100
Standard Con	*z'*	1911	9,466	.....	$17,839	.....	.....
St. John del Rey	*e*	10–11	189,600	.....	£60,078	7.12	1,158,560
Stewart	*z'*	1910	55,347	.....	.....	.....	.....
Sudan Goldfield	*w*	1910	14,662	— £6,227	.....	10.31	6,010
Sulphide Corp	*c*	10–11	226,768	£179,448	£144,375	17.65	.....
Superior	*z'*	1911	162,599	— $51,989	.....	2.97	.....
Superior & Pittsburg	*z'*	1909	251,042	.....	.....	10.81	.....
Surprise	*t*	1909	29,643	— £6,835	.....	6.50	.....
Talisman Con	*r*	10–11	45,850	£110,307	£100,447	10.38	.....
Tamarack	*z'*	1910	525,554	— $191,345	.....	2.67	.....
Taquah Mining & Exp	*z''*	1910	65,222	.....	.....	13.51	102,370
Tasmania Gold Mng	*y*	10–11	53,968	£4,020	.....	9.16	43,084
Temiskaming	*s*	1911	.....	$271,423	$224,999	10.19	.....
Tennessee Copper	*z'*	1910	405,463	.....	$300,000	2.80	4,563,671
Tharsis Sulphur	*x*	1910	362,750	.....	£156,250	.....	.....
Thistle-Etna	*t*	1910	37,261	£1,426	.....	5.55	.....
Tilt Cove	*p*	10–11	29,077	£4,733	.....	.....	40,000
Tincroft	*i*	½–09	36,825	.....	.....	.....	.....
Tomboy	*z'*	10–11	116,222	$272,267	£60,000	4.37	427,000
Tombstone Con	*z'*	1909	15,070	.....	.....	.....	.....
Tonapah Belmont	*z'*	.....	115,563	$1,500,000	.....	.....	706,082
Tonapah Mining	*z'*	1911	177,745	.....	1,550,000	10.07	354,940
Treasury Gold	*z*	10–11	64,230	£16,482	.....	5.06	250,000
Trenton	*z'*	1909	157,167	165,111	470,000	7.04	.....
Trethewey	*s*	1910	19,103	$237,800	.....	.....	.....
Tri-Mountain	*z'*	1910	317,299	$32,250	$150,000	.....	.....
Troitsk Goldfield	*v*	10–11	40,628	— £8,945	.....	6.20	.....
Tronah	*n*	1910	410,777	£72,927	£56,000	.....	.....
Tuolumne	*z'*	1911	30,499	.....	.....	.....	.....
Tyee Copper	*f*	1910	45,760	£1,200	.....	.....	.....
U. S. S. R. & Mng. Co.	*z'*	1911	1,037,685	.....	2,404,669	.....	.....
Utah Apex	*z'*	1908	33,407	$124,185	.....	.....	80,000
Utah Con	*z'*	1911	.....	$438,429	$150,000	.....	263,435
Utah Copper	*z'*	1910	4,340,245	$5,401,775	$4,648,675	.....	152,130,000
Van Roi	*f*	1911	22,427	— £2,661	.....	.....	.....
Van Ryn	*w*	10–11	396,950	£210,847	£225,000	4.36	1,655,910
Village Deep	*w*	1910	508,540	£154,067	£106,067	4.45	1,704,226
Village Main Reef	*w*	1910	492,446	£360,654	£330,400	4.91	1,775,175
Vindicator	*z'*	1911	30,233	$238,804	.....	15.47	.....
Vivian Gold Mining	*c*	1909	41,146	£2,241	.....	5.60	35,219
Vogelstruis Est	*z*	10–11	101,033	— £18,074	.....	7.62	147,315
Waihi	*r*	1910	442,020	£559,191	£396,725	4.04	.....
Waihi Grand Jc	*r*	1910	84,226	£30,646	.....	3.71	.....
Wanderer	*t*	10–11	195,234	$15,190	.....	2.22	424,195
Washoe	*z'*	1909	205,377	.....	.....	.....	.....

DATA OF THE WORLD'S PRINCIPAL MINES.—*Continued.*

By S. F. Shaw.

Name of mine.	Loc.	Year.	Tonnage Produced.	Profits.	Dividends.	Cost per Ton.	Ore Reserves, Tons.
West End Con	*z'*	10–11	16,164	42,490	.....	$18.29	.....
West Rand Prop	*w*	1910	242,000	£46,555	.....	4.78	1,124,631
Wheal Kitty	*i*	1–11	7,816	£383	£841	6.98	.....
Witwatersrand Deep	*w*	1910	474,600	£267,413	£271,824	4.55	1,692,922
Witwatersrand Gold	*w*	1910	426,910	.....	£148,750	3.83	1,232,784
Wolhuter	*w*	1910	304,360	£122,554	£86,000	4.26	733,679
Wolverine	*z'*	10–11	.....	$485,705	$600,000	.....	.....
Worchester Expl. & Gold	*w*	1910	55,330	£25,177	£16,751	3.87	.....
Yukon Gold	*a*	1911	6,276,800	.....	.....	.....	.....
Zinc Corporation	*z'''*	1910	270,637	£97,906	£89,513	.....	.....

Note.—*a*, Alaska; *b*, Argentina; *c*, Australia; *d*, Bolivia; *e*, Brazil; *f*, British Columbia; *g*, Burma; *h*, Korea; *i*, Great Britain; *j*, Costa Rica; *k*, Chile; *l*, Honduras; *m*, India; *n*, Malay States; *o*, Mexico; *p*, Newfoundland; *q*, New South Wales; *r*, New Zealand; *s*, Ontario; *t*, Rhodesia; *u*, Salvador; *v*, Siberia; *w*, South Africa; *x*, Spain; *y*, Tasmania; *z*, Transvaal; *z'*, United States; *z''*, West Africa; *z'''* Yukon.

The above tabulation of data obtained from abstracts of annual mining reports published in the principal mining magazines, was compiled for the purpose of securing relative data on the costs of production of the various metal-mining companies operating in different parts of the world.

It will be seen that English companies form a large percentage of those enumerated, as there is a greater tendency for these companies to make public the items of production, cost, ore reserves, etc., than is the case with American companies.

These data cover for the most part the calendar and fiscal years ending in 1910 and 1911.

The column headed "tonnage produced" refers to the tons of ore actually shipped to the smeltery or treated locally and does not include waste which was sorted out of the ore hoisted. It is customary among the Rand mines to note the amount of waste so hoisted and sorted out, but this applies to but few American mines.

Where sufficient data were given to enable it to be done, the profits are figured as the amount remaining after all expenses of operation, administration, depreciation, development, taxes, etc., were paid. Such money as was put into the capital account, however, is not included.

In figuring the cost per ton, all items in the nature of expenses of operation, administration, depreciation, etc., are included exactly as in the case of computing the profits.

The reserves are, of course, those given in the official report as the number of tons of ore remaining blocked out at the end of the year under review. Probable ore is not so included, neither is that portion of the vein which, though blocked out, is too low grade to return a profit.

AUSTRALASIA

In the following tables the production of minerals and metals in each of the Australian States and New Zealand is separately itemized. In the tables relating to foreign commerce, however, the States are not separately treated, the combined statistics of the Commonwealth now being officially reported.

MINERAL PRODUCTION OF NEW SOUTH WALES. (*a*)
(In metric tons or dollars; £1 = $5.) (*b*)

Year.	Alunite.	Antimony and Ore.	Bismuth Ore.	Chrome Ore.	Coal.	Coke.	Cobalt Ore.	Copper Ore.
1897.....	736	172	3	3,433	4,453,729	65,229	....	169
1898.....	2,988	83	29	2,145	4,781,551	83,538	119	181
1899.....	935	332	16	5,327	4,670,580	98,074	193	445
1900.....	1,946	252	11	3,338	5,595,879	128,238	145	867
1901.....	3,196	90	21	2,523	6,063,921	130,944	112	655
1902.....	3,702	58	10	508	6,037,083	128,902	35	3,190
1903.....	2,524	13	23	1,982	6,456,523	163,161	155	1,750
1904.....	376	111	41	404	6,116,126	173,742	6	2,470
1905.....	2,745	394	56	53	6,738,252	165,568	*Nil.*	487
1906.....	1,886	2,490	25	15	7,748,384	189,038	*Nil.*	(*g*)
1907.....	2,021	1,780	17	30	8,796,451	258,683	*Nil.*	(*g*)
1908.....	1,099	119	9	*Nil.*	9,293,377	288,413	*Nil.*	(*g*)
1909.....	3,556	97	9	*Nil.*	7,132,548	207,553	*Nil.*	(*g*)
1910.....	1,154	99	7	*Nil.*	8,304,693	286,854	10	(*g*)
1911.....	1,023	168	9	*Nil.*	8,830,504	293,709	*Nil.*	(*g*)

Year.	Copper Matte, Ingot and Regulus.	Diamonds. *Karats.*	Gold. (*b*)	Lead, Argentiferous. (*f*)		Lead. Pig. (*f*)	Molybdenite.	Opal.
				Ore.	Metal. (*e*)			
1897....	6,458	9,189	$5,373,596	275,249	18,395	32	.....	$375,000
1898....	5,577	16,493	5,847,680	394,676	10,270	1,745	.....	400,000
1899....	5,574	25,874	7,899,075	431,126	20,614	(*d*)4,896	.....	675,000
1900....	6,243	9,828	5,211,097	426,480	19,400	(*d*)6,807	.....	400,000
1901....	6,184	9,322	3,587,040	406,560	17,191	(*d*)3,394	.....	600,000
1902....	5,560	11,995	3,333,064	371,496	15,660	(*d*)4,685	16	700,000
1903....	8,094	12,239	5,255,421	335,870	18,779	(*d*)3,561	31	500 000
1904....	6,654	14,296	5,576,966	373,362	30,212	(*d*)5,977	26	285,000
1905....	7,899	6,354	5,669,099	420,266	28,244	214	20	295,000
1906....	9,911	2,827	5,249,762	377,890	22,573	60	34	282,500
1907....	10,260	2,539	5,112,852	441,024	20,687	20,084	22	395,000
1908....	9,215	2,205	4,646,451	364,488	(*h*)	15,174	9	209,000
1909....	7,078	5,474	4,231,211	273,628	(*h*)	15,724	29	309,000
1910....	13,096	3,606	4,011,055	322,780	(*h*)	21,534	50	331,000
1911....	12,295	5,771	3,743,672	343,902	(*h*)	17,552	21	286,500

Year.	Platinum. *Kg.*	Shale Oil.	Silver—*Kg.* (*f*)	Tin.		Tungsten Ore.	Zinc. (*c*) (*f*)
				Ore.	Block.		
1897.....	61.2	34,635	4,666	14	799	.....	29,303
1898.....	38.9	30,164	16,580	1	639	.....	39,564
1899.....	19.8	37,307	21,525	5	749	.....	50,677
1900.....	15.6	23,229	24,080	15	1,087	.....	20,594
1901.....	12.1	55,650	13,950	11	659	.....	642
1902.....	11.6	63,886	33,195	23	502	.....	1,281
1903.....	16.5	35,332	34,195	556	949	9	21,086
1904.....	16.6	38,477	34,880	586	1,084	106	58,523
1905.....	12.4	38,838	12,987	726	817	228	105,189
1906.....	6.4	32,965	8,865	(*h*)	1,698	245	105,325
1907.....	8.6	48,088	63,573	(*h*)	1,945	409	241,015
1908.....	4.2	47,044	77,490	(*h*)	1,822	247	281,147
1909.....	13.7	49,500	53,430	(*h*)	1,974	129	379,907
1910.....	10.3	69,385	55,176	(*h*)	1,898	168	476,125
1911.....	14.6	76,306	54,975	(*h*)	1,960	288	524,666

(*a*) From the Annual Report of the Department of Mines, New South Wales. (*b*) Where gold is reported £1 = $4.866. (*c*) Spelter and concentrate. (*d*) Includes minor quantities of lead carbonate and chloride, the product of the leaching plant at Broken Hill. (*e*) Includes a small quantity of silver-sulphide. (*f*) Exported. (*g*) Included with metal. (*h*) Included with ore.

MINERAL PRODUCTION OF QUEENSLAND. (*a*)
(In metric tons or dollars; £1 = $5.)

Year.	Bismuth Ore.	Coal.	Copper.	Gold (*b*).	Lead.	Manganese Ore.
1897.....	1	364,142	293	$16,699,477	391	403
1898.....	8	414,461	63	19,016,763	252	68
1899.....	2	501,913	164	19,571,662	57	747
1900.....	8	505,252	386	20,002,290	207	77
1901.....	20	548,104	3,110	12,367,276	570	221
1902.....	1	509,579	3,845	13,238,500	271	4,674
1903.....	11	515,950	4,995	13,818,653	3,856	1,341
1904.....	20	520,232	4,440	13,210,869	2,079	843
1905.....	15	537,795	7,337	12,249,157	2,464	1,541
1906.....	7	610,480	10,238	11,257,316	2,854	1,131
1907.....	6	694,204	12,959	9,641,789	5,240	1,134
1908.....	22	707,473	14,932	9,613,051	7,207	1,403
1909.....	11	768,720	14,727	9,416,576	5,323	613
1910.....	22	885,108	16,649	9,123,531	2,435	805
1911.....	10	905,568	20,709	7,981,791	1,799	1,167

Year.	Molybdenite.	Opal.	Silver. *Kg.*	Tin Ore.	Tungsten Ore.
1897.....	.....	$51,250	7,280	1,222	13
1898.....	.....	33,225	3,235	1,041	79
1899.....	.....	45,000	4,521	1,322	263
1900.....	.....	37,500	3,514	1,133	193
1901.....	.....	37,000	17,777	1,638	73
1902.....	(*c*) 42	35,000	21,813	2,118	56
1903.....	(*c*) 24	36,500	19,972	3,768	200
1904.....	(*c*) 22	17,750	20,370	3,986	1,564
1905.....	64	15,000	18,716	4,008	1,434
1906.....	108	15,000	24,357	4,900	785
1907.....	68	15,000	28,662	5,222	627
1908.....	89	12,500	36,200	4,903	426
1909.....	94	10,000	31,140	3,379	617
1910.....	108	15,000	26,787	3,000	869
1911.....	232	15,000	17,076	3,140	548

(*a*) From *Annual Reports* of the Under Secretary of Mines, Queensland. (*b*) Where gold is reported £1 = $4.866. (*c*) Includes bismuth and tungsten.

MINERAL PRODUCTION OF SOUTH AUSTRALIA.
(In metric tons or dollars; £1 = $5.) (*b*)

Year.	Copper.		Gold. (*b*)	Iron Ore.	Lead.	Lime-stone.	Phosphate Rock.	Salt.	Other Metals and Minerals.
	Ore.	Metal.							
1898.....	545	4,847	$51,949	.....	321	.....	.....	.....	$ 2,800
1899.....	2,938	5,584	75,822	.....	370	.....	.....	.....	6,785
1900.....	2,405	4,964	70,528	.....	389	.....	.....	.....	2,055
1901.....	1,896	6,844	80,839	.....	69	.....	.....	.....	11,095
1902.....	2,620	6,956	121,056	.....	2,210	.....	.....	.....	3,710
1903.....	7,182	6,594	139,411	86,291	732	.....	1,016	40,640	500
1904.....	3,100	6,378	369,938	47,434	.....	44,135	3,048	40,640	990
1905.....	2,604	6,653	223,121	85,835	53	45,210	5,080	33,020	6,305
1906.....	535	8,339	131,382	76,430	51	32,451	5,944	55,880	11,045
1907.....	.....	8,058	99,948	85,954	.....	31,598	8,128	76,200	12,500
1908.....	.....	5,718	59,852	89,412	.....	29,973	11,177	76,204	22,500
1909.....	1,250	5,788	146,982	16,379	142	13,986	3,833	52,232	19,365
1910.....	.....	5,184	136,248	46,939	406	18,898	5,283	54,862	68,000
1911 (*a*)..	.....	6,017	72,990	42,976	.....	29,159	5,893	66,040	56,600

(*a*) From *Review of Mining Operations* by Hon. Richard Butler, Adelaide, 1911. (*b*) Where gold is reported, £1 = $4.866.

MINERAL PRODUCTION OF TASMANIA. (*a*)
(In metric tons or dollars; £1 = $5.) (*b*)

Year.	Coal.	Copper Ore.	Blister Copper.	Gold. (*b*)	Iron Ore.	Lead-Silver Ore.	Tin Ore.
1898.....	49,902	(*c*)	(*d*)	$1,369,706	1,296	196,707	2,882
1899.....	43,803	(*d*)	(*d*)	1,593,834	6,726	424,552	3,333
1900.....	51,549	4,221	9,343	1,538,727	5,141	453,519	2,693
1901.....	49,963	11,401	10,141	1,436,326	1,422	804,463	2,516
1902.....	49,647	8,630	7,869	1,467,454	2,424	47,226	1,989
1903.....	49,856	3,891	6,791	1,237,925	6,076	43,103	2,414
1904.....	62,090	(*d*)	8,826	1,362,587	6,950	51,959	2,104
1905.....	52,825	(*d*)	9,919	1,520,101	6,401	76,424	3,953
1906.....	53,742	2,270	8,847	1,240,650	2,642	88,513	4,545
1907.....	59,833	1,261	8,378	1,350,836	3,048	91,216	4,412
1908.....	62,044	1,204	8,974	1,179,950	3,657	62,022	4,593
1909.....	67,224	1,613	8,472	951,005	*Nil.*	81,668	4,583
1910.....	83,763	682	8,324	765,784	*Nil.*	52,047	3,760
1911.....	57,980	2,323	6,118	628,375	*Nil.*	62,489	4,016

(*a*) From *Statistics of the Colony of Tasmania*. (*b*) Where value of gold is reported, £1 = $4.866. (*c*) Included with lead-silver ore. (*d*) Not reported.

MINERAL PRODUCTION OF NEW ZEALAND. (*a*) (*b*)
(In metric tons or dollars.) (*c*)

Year.	Anti-mony.	Chrome Ore.	Coal.	Coke.	Copper Ore.	Gold. (*c*)	Kauri-gum.	Manga-nese Ore.	Silver. *Kg.*
1898...	.....	.....	921,546	9	2	$5,258,642	10,063	220	9,140.0
1899...	.....	.....	990,838	18	.....	7,363,100	11,294	137	10,865.6
1900...	3	28	1,111,860	.....	12	7,005,103	10,322	166	10,202.0
1901...	30	.....	1,259,521	.....	3	8,533,908	7,662	211	17,762.0
1902...	.....	128	1,386,881	.....	.....	9,495,673	7,549	.....	20,970.3
1903...	.....	.....	1,542,953	.....	6	9,916,086	9,507	71	28,364.3
1904...	.....	.....	1,537,838	.....	.....	9,671,180	9,203	196	34,042.3
1905...	.....	.....	1,585,756	15	4	10,189,093	10,883	55	36,695.0
1906...	.....	.....	1,613,301	5	.....	11,050,219	9,300	16	43,251.5
1907...	100	.....	1,860,305	15	57	9,865,766	8,847	5	48,603.5
1908...	5	.....	1,890,751	2	13	9,755,303	5,618	*Nil.*	53,834.8
1909...	2	*Nil.*	1,941,923	23	5	9,765,575	8,382	6	56,410.0
1910...	*Nil.*	*Nil.*	2,232,520	*Nil.*	*Nil.*	9,227,532	8,832	5	52,176.8
1911...	20	*Nil.*	223,918	*Nil.*	*Nil.*	9,105,000	7,608	1	40,775.2

(*a*) From New Zealand Mines Statement, by the Hon. Roderick McKenzie, Minister of Mines, Wellington. (*b*) The exports are stated to be identical with the production, with the exception of coal, the exports of which were as follows: In 1896, 80,796 long tons; in 1897, 77,280 tons; in 1898, 57,333 tons; in 1899, 90,912 tons; in 1900, 116,216 tons; in 1901, 162,197 tons; in 1902, 191,696 tons; in 1903, 154,769 tons; in 1904, 165,220 tons; in 1905, 122,817 tons; in 1906, 141,641 tons; in 1907, 128,950 tons; in 1908, 100,502 tons, and in 1909, 201,685 long tons. (*c*) Where gold is reported £1 = $4.866.

MINERAL PRODUCTION OF VICTORIA. (a)
(In metric tons or dollars.)

Year.	Coal.	Lignite.	Gold. (c)	Building Stone, etc.	Tin Ore.
1897.....	240,057	4,894	$16,799,824	(e)$125,000	48
1898.....	246,845	2,915	17,305,547	100,000	87
1899.....	266,578	(b)	17,662,410	(b)	158
1900.....	215,052	(b)	16,767,261	175,000	71
1901.....	212,678	152	16,320,029	225,000	78
1902.....	228,777	(b)	14,899,876	266,975	10
1903.....	65,230	5,752	15,860,815	213,245	34
1904.....	123,695	*Nil.*	15,824,952	1,488,075	72
1905.....	157,648	*Nil.*	15,443,438	(b)	126
1906.....	163,201	*Nil.*	15,962,804	(f) 362,725	108
1907.....	140,802	*Nil.*	14,377,166	(f) 401,090	105
1908.....	115,283	*Nil.*	13,867,312	(f) 453,075	80
1909.....	130,230	*Nil.*	13,522,400	397,645	90
1910.....	374,775	660	11,789,077	574,775	42
1911.....	664,326	6,232	11,789,077	.....	33

(a) From annual Reports of the Secretary for Mines of the Colony. (b) Not reported. (c) Where gold is reported, £1 = $4.866. (e) Estimated value. (f) Includes crude salt.

MINERAL PRODUCTION OF WESTERN AUSTRALIA. (a)
(In metric tons or dollars.)

Year.	Tungsten Ore.	Coal.	Copper Ore.	Copper and Matte.	Gold. (b) (c)	Iron Ore.	Lead Ore.	Silver. *Kg.*	Tin Ore.
1901.....	.....	119,721	10,319	.....	$32,698,941	20,898	(d) 21	1,893	748
1902.....	.....	143,145	2,298	.....	37,026,119	4,877	(d) 36	2,590	630
1903.....	.....	135,568	20,854	.....	40,560,927	224	*Nil.*	5, 229	830
1904.....	.....	140,773	4,033	.....	39,557,933	1,465	*Nil.*	12,416	869
1905.....	.....	129,402	2,389	.....	38,045,366	3,264	*Nil.*	11,189	1,096
1906.....	.....	152,151	7,548	(f)3,676	35,888,278	1,300	*Nil.*	8,776	1,518
1907.....	.....	144,651	19,282	1,628	35,087,500	1,112	(d) 214	5,887	(e) 1,526
1908.....	.....	178,061	8,427	486	34,061,426	*Nil.*	(d) 526	5,240	(e) 1,111
1909.....	1	217,741	7,071	846	32,973,349	*Nil.*	(d) 214	5,500	(e) 709
1910.....	2	266,361	6,410	1,301	30,397,162	10	252	5,478	503
1911.....	10	253,900	9,974	832	28,331,272	*Nil.*	884	5,215	503

(a) From the *Report of the Department of Mines of Western Australia.* (b) £1 = $4.866. (c) The value of gold produced in 1895 was $4,280,855; in 1896, $5,200,821; in 1897, $12,481,176; in 1898, $19,418,735; in 1899, $30,396,593; in 1900, $27,461,865. (d) Silver-lead and lead ore. (e) Includes ingots. (f) Total in previous years.

MINERAL IMPORTS OF AUSTRALIA. (a)
(In metric tons, cwts. of 112 lb., or dollars; £1 = $5.) (b)

Year.	Cement. *Cwts.*	Coal.	Coke.	Copper Ore. *Cwts.*	Gold. (b)				
					Ore.	Bullion.	Specie.	Foil. (c)	Total Value
1901	1,422,647	10,141	36,814	14,520	$37,473	$3,709,848	$18,053	$34,704	$3,800,078
1902	1,074,482	5,149	9,846	29,236	2,375,513	3,834,510	505,899	30,028	6,740,950
1903	954,606	389	4,294	5	66,908	5,935,800	6,530	38,680	6,047,918
1904	561,237	398	4,270	12	68,309	5,684,164	6,297	43,215	5,801,985
1905	700,245	7,866	5,553	80	103,709	7,067,534	422,127	52,144	7,645,514
1906	793,928	706	6,202	873	93,116	10,053,463	397,990	53,356	10,597,925
1907	513,326	14,973	9,981	3,652	136,520	6,942,940	48,499	45,283	7,173,242
1908	915,033	14,833	10,368	3,959	42,855	4,625,498	70,197	.....	4,738,550
1909	848,337	16,044	44,668	4,533	36,670	4,865,436	56,280	118,240	5,076,626
1910	1,612,004	314,374	17,161	23,027	35,991	4,634,553	126,556	.....	(e)4,816,100
1911	1,670,117	10,498	22,608	10,201	59,613	7,707,920	124,248	.....	7,891,781

Year.	Graphite Cwts.	Iron and Steel.			Lead Mfrs. Cwts.	Petroleum Products.		
		Bars, Rods Girders, Sheets etc. Cwts.	Galvanized Plates and Sheets. Cwts.	Pig and Scrap. Cwts.		Kerosene. Gal.	Naphtha Gal.	Paraffin.
1901....	3,419	2,081,423	905,709	732,512	.....	20,924,640	114,092	1,040
1902....	2,659	.....	1,104,701	766,725	8,300	10,399,931	116,170	1,913
1903....	5,557	1,211,437	886,570	989,998	9,525	15,009,609	127,445	2,163
1904....	4,263	1,399,783	1,027,859	883,397	6,243	14,791,319	277,737	530
1905....	4,386	1,482,334	1,112,467	940,757	8,859	16,416,734	292,670	.....
1906....	6,531	1,878,851	1,245,211	1,220,236	14,830	15,473,570	488,961	.....
1907....	6,991	2,045,184	1,502,790	1,276,566	2,940	19,273,955	683,679	2,578
1908....	.....	2,153,528	(d)1,253,624	820,834	2,703	17,154,940	782,859	1,560
1909....	.....	2,559,798	(d)1,658,291	1,178,219	19,338	19,924,622	884,703	2,767
1910....	£9,951	3,663,386	2,131,911	1,073,933	9,111	19,660,890	1,339,381	3,963
1911....	£7,782	4,212,986	2,177,961	1,574,439	1,820	19,378,540	1,772,840	3,314

Year.	Potassium Nitrate. Cwts.	Quick-silver.	Salt. Cwts.	Silver. (b)			Sulphur. Cwts.	Spelter, Sheets, Concentrates, Dross, etc. Cwts.
				Ore. Cwts.	Bullion. Kg.	Specie.		
1901.....	6,559	91.0	560,560	16,385	14.9	$772,020	99,270	14,291
1902.....	7,955	92.6	571,548	5,562	13.6	439,186	173,176	20,965
1903.....	4,659	87.5	312,681	.....	14.2	160,111	180,719	14,197
1904.....	7,812	92.6	355,599	.....	39.8	154,534	252,744	23,316
1905.....	9,010	82.1	492,727	.....	3908.0	261,397	177,304	26,211
1906.....	8,112	78.6	326,042	380	9756.4	703,820	269,704	24,233
1907.....	8,571	59.5	409,852	2	11.4	1,829,309	264,060	24,026
1908.....	6,036	56.4	390,535	.....	189.8	1,019,738	420,098	38,724
1909.....	6,894	58.1	234,092	.....	622.1	157,352	405,396	59,475
1910.....	7,016	57.0	444,081	14,609	1,412.0	1,615,775	357,332	70,658
1911.....	10,257	41.5	753,849	17,340	1,115.8	1,647,830	386,764	86,601

(a) From Trade and Customs Returns, Commonwealth of Australia. Previous to 1900 each Colony reported its own imports and exports. (b) Where gold or silver values are reported, £1 = $4.866. (c) Includes silver and other foils. (d) Includes ungalvanized corrugated. (e) Includes $19,000 in matte.

MINERAL EXPORTS OF AUSTRALIA. (a)

(In metric tons, cwts. of 112 lb., or dollars; £1 = $5.)

Year.	Alunite. Cwts.	Anti-mony Ore. Cwts.	Bis-muth Ore. Cwts.	Cement. Cwts.	Chrome Ore. Cwts.	Coal.	Coke.	Co-balt Ore. Cwts.	Copper.	
									Ore. Cwts.	Ingot and Matte. (Cwts.)
1901...	62,920	2,206	993	.....	41,035	1,750,066	4,465	2,212	231,644	389,041
1902...	72,880	1,428	136	.....	10,000	1,687,621	6,080	748	165,149	464,715
1903...	49,690	947	832	11,168	39,022	2,063,016	27,345	3,060	61,569	616,277
1904...	7,400	2,177	1,918	26,305	7,941	1,637,113	2,771	167	90,098	540,998
1905...	54,040	7,811	2,222	17,283	(c)	2,058,190	2,316	1,320	17,380	632,183
1906...	37,120	66,188	1,574	39,737	(c)	2,094,793	11,382	.....	33,476	744,357
1907...	41,750	74,440	653	75,600	54,503	2,689,917	35,063	.....	157,071	853,236
1908...	21,640	23,931	1,396	49,116	22,300	2,601,944	28,068	.....	103,694	765,298
1909...	73,795	14,976	1,763	23,585	.....	1,608,161	24,798	280	163,612	676,664
1910...	23,290	12,796	1,456	34,259	.....	1,730,375	10,457	186	209,611	769,745
1911...	20,272	17,985	2,307	45,247	.....	1,714,947	8,492	.....	263,078	806,005

Year.	Gold. (b)				Iron and steel bars, rods, etc. *Cwts.*	Lead.		
	Ore.	Bullion.	Specie.	Total Value.		Pig and matte. *Cwts.*	Argentiferous. *Cwts.*	Manufactures *Cwts.*
1901....	$65,341	$22,416,198	$43,233,515	$65,715,054	4,396	281,391	668,955	22,611
1902....	1,214,208	20,736,800	41,954,939	63,905,947	3,182	365,830	638,359	17,429
1903....	80,591	29,691,889	53,634,629	83,407,109	5,753	633,816	553,308	28,783
1904....	46,894	27,073,767	49,284,833	76,405,494	4,952	1,626,292	790,435	20,552
1905....	49,507	25,788,574	27,523,288	53,361,369	4,821	1,302,428	753,008	34,629
1906....	20,296	24,113,950	47,937,681	72,071,927	11,560	1,031,605	781,426	20,358
1907....	17,513	(c) 19,639,502	33,370,240	53,027,255	13,163	1,774,207	.....	21,765
1908....	20,539	(c) 18,924,716	50,794,544	69,739,799	13,015	2,132,199	.....	20,161
1909....	270,131	(c) 17,265,872	26,028,555	43,564,558	12,040	1,381,369	.....	14,067
1910....	429,288	(c) 11,970,906	10,598,747	22,998,940	17,106	1,800,856	845,856	13,350
1911....	327,101	10,456,322	47,831,267	58,614,700	117,131	1,755,650	430,073	14,610

Year.	Molybdenum Ore. *Cwts.*	Salt. *Cwts.*	Shale oil.	Silver.		Tin.		Spelter, Sheets, Concentrates, Dross, etc. *Cwts.*
				Ore. (d) *Cwts.*	Bullion. *Kg.*	Ore. *Cwts.*	Block. *Cwts.*	
1901.....	.....	156,760	19,587	1,630,252	196,136	5,012	60,129	1,732
1902.....	160	238,192	27,896	1,439,374	189,703	10,291	63,424	4,461
1903.....	783	155,613	14,483	1,653,794	202,730	26,900	82,473	60,206
1904.....	1,100	141,553	8,202	2,235,385	227,972	40,339	99,476	309,422
1905.....	1,381	174,987	11,818	581,651	208,134	55,153	108,963	3,006,372
1906.....	1,867	198,851	7,203	1,010,707	174,457	51,793	130,120	2,592,018
1907.....	2,025	189,194	5,686	907,790	294,679	65,005	131,407	5,393,784
1908.....	2,116	240,348	19,173	1,137,746	326,250	49,409	121,979	6,235,024
1909.....	1,055	230,486	3,983	1,914,479	90,989	8,559	111,262	6,994,745
1910.....	1,972	184,892	9,307	(f) 2,524,487	137,916	40,469	87,529	8,161,081
1911.....	1,876	63,607	6,344	2,795,970	(e)218,388	(f)55,721	82,935	8,648,818

(a) From "Trade and Customs Returns," Commonwealth of Australia.—Note. Previous to 1900 each Colony reported its own exports separately. (b) Where gold or silver values are reported, £1 = $4.866. (c) Included with iron ore. (d) Includes lead ore. (e) Includes that contained in matte. (f) Includes concentrates.

AUSTRIA-HUNGARY

In the following tables the mineral and metal productions of the two Kingdoms are reported separately, together with that of Bosnia and Herzegovina.

MINERAL AND METALLURGICAL PRODUCTION OF AUSTRIA. (a)
(In metric tons.)

Year.	Alum.	Alum and Pyritic Shale.	Antimony.		Asphaltic Rock.	Bismuth Ore.	Coal.	
			Ore.	Metal.			Bituminous.	Lignitic.
1898.....	1,037	28,914	679	343	643	*Nil*	10,947,522	21,083,361
1899.....	604	19,879	410	271	2,635	0.3	11,455,139	21,751,794
1900.....	620	3,004	201	153	887	4.0	10,992,515	21,539,917
1901.....	442	2,551	126	114	541	16.0	11,738,840	22,473,510
1902.....	62	2,866	18	24	897	8.0	11,045,039	22,139,683
1903.....	*Nil*	2,978	41	14	1,273	10.0	11,498,111	22,157,521
1904.....	*Nil*	2,337	103	36	1,435	1.7	11,868,245	21,987,651
1905.....	*Nil*	1,657	1,673	90	4,363	1.7	12,585,263	22,692,076
1906.....	*Nil*	1,020	1,071	*Nil*	2,840	2.7	13,473,307	24,167,714
1907.....	*Nil*	*Nil*	910	207	3,858	*Nil*	13,850,420	26,262,110
1908.....	*Nil*	*Nil*	193	162	3,695	*Nil*	13,875,382	26,728,926
1909.....	*Nil*	*Nil*	450	*Nil*	2,975	*Nil*	13,713,043	26,043,716
1910.....	*Nil*	*Nil*	225	*Nil*	1,066	*Nil*	14,056,858	25,157,541
1911.....	*Nil*	*Nil*	.....	.....	.....	.....	13,773,985	25,132,855

Year.	Copper.			Copperas.	Gold.		Graphite.	Iron.	
	Ore.	Metal.	Sulphate.		Ore.	Bullion.		Ore.	Pig & Cast
1897.....	7,405	1,083	276	125	647	$44,924	38,504	1,613,876	762,685
1898.....	6,791	1,041	209	360	448	47,515	33,062	1,733,649	837,767
1899.....	6,731	1,123	235	475	387	50,306	31,819	1,725,143	872,352
1900.....	5,825	881	234	474	227	47,183	33,663	1,894,458	879,132
1901.....	7,406	776	256	472	143	31,234	29,992	1,963,246	884,844
1902.....	8,455	914	248	271	74	4,652	29,527	1,742,498	991,827
1903.....	12,688	961	310	298	2,148	5,316	29,590	1,715,984	970,832
1904.....	10,701	889	808	414	12,653	47,183	28,620	1,719,219	988,364
1905.....	10,677	870	540	116	35,937	133,218	34,416	1,913,782	1,119,614
1906.....	20,255	877	578	154	33,033	83,401	38,117	2,253,662	1,222,230
1907.....	10,400	592	579	*Nil*	30,711	92,471	49,425	2,540,118	1,383,524
1908.....	8,381	683	556	*Nil*	28,907	98,504	44,425	2,632,407	1,466,897
1909.....	11,826	985	591	70	29,709	98,295	40,710	2,490,277	1,465,051
1910.....	8,005	1,468	715	*Nil*	31,744	118,895	33,131	2,627,513	1,504,786

Year.	Lead.			Manganese. Ore.	Mineral Paint.	Petroleum.	Quicksilver.		Salt.
	Ore.	Pig.	Litharge.				Ore.	Metal.	
1897.....	14,145	9,680	1,626	6,012	3,653	275,204	88,238	532	331,084
1898.....	14,363	10,340	1,520	6,132	3,213	323,142	88,519	491	341,959
1899.....	12,820	9,736	1,526	5,411	2,055	309,590	92,323	536	342,059
1900.....	14,314	10,650	1,288	8,804	2,828	347,213	94,747	510	330,277
1901.....	16,688	10,161	1,317	7,796	1,701	404,662	97,360	525	333,238
1902.....	19,055	11,264	1,023	5,646	1,486	520,845	90,040	511	311,806
1903.....	22,196	12,162	923	6,179	1,691	672,508	83,321	523	359,015
1904.....	22,514	12,645	783	10,189	1,829	.....	88,279	536	369,877
1905.....	23,339	12,968	865	13,788	798	.....	86,856	520	343,375
1906.....	19,683	14,846	1,059	13,402	943	.....	91,494	526	378,912
1907.....	22,792	13,598	863	16,756	1,091	.....	89,370	527	395,053
1908.....	21,513	12,669	1,010	16,656	475	.....	90,145	572	388,133
1909.....	20,550	12,941	840	18,045	620	.....	92,337	585	359,801
1910.....	22,841	15,476	602	15,694	698	1,766,018	100,899	603	345,629

Year.	Silver.		Sulphuric Acid.	Sulphur. Ore.	Tin.		Tungsten Ore.	Uranium.		Zinc.	
	Ore.	Bullion. (*Kg.*)			Ore.	Block.		Ore.	Salts.	Ore.	Spelter.
1897.	20,628	40,026	8,515	530	16	48	31	44	4.4	27,463	6,236
1898.	20,886	40,304	7,003	496	13	48	36	51	4.3	27,395	7,302
1899.	21,554	39,564	7,814	555	54	41	50	49	7.6	37,100	7,192
1900.	21,641	39,572	7,067	862	51	40	50	52	11.3	38,243	6,742
1901.	21,363	40,205	7,073	4,911	42	49	45	48	13.0	36,072	7,558
1902.	22,288	39,544	8,781	3,721	47	50	45	46	10.0	31,927	8,309
1903.	21,958	39,812	9,105	4,475	57	34	49	45	6.0	29,544	8,949
1904.	21,949	39,032	8,742	6,288	77	38	52	17	11.0	29,226	9,159
1905.	21,047	38,453	1,007	8,407	52	53	55	16	13.9	29,983	9,326
1906.	21,944	38,940	745	15,125	55	42	56	16	16.1	32,037	10,804
1907.	22,636	38,742	*Nil*	24,099	53	47	44	11	11.2	31,970	11,208
1908.	22,241	39,867	*Nil*	17,429	68	39	37	9	8.4	31,266	12,770
1909.	21,102	39,002	*Nil*	12,725	36	52	39	13	10.2	33,955	11,688
1910.	23,629	49,692	"	15,839	37	40	40	6.5	10.3	34,637	12,458

(*a*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.*

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*)

(In metric tons or dollars; 1 crown = $0.203.)

Year.	Antimony.		Asphalt	Asphaltic Rock.	Bismuth.	Carbon Bisulphide.	Coal.			
	Ore.	Regulus.					Bituminous. (*d*)	Lignite. (*d*)	Coke.	Briquets.
1897...	1,800	523	3,057	.....	4.7	432	1,118,024	3,870,530	(*c*)	27,022
1898...	2,201	855	3,125	.....	3.1	771	1,239,498	4,516,581	(*c*)	31,781
1899...	1,965	940	3,060	.....	3.0	1,120	1,238,855	4,292,584	10,336	31,137
1900...	2,373	846	2,700	.....	2.0	1,250	1,447,047	5,128,277	12,973	69,353
1901...	(*b*) 323	706	2,878	25,161	1.6	2,087	1,365,270	5,179,829	10,975	40,182
1902...	(*b*) 748	683	2,774	24,873	0.9	2,320	1,162,785	5,132,053	8,204	88,069
1903...	(*b*) 205	732	2,422	21,552	1.5	2,357	1,233,410	5,271,781	9,442	101,197
1904...	1,080	1,007	2,221	17,660	0.9	2,512	1,155,320	5,519,349	5,103	103,481
1905...	949	756	173	19,372	1.4	2,760	919,193	6,015,452	69,303	144,697
1906...	1,807	954	4,111	34,664	2.0	2,756	1,103,529	6,229,712	79,930	151,657
1907...	2,035	841	3,920	33,096	0.4	2,950	1,038,819	6,408,322	97,447	154,783
1908...	1,316	670	4,818	72,972	(*c*)	2,966	982,017	7,034,499	141,954	109,178
1909...	1,971	695	5,054	.....	.....	3,086	1,397,424	7,658,719	157,415	117,594
1910...	1,939	782	4,994	.....	.....	3,245	1,302,103	7,734,166	156,048	108,873

Year.	Copper.	Copperas.	Gold.	Iron.			Lead.		Litharge.	Manganese Ore.
				Ore. (*d*)	Pig.	Cast.	Ore.	Pig.		
1897...	213	592	$2,038,839	1,421,130	402,503	.....	525	2,527	155	4,030
1898...	153	745	1,839,474	1,666,837	448,621	20,784	771	2,305	188	8,087
1899...	165	771	2,039,504	1,587,600	451,637	19,631	526	2,166	213	5,073
1900...	181	700	2,173,079	1,666,363	432,817	22,738	612	2,030	201	5,746
1901...	162	805	2,189,692	1,557,300	430,686	20,640	(*b*) 10	2,029	238	4,591
1902...	89	909	2,260,135	1,562,238	416,835	18,569	(*b*) 20	2,244	219	7,237
1903...	45	982	2,243,521	1,439,132	396,674	18,875	(*d*)3,698	2,057	257	5,311
1904...	63	1,277	2,437,988	1,524,036	370,297	17,203	(*d*)3,922	2,104	710	11,527
1905...	73	920	2,439,451	1,661,358	403,719	17,563	686	2,146	209	5,708
1906...	69	1,306	2,487,156	1,698,291	402,527	17,164	564	1,925	698	7,176
1907...	85	1,212	2,330,292	(*e*)622,518	423,134	17,103	8	1,468	441	8,198
1908...	166	1,372	2,189,801	(*e*)727,019	505,559	17,415	3	1,544	190	10,601
1909...	265	1,414	1,813,024	1,965,487	514,853	15,577		1,590	625	11,921
1910...	214	1,313	2,022,032	1,905,749	487,421	14,635	91	2,077	570	13,270

Year.	Mineral Paints.	Petroleum.	Pyrites.	Quicksilver. Kg.	Salt.	Silver. Kg.	Sulphur.	Sulphuric Acid.	Zinc.	
									Ore.(b)	Spelter.
1898...	247	2,471	58,079	6,800	197,593	18,799	93	1,318	30	.....
1899...	394	2,125	79,519	27,000	200,525	20,991	116	1,463	1,197	.....
1900...	370	2,199	87,000	31,800	212,957	20,202	123	1,371	326	.....
1901...	305	3,296	93,907	33,003	215,581	23,636	137	1,464	693	14
1902...	283	4,347	106,490	44,600	217,079	23,020	105	1,193	364	.....
1903...	263	3,010	96,619	43,700	214,536	19,281	135	1,543	46	26
1904...	273	2,134	97,148	45,169	230,943	16,352	143	1,329	203	.....
1905...	196	471	106,848	36,000	238,642	15,946	135	1,410	173	.....
1906...	221	2,692	112,,623	50,100	245,402	13,642	133	1,457	243	146
1907...	259	2,404	99,503	40,400	(c)	12,695	(c)	1,232	(c)	(c)
1908...	294	2,427	95,824	78,000	(c)	12,612	144	1,444	135	(c)
1909...	63	2,590	98,971	71,500	231,182	10,932	131	1,307	(0)	.....
1910...	55	2,501	92,464	90,000	230,315	12,547	128	1,334	280	.....

(a) From the *Annuaire Statistique Hongrois*. (b) Includes only that part of the crude output that was not smelted into a refined product. (c) Not reported. (d) Total production. (e) Exported.

MINERAL AND METALLURGICAL PRODUCTION OF BOSNIA AND HERZEGOVINA. (a)
(In metric tons.)

Year.	Chrome Ore.	Copper		Iron.		Lignite.	Manganese Ore.	Pyrites.	Quicksilver.	Salt.
		Ore.	Metal.	Ore.	Pig.					
1899...	200	3,980	180	67,030	13,730	303,000	5,270	.....	3.3	15,030
1900...	100	3,008	141	133,454	38,960	394,516	7,939	1,710	6.7	15,791
1901...	505	3,696	199	122,569	39,296	445,007	6,346	4,570	9.3	16,865
1902...	270	3,657	166	133,348	43,992	424,753	5,760	5,170	7.2	17,348
1903...	147	1,073	191	114,059	39,833	467,962	4,538	6,589	8.1	18,459
1904...	279	640	115	127,297	47,678	483,617	1,114	10,412	8.1	18,021
1905...	186	670	39	122,540	43,074	540,237	4,129	19,045	10.0	(b)
1906...	320	765	25	136,513	45,660	594,172	7,651	11,347	5.1	22,671
1907...	310	245	*Nil.*	150,684	48,946	621,179	7,000	7,229	1.2	21,148
1908 (c)	315	(b)	*Nil.*	150,075	48,900	630,500	6,000	(b)	*Nil.*	22,100
1909 (c)	310	(b)	*Nil.*	145,200	48,850	675,400	5,000	(b)	*Nil.*	22,500
1910...	320	(b)	*Nil.*	132,720	48,841	706,659	4,000	571	*Nil.*	23,579
1911...	250	(b)	*Nil.*	140,283	44,354	769,763	3,600	3,118	*Nil.*	22,591

(a) From *Oestr. Zeit. f. B.-u. H.* (b) Not reported. (c) Estimated.

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (a)
(In metric tons or dollars; 5 crowns = $1.)

Year.	Alum.	Aluminum Sulphate and Chloride.	Antimony.		Arsenic, Metallic, Oxide, Orpiment and Realgar.	Asbestos.		Asphalt.	
			Ore.	Regulus.		Crude.	Manufactured.	Rock and Earth.	Mastic and Bitumen.
1898....	83	253	266	679	29	150	28	183	2,126
1899....	54	233	562	240	47	71	60	1,143	2,619
1900....	44	164	247	276	65	47	168	1,218	2,177
1901....	55	211	179	385	80	36	165	198	1,909
1902....	102	135	174	290	89	65	275	520	301
1903....	77	14	128	249	63	89	495	921	483
1904....	38	2	200	673	72	290	1,582	403	728
1905....	68	34	178	774	42	330	1,397	1,060	457
1906 (f)..	68	80	314	912	66	376	1,708	2,824	799
1907....	75	81	92	698	59	351	630	3,787	771
1908....	147	92	161	527	51	442	450	1,312	1,030
1909....	65	57	97	185	117	264	561	1,800	758
1910....	346	85	54	416	9	262	604	1,545	954
1911....	157	16	77	267	13	141	992	1,845	1,218

Year.	Barium.		Chloride of Lime.	Cement.	Chrome Ore.	Kaolin and Feldspar.	Coal.		Coke.
	Sulphate (b)	Chloride.					Bituminous.	Lignitic.	
1898	.....	.....	113	23,989	121	74,003	824,730	8,351,955	194,289
1899	65	.....	203	38,193	53	78,537	879,337	8,662,788	252,971
1900	23	.....	192	46,761	22	103,178	815,097	7,864,410	262,793
1901	55	4,008	738	44,723	62	97,037	748,802	8,076,575	303,651
1902	64	4,552	426	39,920	51	100,546	691,680	7,888,218	234,911
1903	52	5,091	674	40,239	100	110,181	754,957	8,027,347	280,395
1904	74	4,233	254	43,110	36	127,984	815,570	7,588,555	353,695
1905	26	4,626	978	52,830	46	137,125	*903,156	8,035,718	287,790
1906(f)	2,395	4,503	271	64,883	102	133,326	750,420	7,150,339	246,914
1907	3,119	5,220	308	81,407	161	157,894	849,792	8,876,408	323,243
1908	2,987	2,974	519	65,597	144	154,146	762,867	8,600,683	183,279
1909	2,585	3,737	462	63,463	174	150,878	633,253	8,241,723	198,313
1910	13	4,344	1,213	98,888	140	164,859	615,082	7,492,447	230,735
1911	12	3,741	1,175	168,321	70	174,493	609,737	7,063,981	299,915

Year.	Fluorspar.	Copper.			Copper Sulphate.	Copperas.	Cryolite.
		Ore.	Crude and Old.	Bars, Sheets Plates, etc.			
1898	22	12	173	266	29	539	23
1899	309	74	534	298	67	808	101
1900	45	801	471	200	57	748	237
1901	6	1,042	435	334	23	548	231
1902	42	1,018	436	381	44	857	363
1903	12	1,308	1,226	451	45	898	521
1904	36	574	747	577	50	1,170	574
1905	5	2,328	1,253	746	49	836	638
1906(f)	*Nil.*	341	1,007	816	99	861	*Nil.*
1907	*Nil.*	489	624	870	11	1,580	*Nil.*
1908	7	206	1,126	791	63	2,199	*Nil.*
1909	12	136	1,250	541	40	1,877	*Nil.*
1910	4	101	977	700	83	2,273	42
1911	0.2	126	1,309	1,109	14	2,124	122

Year.	Gold.			Graphite.	Gypsum.		Hydrochloric Acid.
	Ore.	Bullion. (e)	Specie. (e)		Crude.	Calcined.	
1898	13	17,943	$23,779,858	17,109	718	2,163	1,614
1899	67	17,864	12,711,454	19,451	634	1,539	1,495
1900	1	120,988	11,582,571	18,995	502	1,723	1,659
1901	0	42,427	6,880,888	14,900	461	1,206	1,632
1902	.....	22,939	13,485,087	16,771	550	1,041	791
1903	3	10,150	11,052,944	17,302	342	1,510	3,530
1904	64	5,278	9,649,605	17,430	392	1,510	3,722
1905	1,059	9,338	10,995,089	18,535	363	1,652	4,085
1906 (f)	936	88,264	8,015,967	16,871	1,970	686	2,942
1907	996	1,234,291	13,061,517	21,704	3,841	801	3,708
1908	613	70,005	11,978,828	16,535	7,241	807	3,720
1909	548	3,426,539	19,470,382	18,484	3,675	809	3,263
1910	(h)	1,750,600	10,891,800	21,191	5,110	590	4,856
1911	(h)	6,909,408	15,220,358	21,204	3,689	914	5,834

(h) Values in gold $40,200 and $38,760.

Year.	Iron.				Lime.	Magnesium Chloride and Glauber Salts.	Magnesite (Calcined).
	Ore.	Pig and Old.	Manufactures.	Iron and Steel Bars, Sheets, Wire, etc.			
1898 ...	362,317	15,803	22,724	23,231	89,067	7,248	(c)
1899 ...	326,951	27,738	30,822	50,197	85,570	5,721	(c)
1900 ...	263,421	53,426	40,344	65,019	86,273	7,321	(c)
1901 ...	229,624	26,304	46,508	28,841	82,399	7,960	40,236
1902 ...	241,806	42,592	30,137	45,517	81,634	5,333	53,467
1903 ...	252,520	60,237	40,807	63,031	95,644	2,360	69,058
1904 ...	295,017	66,442	60,252	64,698	101,753	2,151	53,781
1905 ...	373,077	63,780	63,828	69,672	94,751	1,272	92,359
1906(*f*)	234,924	43,694	73,575	50,247	87,468	4,094	87,765
1907 ...	220,767	37,581	56,472	69,669	89,305	6,905	113,695
1908 ...	220,357	17,494	32,110	29,227	62,938	2,622	87,049
1909 ...	178,464	22,627	25,455	40,837	77,586	5,446	125,666
1910 ...	144,827	96,114	73,473	100,541	391,989	6,332	182,911
1911 ...	114,166	87,975	60,918	75,630	511,996	2,715	147,481

Year.	Lead.						Manganese ore.	Millstones.	Mineral Paints.	Nickle and Cobalt ores.
	Ore.	Dross.	Litharge	Metal and Alloys.	Red and Yellow.	White.				
1898....	2,253	100	188	545	45	55	1,961	2,109	2,153	121
1899....	2,502	99	188	258	45	41	1,127	1,904	2,061	75
1900....	2,628	66	242	393	34	34	463	1,871	1,906	114
1901....	4,143	112	179	68	32	23	398	1,971	1,947	120
1902....	5,478	154	124	109	25	37	411	1,886	2,136	34
1903....	8,961	147	145	152	19	25	724	2,311	1,873	12
1904....	7,575	144	167	464	54	52	1,234	2,276	1,840	26
1905....	7,944	342	141	957	60	39	995	2,232	2,091	16
1906(*f*)	4,891	223	302	602	16	52	4,170	1,763	1,367	42
1907....	8,360	420	255	197	9	54	5,273	2,422	1,697	29
1908....	7,107	488	312	199	22	50	2,109	3,293	2,292	*Nil.*
1909....	5,711	294	343	306	125	80	787	3,849	2,577	*Nil.*
1910....	6,163	322	160	418	44	510	285	3,970	3,542	13
1911....	5,493	154	768	670	24	35	588	3,513	3,426	1

Year.	Nitric Acid.	Ozokerite.	Peat and Peat Coke.	Petroleum. (*d*)	Benzine.	Paraffin.	Potash.	Potassium Chloride.	Pyrites.	Sulphur.
1898....	294	4,462	3,400	4,138	.....	.....	7,252	994	3,039	923
1899....	420	5,441	4,010	11,756	20,646	10	10,113	974	5,201	885
1900....	519	5,162	5,607	33,032	18,361	26	7,792	879	17,162	1,285
1901....	632	2,717	4,558	19,804	17,021	14	4,234	909	16,491	1,225
1902....	769	2,285	4,927	40,683	13,884	24	3,229	772	9,547	1,136
1903....	908	2,258	3,638	74,454	14,000	1,153	3,409	802	10,857	1,123
1904....	858	2,093	3,980	122,419	13,706	5,992	4,604	445	9,891	988
1905....	1,377	1,614	3,746	200,736	8,187	8,996	5,511	1,048	9,168	859
1906(*f*)	1,303	2,034	2,517	198,325	13,472	9,996	3,814	1,005	7,208	760
1907....	754	1,813	4,001	212,527	12,638	14,758	5,864	1,280	5,646	784
1908....	882	1,648	4,416	351,262	25,599	28,808	4,697	776	6,286	998
1909....	745	2,321	3,395	470,085	32,532	38,043	5,667	793	4,975	1,295
1910....	1,121	2,585	1,963	271,467	39,452	44,432	5,271	957	4,565	658
1911....	1,047	1,859	2,869	220,499	41,923	38,058	4,801	943	1,190	534

Year.	Sulphuric Acid.	Tin.			Whetstones.	Zinc.				
		Ingot and Alloys.	Bars, Plates, Sheets, etc.	Dross.		Ore.	Metallic and Alloys.	Sheets, etc.	White.	Dross.
1898...	9,880	96	72	324	2,316	14,065	1,184	757	1,240	298
1899...	12,422	167	77	273	2,215	20,461	1,614	1,313	1,096	73
1900...	12,693	153	102	208	2,270	20,379	1,088	502	1,719	149
1901...	10,373	162	109	257	2,359	23,150	1,374	813	2,720	167
1902...	9,451	193	128	188	2,852	24,519	2,002	1,127	3,113	237
1903...	8,369	292	111	158	2,569	15,108	4,420	729	3,446	267
1904...	9,101	126	102	123	2,159	17,314	4,606	532	3,666	158
1905...	12,823	197	94	78	2,355	19,602	5,023	498	3,861	113
1906(f)	10,493	221	62	83	1,541	15,933	4,578	323	3,504	4,295
1907...	15,190	333	84	160	1,900	19,516	4,608	585	4,873	4,873
1908...	13,581	257	49	172	2,009	19,233	6,604	880	4,131	4,131
1909...	11,232	558	82	59	1,677	19,970	7,031	867	5,078	5,078
1910...	12,016	509	58	70	2,411	22,397	8,720	303	4,968	(g)
1911...	12,111	759	77	59	1,769	16,744	710	411	4,639	(g)

(a) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (b) Includes artificial barium sulphate. (c) Previous to 1901, magnesite was included with other minerals not elsewhere specified. (d) From 1895 to 1898 inclusive, includes crude and refined petroleum; from 1899 to 1905 inclusive, lubricating oil is also included. (e) Where gold or silver values are reported 1 crown = $0.203. (f) Last 10 months only. (g) Not reported.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (a)
(In metric tons or dollars; 5 Crowns = $1.)

Year.	Alum.	Aluminum and Alloys.	Aluminum Sulphate and Chloride.	Antimony.		Arsenic. (b)	Asbestos.	
				Ore.	Regulus—Kg.		Crude. (c)	Manufactures.
1898.....	338	101	1,822	12	28,200	287	609	138
1899.....	332	121	1,299	10	30,400	284	866	1,347
1900.....	430	154	1,435	46	23,000	320	1,085	1,238
1901.....	413	153	1,882	27	1,500	351	1,678	1,032
1902.....	537	151	2,161	40	18,200	351	2,038	798
1903.....	508	150	2,670	42	87,200	371	3,395	1,221
1904.....	602	231	2,346	64	21,000	384	2,517	1,240
1905.....	774	477	2,775	42	24,700	342	5,962	208
1906(f)..	513	216	2,840	79	38,900	83	7,025	111
1907.....	545	255	3,200	231	89,900	325	5,729	180
1908.....	567	323	2,606	304	128,200	349	9,484	139
1909.....	359	418	1,831	95	80,100	370	12,003	200
1910.....	589	303	1,474	96	121,800	416	11,602	250
1911.....	375	516	1,317	98	124,600	377	16,071	276

Year.	Asphalt.		Barytes.	Borax.		Cement.	Chloride of Lime.	Chrome Ore.
	Crude Rock	Mastic and Bitumen.		Crude and Boric Acid.	Refined.			
1898.....	5,973	1,117	5,012	784	185	30,745	2,851	2,206
1899.....	7,301	1,546	5,443	2,212	130	21,410	3,749	1,874
1900.....	8,301	1,564	5,945	3,056	93	25,747	3,326	2,823
1901.....	5,702	1,106	6,336	1,687	233	23,559	3,326	860
1902.....	5,732	1,273	6,266	2,168	174	18,658	2,596	2,868
1903.....	5,871	1,272	7,057	2,192	150	23,256	2,791	2,121
1904.....	8,211	1,064	6,238	2,752	142	20,259	3,407	1,209
1905.....	8,553	1,139	6,187	3,099	205	21,950	1,847	2,305
1906(f).....	13,381	895	9,654	3,519	126	21,833	2,491	1,612
1907.....	9,394	1,637	11,669	3,763	138	23,697	2,534	2,795
1908.....	11,678	1,305	11,241	4,105	158	39,135	2,395	1,837
1909.....	9,113	1,309	13,940	3,240	139	53,262	3,105	2,223
1910.....	15,897	1,101	21,486	5,284	166	81,251	1,885	1,271
1911.....	18,394	1,990	20,501	4,407	193	80,613	3,131	2,217

Year.	Kaolin and Feldspar.	Coal.		Coke.	Copper.			Copper Sulphate.
		Bituminous.	Lignitic.		Ore.	Bullion and Scrap.	Bars, Sheets, Wire, etc.	
1898.....	7,991	5,396,760	19,393	606,783	64	17,443	159	5,271
1899.....	8,152	5,296,700	20,879	564,005	*Nil*	16,185	156	2,345
1900.....	6,847	6,242,939	67,740	620,776	16	18,970	121	3,516
1901.....	7,687	5,827,332	22,253	612,209	112	17,504	83	2,822
1902.....	9,085	5,766,377	29,601	547,406	100	18,498	149	2,839
1903.....	9,940	5,907,660	30,007	519,281	209	18,701	89	3,526
1904.....	10,854	6,190,030	30,001	548,272	1,107	22,532	89	4,508
1905.....	13,656	•6,418,042	36,000	554,147	1,397	22,652	73	3,791
1906 (*f*)...	13,219	5,942,897	17,464	406,088	267	20,943	481	1,597
1907.....	17,961	9,692,645	23,699	677,750	44	26,181	818	3,981
1908.....	17,417	9,995,415	30,433	851,099	121	33,270	1,140	8,402
1909.....	16,185	10,482,264	38,679	701,281	13	30,606	568	4,129
1910.....	17,817	9,864,462	37,867	670,089	49	32,217	926	3,445
1911.....	22,828	10,873,799	34,381	702,707	716	37,251	538	6,598

Year.	Cop-peras.	Cryolite.	Fluor-spar.	Gold. (*d*)		Graph-ite.	Gypsum.		Hydro-chloric Acid.
				Bullion.	Coin.		Crude.	Burned	
1898.....	466	275	4,169	$323,636	8,853,354	1,109	991	13,300	766
1899.....	409	343	4,959	432,187	7,662,641	815	1,336	13,441	350
1900.....	343	342	5,649	1,111,831	7,230,251	302	1,348	15,462	577
1901.....	269	428	5,774	13,865,103	20,353,592	318	1,405	15,830	576
1902.....	274	447	5,902	14,509.019	15,695,960	221	1,588	16,430	588
1903.....	155	521	5,445	9,825,200	9,817,283	405	1,969	18,655	603
1904.....	238	313	7,061	12,703,740	8,586,394	423	2,384	19,387	459
1905.....	169	220	7,601	1,047,792	9,204,968	735	1,553	21,286	656
1906(*f*)..	186	217	7.795	989,604	5,229,591	854	4,104	10,308	476
1907.....	187	613	8,779	1,106,002	5,755,918	934	5,813	11,981	629
1908.....	74	564	7,359	7,303,982	7,969,538	755	4,993	10,842	924
1909.....	56	556	8,106	36,744,037	8,631,453	660	4,907	10,410	898
1910.....	89	703	8,292	1,399,400	4,485,600	1,124	5,733	20,390	635
1911.....	66	944	9,446	3,163,554	1,902,798	1,173	6,616	16,489	612

Year.	Iron.				Lead.				
	Ore.	Pig and Old.	Manufac-tures.	Iron and Steel Bars Sheets Wire etc.	Ore.	Pig, Alloys, Crude.	Lith-arge.	Red and Yellow.	White.
1898.....	178,507	173,919	$4,627,200	26,421	459	9,746	280	555	115
1899.....	212,412	126,371	4,395,356	12,340	465	8,836	224	466	80
1900.....	233,156	95,530	4,533,599	10,313	501	7,916	141	354	106
1901.....	218,476	90,287	4,443,670	10,902	1,270	10,722	189	433	135
1902.....	197,525	43,314	4,304,818	11,584	1,879	8,706	149	428	221
1903.....	217,979	47,354	4,508,224	11,025	1,355	9,190	141	423	173
1904.....	182,515	35,091	4,976,342	9,402	1,436	7,917	146	372	138
1905.....	228,149	49,383	5,722,976	33,256	247	7,282	101	349	88
1906(*f*).....	232,558	57,341	6,153,698	5,085	189	11,732	82	310	75
1907.....	390,322	151,848	(*g*)29,520	29,268	204	12,547	98	381	126
1908.....	423,940	224,740	(*g*)45,235	88,396	559	17,116	161	616	201
1909.....	374,088	147,459	(*g*)48,344	34,025	7,560	20,697	230	599	57
1910.....	307,190	185,604	51,869	44,277	6,354	17,065	411	574	139
1911.....	469,552	112,907	53,590	32,687	1,200	17,238	299	622	108

Year.	Magnesium Chloride.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel, Ol and Crud
1898.....	2,096	5,396	1,429	4,979	137
1899.....	2,043	5,855	1,458	5,106	119
1900.....	2,100	7,016	1,672	4,958	258
1901.....	2,529	6,367	1,595	5,109	277
1902.....	2,621	15,595	1,410	4,831	265
1903.....	3,118	38,529	1,395	4,733	268
1904.....	2,997	35,357	1,282	5,563	402
1905.....	3,495	30,483	1,467	6,018	632
1906(*f*)...	4,050	33,406	1,176	4,660	773
1907.....	5,006	70,067	1,469	6,043	1,192
1908.....	5,011	31,023	1,474	5,909	1,521
1909.....	6,095	44,970	1,357	5,773	2,116
1910.....	6,616	59,951	1,510	7,340	1,606
1911.....	7,171	78,790	1,494	7,157	1,598

Year.	Petroleum Products.			Phosphorus and Phosphoric Acid.	
	Crude oil.	Refined Oil.	Paraffin.		Carbor
1898.....	58,580	22,299	.....	209	30
1899.....	75,885	21,823	6,968	221	52
1900.....	20,813	22,963	5,080	204	1,02
1901.....	22,545	18,067	5,294	222	1,44
1902.....	24,830	15,864	4,238	225	48
1903.....	19,710	19,382	2,598	237	19
1904.....	20,110	22,715	1,470	193	22
1905.....	18,974	24,961	888	222	15
1906(*f*).....	13,522	9,693	403	178	60
1907.....	18,345	11,441	524	219	17
1908.....	3,118	9,705	358	234	32
1909.....	1,357	9,093	508	231	31
1910.....	18,977	16,868	457	253	18
1911.....	19,070	19,737	632	233	20

Year.	Quicksilver *Kg.*	Salt.	Sand.	Silve
				Bullion, *Kg.*
1898.....	6,300	41,870	70,296	15,400
1899.....	2,600	37,883	71,279	28,900
1900.....	1,300	39,822	77,930	29,300
1901.....	2,600	39,625	83,401	41,800
1902.....	1,300	46,128	92,617	177,900
1903.....	1,600	48,793	94,492	150,400
1904.....	2,500	94,103	97,364	36,700
1905.....	2,400	.....	104,195	36,100
1906(*f*)	1,800	32,182	134,526	43,000
1907.....	1,200	41,660	180,280	88,182
1908.....	2,000	54,525	177,529	96,700
1909.....	2,600	75,397	183,169	150,980
1910.....	2,300	96,936	188,762	161,600
1911.....	2,900	99,921	225,743	195,000

Year.	Sodium salts.						Sulphur.
	Bi-sulphate.	Carbonate.	Carbonate (Calcined).	Hydrate.	Nitrate.	Sulphate.	
1898 ..	89	53	2,408	1,498	41,773	4,476	20,655
1899 ..	85	62	1,123	1,669	47,301	5,394	23,504
1900 ..	73	104	1,141	1,836	54,559	5,110	27,795
1901 ..	98	77	911	1,280	63,283	4,452	25,300
1902 ..	17	97	312	1,030	39,958	5,997	23,878
1903 ..	13	110	327	956	54,896	6,116	22,625
1904 ..	103	103	1,109	659	54,887	5,409	30,505
1905 ..	167	168	965	475	66,740	5,258	30,227
1906(f)	86	382	303	218	49,862	7,508	26,755
1907 ..	72	153	283	305	57,023	7,342	34,261
1908 ..	32	124	288	358	69,722	7,596	30,985
1909 ..	26	149	297	832	80,533	9,402	28,639
1910 ..	15	224	354	341	81,781	10,150	34,936
1911 ..	5	297	337	297	65,781	10,842	34,916

Year.	Sulphuric Acid.	Tin. Ingot, Crude, Old, etc.	Whet-stones.	Zinc.			
				Calamine and Other Ores.	Spelter.	Bars, Sheets, Wires, etc.	White.
1898.....	9,724	3,769	3,490	14,112	17,471	453	697
1899.....	10,245	3,005	3,717	12,730	15,225	481	750
1900.....	10,643	3,439	3,643	14,181	17,844	667	875
1901.....	11,712	3,671	3,445	18,403	16,921	579	718
1902.....	12,474	3,638	3,599	20,723	17,034	651	636
1903.....	16,148	3,564	3,774	22,344	17,973	746	698
1904.....	19,878	3,528	4,272	24,039	20,787	731	840
1905.....	17,320	3,845	4,376	22,890	21,874	568	972
1906(f).....	17,020	3,320	4,377	24,014	19,467	595	347
1907.....	20,430	4,433	5,552	24,289	24,092	604	219
1908.....	25,251	4,295	5,916	19,366	26,472	1,010	361
1909.....	27,368	4,587	6,826	24,639	27,148	1,263	349
1910.....	27,421	4,674	6,915	30,880	29,168	1,898	511
1911.....	26,630	4,795	8,066	31,357	33,740	388	471

(a) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (b) Includes arsenious acid and sulphide. (c) Includes burned asbestos. (d) The valves of gold are figured at the rate of one crown = $0.203. (e) Potassium and Sodium. (f) Last 10 months only. (g) Metric tons.

BELGIUM

The mining and metallurgical production in Belgium, and the imports and exports, according to the latest official statistics, are as follows:

MINERAL, METALLURGICAL AND QUARRY PRODUCTION OF BELGIUM. (a)

(In metric tons except where otherwise noted.)

Year.	Barytes.	Chalk-marl. *Cubic Meters.*	Coal.		Coke.	Flint. *Cubic Meters.*	Iron Ore.
			Bituminous.	Briquets.		For Earthenware.	
1898....	21,700	287,805	22,088,335	1,351,884	2,161,162	22,150	217,370
1899....	25,900	351,800	22,072,068	1,276,050	2,304,607	25,185	201,445
1900....	38,800	377,550	23,462,817	1,395,910	2,434,678	25,700	247,890
1901....	22,800	449,000	22,213,410	1,587,800	1,847,780	17,700	218,780
1902....	33,000	390,700	22,877,470	1,616,520	2,048,070	17,430	166,480
1903....	21,000	501,920	23,796,680	1,686,415	2,203,020	16,250	184,400
1904....	60,000	450,400	22,761,430	1,735,480	2,211,820	18,070	206,730
1905....	26,000	372,000	21,775,280	1,711,920	2,238,920	12,800	176,940
1906....	22,365	568,170	23,569,860	1,887,090	2,414,490	14,900	232,570
1907....	23,000	478,880	23,705,190	2,040,670	2,473,790	15,050	316,250
1908....	25,070	289,690	23,557,900	2,341,210	2,307,990	15,430	188,780
1909....	4,900	300,690	23,517,550	2,707,390	2,972,920	15,280	199,710
1910....	31,400	307,390	23,919,090	2,651,190	3,110,820	16,400	122,960
1911....	25,200	1,667,125	23,053,540	2,778,620	3,160,950	10,950	150,500

Year.	Iron, crude.					Iron, manufactures of.			
	Forge Pig.	Foundry Pig.	Bessemer Pig.	Basic Pig.	Total Pig.	Merchant Bars.	Sheet and Plate.	Wrought.	Other Mfres.
1898....	308,875	93,645	173,085	397,891	979,755	123,993	91,686	993	267,521
1899....	317,029	84,165	169,664	453,718	1,024,576	93,601	97,604	662	283,331
1900....	305,344	88,335	176,557	447,271	1,018,561	61,458	73,572	1,411	284,591
1901....	178,250	86,170	166,820	332,940	764,180	249,380	65,760	550	64,900
1902....	104,540	254,710	199,170	510,630	1,069,050	260,290	62,740	450	58,150
1903....	91,600	256,890	229,160	638,430	1,216,080	274,520	56,550	390	60,920
1904....	99,350	224,410	217,390	742,040	1,287,597	246,240	41,000	370	67,580
1905....	98,170	206,390	220,210	784,850	1,311,120	270,840	39,250	40	67,490
1906....	96,090	218,225	177,900	870,860	1,375,775	265,010	37,540	20	55,680
1907....	92,280	189,190	88,650	1,008,170	1,406,980	274,400	37,950	20	46,130
1908....	116,740	76,290	78,950	996,870	1,270,050	239,670	30,130	20	36,830
1909....	127,080	91,040	56,430	1,340,060	1,616,370	235,020	28,260	20	(c)41,610
1910....	116,740	76,290	78,950	996,870	1,270,050	225,220	27,520	20	(c)46,740
1911....	110,350	52,970	46,240	1,836,720	2,046,280	223,830	30,980	.....	35,460

Year.	Steel.					Lead.		Manganese Ore.
	Ingots, Blooms and Billets.	Rails.	Tires.	Wrought.	Plates.	Ore.	Pig. (*a*)	
1898...	653,523	117,751	10,953	17,902	87,219	133	19,330	16,440
1899...	731,249	123,119	11,212	32,180	68,051	137	15,727	12,120
1900...	655,199	134,428	11,934	25,985	55,307	230	16,365	10,820
1901...	529,840	132,260	12,380	3,310	83,810	220	18,760	8,510
1902...	786,980	(*c*)268,220	12,790	2,910	94,360	164	73,357	14,440
1903...	988,160	(*c*)351,540	17,810	2,920	118,200	90	68,700	6,100
1904...	1,065,870	(*c*)266,900	23,540	4,300	149,270	91	23,470	485
1905...	1,227,110	241,640	25,810	6,080	179,470	126	22,885	*Nil.*
1906...	1,440,860	274,920	32,070	5,070	186,610	121	23,765	120
1907...	1,521,610	314,760	34,700	5,190	157,850	210	27,450	2,100
1908...	1,198,000	(*c*)191,370	29,000	2,870	199,050	195	37,325	7,130
1909...	1,632,394	214,000	33,960	2,330	254,650	150	40,306	6,270
1910 ..	1,944,820	347,890	31,860	2,410	272,420	162	40,715	*Nil.*
1911...	2,192,630	337,520	35,450	6,320	428,670	82	44,308	*Nil.*

Year.	Mineral Paints.	Phosphate of Lime.	Pyrites.	Slate. *Pieces.*	Silver. *Kg.*	Zinc.			
	Ochers.					Ore. (Blende)	Ore. (Calamine)	Spelter.	Sheets.
1897	(*b*) 350	(*b*)350,056	1,828	41,422,000	30,073	6,804	4,150	116,067	37,011
1898	(*b*) 290	(*b*)156,920	147	42,311,000	116,035	7,350	4,125	119,671	35,587
1899	(*b*) 300	(*b*)190,090	283	44,167,000	134,854	5,736	3,730	122,843	34,289
1900	(*b*) 300	(*b*)215,670	400	43,941,000	146,548	5,715	3,000	119,317	38,825
1901	2,100	222,520	560	39,030,000	169,450	4,445	2,200	127,170	37,380
1902	200	135,850	710	37,120,000	212,249	3,568	284	124,780	37,070
1903	200	184,120	720	38,953,000	232,740	3,565	65	131,740	42,280
1904	450	202,480	1,075	41,240,000	252,920	3,698	4	137,323	41,490
1905	300	193,305	976	41,435,000	201,935	3,929	*Nil.*	142,555	45,320
1906	250	152,140	908	43,801,000	173,535	3,858	*Nil.*	148,035	44,525
1907	200	181,230	397	40,102,000	178,020	3,485	5	152,370	45,330
1908	.....	198,030	357	41,180,000	228,000	2,099	3	161,940	43,410
1909	700	205,260	214	41,630,000	271,270	1,229	*Nil.*	174,490	44,850
1910	600	202,880	214	36,440,000	264,655	1,434	*Nil.*	181,745	47,970
1911	540	196,780	122	37,620,000	252,720	836	*Nil.*	198,230	48,450

(*a*) From *Statistique des Industries Extractives et Metallurgiques et des Appareils à vapeur en Belgique.*
(*b*) Cubic meters. (*c*) Includes beams. (*d*) Not including 48,595 tons imported silver lead treated.
(*e*) Includes corrugated iron.

CANADA

The statistics of mineral production in the Dominion of Canada as reported by the Geological Survey are summarized in the following tables. The statement of imports and exports for 1907 is for the nine months ending March 31, in consequence of a change in the law whereby the fiscal year was changed from June 30:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (a)

(In metric tons or dollars.)

Year.	Arsenic.	Asbestos and Asbestic	Barytes.	Cement—*Barrels.*		Chromite.	Coal.	Cobalt.	Coke.
				Natural Rock.	Portland.				
1898....	*Nil.*	21,577	971	87,125	163,084	1,833	3,784,532	.....	79,453
1899....	52	22,938	653	131,387	255,366	1,796	4,467,021	.....	91,444
1900....	275	27,797	1,213	125,428	292,124	2,335	5,087,060	.....	142,521
1901....	630	36,477	592	133,328	317,066	1,274	5,648,208	.....	331,537
1902....	726	36,657	994	127,931	594,594	900	6,524,180	.....	455,353
1903....	725	37,902	1,055	92,252	627,741	3,509	6,933,107	.....	509,115
1904....	(*d*) 66	44,131	1,253	51,555	771,650	5,511	6,812,834	.....	493,107
1905....	*Nil.*	61,928	3,049	14,184	1,346,547	7,781	7,961,397	.....	622,154
1906....	*Nil.*	72,025	3,628	8,610	2,139,164	7,936	9,033,973	.....	(*b*)
1907....	317	82,117	1,829	5,775	2,368,593	6,527	9,533,442	.....	(*b*)
1908....	634	82,448	3,911	1,044	2,665,289	7,341	9,857,754	841	784,788
1909....	1,024	79,197	(*b*)	(*b*)	4,067,709	2,240	9,526,938	(*i*)	782,016
1910....	1,362	92,800	(*b*)	(*b*)	4,753,975	276	11,711,000	(*i*)	818,940
1911....	1,902	115,136	45	(*b*)	5,635,950	24	10,243,528	(*i*)	768,763

Year.	Copper. (In Ore, etc.)	Corundum.	Feldspar.	Gold. (*c*)	Graphite.	Grindstones.	Gypsum.	Iron Ore.	Iron, Pig. All kinds.
1898..	8,048	(*b*)	2,268	$13,775,420	1,107	4,476	198,864	52,917	69,853
1899..	6,838	(*b*)	2,721	21,261,584	1,188	4,091	221,821	67,678	93,267
1900..	8,588	3	288	27,908,153	1,743	5,024	228,656	110,654	87,394
1901..	17,155	403	4,852	24,128,503	2,004	4,155	266,476	284,477	248,830
1902..	17,598	697	6,871	21,336,667	993	5,835	301,165	366,431	324,617
1903..	19,357	880	12,633	18,843,590	660	5,023	285,242	239,715	270,182
1904..	19,497	834	10,057	16,400,000	410	4,091	309,133	317,387	275,367
1905..	21,596	1,492	10,617	14,486,833	491	4,693	395,341	263,113	475,491
1906..	25,863	2,063	14,397	12,023,932	405	5,029	378,904	269,842	550,625
1907..	26,025	1,716	11,414	8,264,765	525	4,881	431,286	.....	590,444
1908..	28,895	988	7,144	9,842,105	227	3,285	309,254	184,565	572,284
1909..	23,811	1,353	11,594	9,382,230	783	3,877	429,218	(*d*) 19,925	(*g*)686,858
1910..	25,262	1,696	14,114	10,205,835	1,263	3,604	476,496	(*d*) 103,827	(*g*)726,471
1911..	25,333	1,336	16,074	9,762,096	1,151	4,818	458,550	35,528	832,290

Year.	Iron and Steel, Rolled.	Lead. (In ore, etc.)	Manganese Ore.	Mica.	Mineral Paints. (Ochers.)	Natural Gas.	Nickel. (In ore, etc.)	Petroleum, Crude. *Barrels.* (*e*)
1898....	101,748	14,469	45	$118,375	2,019	322,123	2,502	758,381
1899....	112,412	9,914	1,434	163,000	3,555	387,271	2,605	808,570
1900....	102,301	28,648	27	166,000	1,783	417,094	3,211	710,498
1901....	113,799	23,537	(*d*)399	160,000	2,025	339,476	4,167	623,392
1902....	164,069	10,411	(*d*)156	135,904	4,494	195,992	4,849	530,624
1903....	131,588	8,226	83	177,857	5,683	202,210	5,671	486,637
1904....	(*b*)	17,241	(*d*)112	152,919	3,562	247,370	4,786	552,575
1905....	(*b*)	25,391	(*d*) 20	168,170	4,632	314,249	8,565	634,095
1906....	(*b*)	24,580	(*d*) 84	(*d*)581,043	6,201	528,868	9,745	569,753
1907....	(*b*)	21,570	.....	333,022	7,115	748,541	9,610	788,873
1908....	(*h*)534,117	19,593	.....	139,871	4,305	1,012,660	8,685	527,987
1909....	(*h*)684,670	20,801	.....	147,782	3,573	1,207,029	11,922	420,755
1910 ...	(*h*)745,963	14,963	.....	190,385	4,365	1,346,471	16,905	315,895
1911....	794,903	10,617	5	119,863	3,285	1,820,923	15,457	291,092

Year.	Phosphate (Apatite).	Pyrites.	Salt.	Silver—*Kg.* (In ore, etc.)	Soapstone. and Talc.
1898	665	29,223	51,828	138,486	367
1899	2,721	25,112	53,820	106,116	408
1900	1,283	36,308	56,284	138,980	1,288
1901	937	31,982	53,901	172,292	235
1902	776	32,304	58,462	133,478	625
1903	1,205	30,822	56,644	99,489	898
1904	832	29,980	62,411	115,666	762
1905	1,180	29,713	41,159	185,839	454
1906	(*b*)	35,927	69,283	266,521	1,119
1907	680	35,494	65,936	390,359	1,391
1908	1,447	42,934	72,537	687,504	976
1909	905	58,645	76,237	856,263	3,945
1910	1,341	48,871	76,286	1,022,350	6,450
1911	506	74,994	83,083	1,018,352	6,621

(*a*) From *Reports Compiled by the Geological Survey of Canada.* (*b*) Not reported. (*c*) Gold values are calculated at the rate of $20.67 per oz. (*d*) Export. (*e*) One barrel contains 35 imp. gal. (*g*) From Canadian ore, 149,444, short tons in 1909, 104,906 tons in 1910 and 42,186 tons in 1911. (*h*) Steel ingots and castings. (*i*) 1909, $94,609; 1910, $51,986; 1911, $221,790; values received.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (*a*)
(In metric tons or dollars.)

Year. (*b*)	Aluminum.		Antimony. (*c*)	Arsenic.	Asbestos. (*d*)	Asphalt.	Cement.
	Manufactures. (*m*)	Ingots, Sheets, etc.					
1899	$9,275	.....	131	264	$32,607	8,196	$477,617
1900	12,543	.....	90	105	43,455	2,825	513,770
1901	16,202	.....	159	72	50,829	2,849	666,350
1902	30,496	.....	229	48	52,464	3,426	863,646
1903	14,201	$ 13,930	393	135	75,465	3,037	890,745
1904	16,065	101,427	190	188	83,827	7,093	1,014,713
1905	28,418	154,569	85	122	116,836	5,096	1,263,828
1906	23,565	168,405	183	202	138,000	7,178	1,003,022
1907(*o*)	20,656	218,399	146	158	127,509	11,929	540,006
1908(*r*)	37,197	131,762	220	228	191,204	14,113	865,275
1909(*r*)	30,076	167,019	201	58	181,710	15,979	473,211
1910(*r*)	42,493	424,006	232	12	198,756	19,661	159,040
1911 (*r*)	83,638	782,254	257	115	254,331	23,248	(*t*)494,446

Year.	Coal.		Coke.	Copper.	Copper, Sulphate.	Gold and silver.
	Anthracite. (*f*)	Bituminous. (*f*)		Ingots, Pig and Scrap.		Coin and Bullion. (*g*)
1899	1,583,132	2,220,250	128,145	751	726	$4,705,134
1900	1,500,542	2,512,334	170,405	519	752	8,297,438
1901	1,753,488	2,658,257	280,069	432	673	3,537,294
1902	1,498,773	3,208,005	242,298	801	711	6,311,405
1903	1,320,239	3,684,502	232,848	924	1,010	8,976,797
1904	2,064,444	4,230,436	200,590	960	795	7,874,313
1905	2,361,952	4,377,667	337,035	882	934	10,308,435
1906	1,996,183	5,003,029	435,561	1,191	844	7,078,603
1907(*o*)	1,260,723	4,022,843	363,286	1,186	897	7,029,047
1908(*r*)	2,803,681	7,681,464	561,677	1,638	1,161	6,548,661
1909(*r*)	2,775,680	6,526,797	423,013	1,239	963	9,988,442
1910(*r*)	2,860,196	10,382,156	636,888	2,329	859	6,017,589
1911	3,144,130	7,785,259	692,292	2,279	772	10,206,210

Year.	Graphite.		Gypsum.		Iron and Steel.		
	Crude.	Manufactures. (h)	Crude and Ground.	Plaster of Paris.	Pig and Scrap.	Slabs, Blooms Bars, etc.	Alloys of Iron.
1899.....	$4,979	$37,824	310	225	69,819	5,640	1,053
1900.....	4,437	60,518	72	385	94,489	11,576	1,643
1901.....	2,357	75,536	289	228	59,033	10,659	1,572
1902.....	3,649	64,123	516	215	71,882	18,208	3,910
1903.....	2,870	69,676	1,007	286	129,641	17,896	5,761
1904.....	1,802	67,563	626	291	86,087	9,088	2,760
1905.....	2,499	75,288	2,972	3,595	90,698	14,420	11,736
1906.....	2,791	86,028	5,743	6,579	112,937	29,520	13,426
1907(o).....	3,176	57,430	8,334	9,730	137,654	17,369	17,785
1908(r).....	3,030	78,380	8,519	6,955	190,994	35,534	16,139
1909(r).....	1,408	75,608	9,359	7,712	62,967	43,263	13,571
1910(r).....	5,223	95,900	3,618	19,095	159,515	97,036	14,471
1911 (r).....	4,300	97,057	11,443	19,664	266,269	139,060	15,772

Year.	Kainite.	Lead.				Lime.		Mineral Paints (Ochers).
		Pig and Scrap.	Bars and Sheets.	Litharge.	Pigments and Zinc White.	Burned. *Barrels.*	Chloride of.	
1899...	30	5,202	2,032	432	6,583	15,720	1,857	1,110
1900...	143	2,829	703	415	6,661	12,865	1,967	1,122
1901...	88	(i)3,871	739	505	4,647	19,657	1,605	1,031
1902...	85	(i)5,548	844	590	7,071	24,602	1,806	1,145
1903...	259	(i)4,471	523	632	8,715	31,108	2,104	1,459
1904...	339	4,292	.....	.....	7,679	54,359	2,080	1,256
1905...	306	2,589	800	811	9,695	98,676	2,507	1,417
1906...	306	3,751	730	461	6,947	134,334	2,645	809
1907(o)	511	3,811	622	513	2,215	88,919	2,302	570
1908(r)	743	2,902	782	864	5,743	129,379	3,421	788
1909(r)	562	2,273	623	550	3,998	153,934	2,697	546
1910(r)	687	5,755	580	915	4,433	191,527	4,554	914
1911(r)	339	5,292	707	750	5,312	176,730	4,745	1,169

Year.	Nickel.	Petroleum Products.		Platinum.	Potassium Salts.		Quicksilver.
		Illuminating Oil etc., Crude or Refined. *Gallons.*	Paraffin Wax and Candles.		Except Saltpeter.	Saltpeter.	
1899.....	$9,446	10,394,208	70	$9,671	472	930	47
1900.....	6,988	9,633,647	35	57,910	733	602	39
1901.....	12,029	11,082,822	74	20,263	476	581	64
1902.....	15,448	13,220,005	123	19,357	771	690	44
1903.....	26,177	18,799,312	307	21,251	1,060	916	75
1904.....	14,682	24,521,115	228	28,112	1,151	898	69
1905.....	19,076	13,229,855	98	61,719	945	1,048	47
1906.....	15,976	10,981,611	375	54,494	1,317	1,141	68
1907(o).....	19,461	8,066,403	189	113,967	1,074	638	44
1908(r).....	.....	8,844,129	102	63,582	3,396	2,653	81
1909(r).....	(s)67,107	13,095,593	103	47,371	1,570	925	42
1910(r).....	(s)98,238	(u)48,384,763	253	80,180	2,497	665	129
1911 (r).....	137,114	(u)66,119,105	931	136,823	3,152	1,490	59

Year.	Sal-Ammoniac.	Salt.	Silex.	Sodium Salts Except Chloride.	Sulphur.	Tin and Tin-ware.	Zinc.
1899	53	88,397	179	20,742	11,121	$1,372,813	852
1900	60	92,823	182	16,748	9,584	2,418,455	1,304
1901	76	103,402	162	18,631	10,827	2,339,109	931
1902	78	114,629	199	17,133	11,180	2,293,958	1,582
1903	114	112,188	159	18,887	11,077	2,712,168	1,209
1904	93	103,635	252	25,118	8,786	2,389,557	1,540
1905	143	97,723	405	26,219	10,633	2,791,757	1,721
1906	209	99,788	338	30,401	19,512	3,105,876	3,383
1907(o)	130	73,156	542	25,068	11,725	2,473,572	2,761
1908(r)	172	105,286	1,131	39,154	23,494	1,619,647	2,521
1909(r)	162	119,660	417	33,787	19,981	2,984,065	2,993
1910(r)	303	121,473	582	47,913	19,484	3,512,615	(v)7,606
1911(r)	346	111,174	468	62,969	22,935	4,235,848	(v)6,069

EXPORTS OF DOMESTIC MINERAL PRODUCE FROM THE DOMINION OF CANADA (a)

(In metric tons or dollars.)

Year (b)	Antimony Ore.	Asbestos.	Chromite.	Coal.	Coke.	Copper (e).
1899	.....	13,176	(k)1,369	1,035,245	4,024	3,843
1900	6	16,483	(k) 334	1,489,139	12,558	6,274
1901	219	24,242	(k)2,049	1,713,737	60,129	11,954
1902	13	30,011	(k) 672	1,649,278	52,873	13,789
1903	128	27,823	658	1,796,689	39,616	13,445
1904	87	31,444	2,103	1,494,106	61,750	20,279
1905	340	37,320	3,702	1,465,809	116,387	17,431
1906	388	40,367	1,640	1,651,203	50,004	20,082
1907(o)	832	37,194	604	1,165,809	44,669	11,845
1908(r)	693	53,543	1,585	1,702,673	50,343	25,824
1909(r)	1	54,188	3,707	1,548,468	70,024	24,642
1910(r)	33	58,083	1,052	1,822,398	44,343	26,107
1911(r)	222	63,348	14	2,100,309	54,380	24,950

Year.	Gold, Quartz, Dust, etc.	Graphite.	Grindstones.	Gypsum crude.	Iron ore.	Lead (p).
1899	$3,272,702	662	$18,619	148,565	(n) 2,881	15,445
1900	14,148,543	1,742	22,196	211,792	(n) 5,012	8,998
1901	24,445,156	1,246	38,304	156,080	(n) 54,208	29,747
1902	19,668,015	783	21,878	243,629	(n)478,503	13,890
1903	16,437,528	530	14,169	271,899	(n)267,000	7,386
1904	18,715,539	269	12,676	247,741	(n)214,309	7,329
1905	15,208,380	201	27,985	290,574	204,091	23,094
1906	12,991,916	180	15,793	367,203	134,270	6,158
1907(o)	7,226,954	3	33,929	249,780	31,011	8,330
1908(r)	8,817,041	167	28,726	340,235	23,863	12,650
1909(r)	7,392,610	396	18,019	239,139	3,568	5,459
1910(r)	6,110,473	1,164	13,754	304,736	28,608	8,395
1911(r)	5,344,465	362	23,914	325,928	95,080	1,451

Year.	Manganese Ore.	Mica.	Nickel in Ore, Matte, etc.	Petroleum, Crude and Refined.	Pyrites.	Salt. *Bushels.*	Silver, *Kg.* (in ore, matte, etc).
1899	24	538	6,546	4,268	11,707	5,209	137,400
1900	57	490	6,122	6,758	13,507	15,151	71,015
1901	33	444	4,327	19,942	22,146	56,461	125,110
1902	500	452	1,762	2,478	24,088	21,778	114,610
1903	137	632	4,098	413	16,762	7,959	100,861
1904	62	393	6,456	1,208	15,582	42,662	99,472
1905	84	461	5,431	6,441	20,473	5,663	112,076
1906	15	603	10,866	1,741	18,398	23,168	203,323
1907(*o*)...	84	631	7,355	(*q*)3,167	20,148	5,113	274,178
1908(*r*)...	1	409	8,596	(*q*)3,389	17,835	35,543	515,161
1909(*r*)...	3	243	8,895	(*q*)195	23,087	(*w*)3,040	733,248
1910(*r*)...	5	397	13,891	(*q*)9,093	24,439	(*w*)6,088	932,098
1911(*r*)...	2	370	15,771	(*q*)1,493	32,396	(*w*)6,625	1,049,153

(*a*)From Tables of the *Trade and Navigation of the Dominion of Canada.* (*b*) Fiscal year ending June 30. (*c*) Includes regulus and salts of antimony. (*d*) Asbestos in any form except crude, and all manufactures of. (*e*) Includes copper in ore, matte, regulus, etc. (*f*) Includes coal dust. (*g*) Coin, gold and silver, except U. S. silver coin. (*h*) Includes black lead and crucibles (clay or graphite). (*i*) Includes Canadian lead ore refined in the United States. (*k*) Calendar year. (*m*) Unclassified. (*n*) Includes chromic iron ore. (*o*) Returns for the 9 months of the fiscal year ending March 31. (*p*) Includes lead contained in ore, etc. (*q*) Gallons. (*r*) Fiscal year ending March 31. (*s*) Includes silver-nickel and German silver. (*t*) Weight, 64,176 net tons. (*u*) Includes all oils. (*v*) Not including manufactures. (*w*) Using 60 lbs. per bushel.

CHINA

The official statistics of mineral imports and exports are summarized in the following tables:

MINERAL IMPORTS OF CHINA. (*a*)

(In metric tons.)

Year.	Brass and Yellow Alloys.	Copper.	Lead.	Nickel.	Petroleum, Gal.	Quicksilver.	Tin.	Tinplate.	Zinc.	
									Spelter.	Sheet, etc.
1908...	1,500	13,129	10,707	42	186,175,950	40	3,716	13,465	519	646
1909 (*c*)	1,516	1,012	8,198	127	145,719,886	40	3,076	11,774	113	727
1910...	1,674	2,428	9,247	134	161,399,583	51	2,684	11,569	.....	858
1911...	1,244	4,408	8,138	39	....	49	1,994	21,676	81	769

MINERAL EXPORTS OF CHINA. (*a*)

(In metric tons.)

Year.	Antimony.		Iron.		Lead.		Quicksilver.	Tin.	Zinc.	
	Ore.	Metal(*b*)	Ore.	Pig and Mnfd.	Ore.	Pig.			Ore.	Spelter.
1908.....	544	9,356	133,458	30,897	1,283	5	44	4,836	7,619	169
1909.....	776	7,937	89,100	38,714	4,039	12	51	4,516	7,578	242
1910.....	6,643	5,762	132,561	65,381	5,176	3	48.5	6,510	9,786	328
1911.....	6,986	6,812	112,295	70,906	5,410	11.2	18.5	6,056	4,599	710

(*a*) From annual reports of the Imperial Chinese Maritime Customs. (*b*) Regulus and refined. (*c*) Reported jointly by U. S. Consuls at Hongkong and Tientsin. (*d*) Through Consul General Anderson, Hongkong, figures directly from the Inspectorate General of Customs at Shanghai.

FRANCE

In the following tables are given the statistics of mineral and metal production in France and the French colonies—Algeria, New Caledonia and Tunis—together with the foreign commerce of France in mineral and metal products:

MINERAL AND METALLURGICAL PRODUCTION OF FRANCE. (*a*)
(In metric tons.)

Year.	Alumi-num.	Antimony.		Arsenic Ore.	Asphal-tum.	Barytes.	Baux-ite.	Bitu-men. (*c*)	Cement.
		Ore.	Metal.						
1898.....	565	4,433	1,226	.....	18,832	2,763	36,723	229,108	1,072,025
1899.....	763	7,392	1,499	2,600	22,100	4,058	48,215	258,449	1,144,271
1900.....	1,026	7,843	1,573	4,705	25,228	3,635	58,530	266,474	1,147,670
1901.....	1,200	9,867	1,786	7,491	20,391	4,145	76,620	249,655	1,127,200
1902.....	1,355	9,715	1,725	5,372	.....	4,323	96,900	258,295	962,930
1903.....	1,570	12,380	2,748	6,658	.....	5,731	133,890	243,295	898,393
1904.....	1,650	9,065	2,116	3,117	22,000	6,944	75,640	227,177	903,632
1905.....	1,905	12,543	2,396	3,627	20,000	5,504	103,207	188,403	922,531
1906.....	3,396	18,567	3,433	6,534	38,231	11,680	117,781	196,375	1,257,861
1907.....	4,700	24,000	3,950	7,900	33,000	11,150	158,000	177,000	1,253,546
1908.....	4,681	26,026	3,850	2,381	41,000	16,277	170,679	171,158	1,359,658
1909.....	6,092	28,105	5,444	2,141	44,800	14,111	130,149	169,000	1,374,574
1910.....	6,425	28,130	4,640	8,045	38,500	11,632	196,056	169,769	1,521,131

Year.	Coal.	Lignite.	Peat.	Copper.		Gold.	Gypsum.	
				Ore.	Metal.		Crude.	Calcined.
1899.....	32,256,148	606,564	99,230	2,021	6,640	$179,429	263,879	1,372,087
1900.....	32,721,562	682,736	95,630	3,031	6,446	134,904	192,916	1,405,845
1901.....	31,633,300	691,700	118,433	3,413	7,000	85,727	355,995	1,623,710
1902.....	29,365,047	632,423	109,941	828	6,300	(*b*)	219,487	1,572,687
1903.....	34,217,661	688,757	100,348	10,892	6,921	(*b*)	162,766	1,468,830
1904.....	33,502,394	665,572	95,716	2,756	6,900	(*b*)	106,173	1,481,303
1905.....	35,218,000	709,000	98,500	5,068	7,576	235,447	78,832	1,299,313
1906.....	33,458,000	738,000	92,469	2,547	5,770	511,665	79,568	1,297,861
1907.....	35,989,000	765,000	90,952	2,400	7,800	847,290	87,370	1,316,567
1908.....	36,633,000	751,000	79,759	766	7,935	960,666	92,898	1,326,131
1909.....	37,115,500	724,500	78,600	458	7,823	1,147,400	76,057	1,263,697
1910.....	37,634,893	715,000	48,415	222	12,933	(*b*) 1,706,580	64,181	1,533,298
1911.....	38,643,531	.....	.....	.....	.....	.....	.....	.....

Year.	Ore.	Iron.			Lead.		Lime.	Manga-nese Ore.	Mill-stones.
		Pig.	Wrought. Iron.	Wrought. Steel.	Ore. (*d*)	Pig. (*e*)			
1898.	4,731,394	2,525,100	766,000	1,174,000	23,342	10,920	2,339,850	31,935	38,929
1899.	4,985,702	2,578,400	834,000	1,240,000	17,505	15,981	2,343,377	39,897	41,535
1900.	4,676,740	2,714,298	672,172	1,226,537	24,276	15,210	2,377,110	28,992	41,103
1901.	4,260,747	2,388,823	612,362	1,175,454	20,644	21,000	2,443,062	22,304	33,286
1902.	5,003,782	2,405,000	639,600	1,245,800	22,634	19,000	4,796,807	12,536	34,504
1903.	6,219,541	2,840,517	598,910	1,305,709	23,080	23,258	4,727,543	11,583	35,031
1904.	7,022,841	2,999,787	554,632	1,482,708	14,173	18,800	4,583,522	11,254	37,409
1905.	7,395,409	3,077,000	670,000	1,442,000	12,118	24,100	3,694,725	6,751	33,468
1906.	8,481,423	3,314,100	747,900	1,683,500	11,795	25,614	3,869,772	11,189	32,407
1907.	10,008,000	3,590,000	579,900	1,860,000	18,000	24,800	2,438,409	18,200	30,480
1908.	10,057,145	3,400,700	560,200	1,851,900	13,403	26,112	2,535,833	15,865	30,522
1909.	11,890,000	3,573,848	558,000	2,040,364	13,794	26,927	2,211,653	9,378	30,503
1910.	14,605,542	4,038,000	526,000	2,324,000	14,536	20,226	2,110,540	7,925	32,461

ear.	Mineral Paints (Ochers).	Nickel.	Phosphate Rock.	Pyrites.	Salt.	Silver. *Kg.*	Sulphur. Ore .(*g*)	Zinc. Ore.	Zinc. Metal.
3....	33,780	1,540	568,558	310,972	999,283	.....	9,818	85,550	37,155
)....	32,750	1,740	645,868	318,832	1,193,532	.....	11,744	84,813	39,274
)....	33,080	1,700	587,919	305,073	1,088,634	.....	11,551	67,059	36,305
l....	35,704	1,800	535,676	307,447	910,000	.....	7,000	61,539	37,600
)....	34,770	1,600	543,900	318,235	863,927	.....	8,021	57,982	36,300
)....	34,042	1,500	475,783	322,118	967,531	.....	7,375	66,922	37,416
l....	34,945	1,500	423,521	271,544	1,153,754	.....	5,447	52,842	41,600
5....	37,800	1,800	476,720	267,114	1,130,088	.....	4,637	62,150	43,200
)....	35,550	1,750	469,408	265,261	1,335,420	50,058	2,713	53,466	46,536
7....	32,856	1,500	432,237	283,000	1,226,000	47,009	2,000	44,000	47,900
3....	33,060	1,800	485,607	284,717	1,100,000	61,184	2,189	52,611	47,880
)....	33,540	1,600	397,908	273,221	1,113,000	63,671	2,900	96,903	49,956
)....	32,870	2,100	333,506	250,432	1,051,427	52,957	2,641	50,624	52,598

(*a*) From *Statistique de l'Industrie Minérale.* (*b*) Not reported. (*c*) Includes pure bitumen, minous schist and sand, and asphaltic limestone. (*d*) Argentiferous lead ore. (*e*) Lead pro-ed from native ores only. (*g*) Sulphur and limestone impregnated with sulphur. (*h*) From ores.

MINERAL PRODUCTION OF ALGERIA. (*a*)

(In metric tons.)

ear.	Anti-mony Ore.	Copper Ore.	Gypsum. Crude.	Gypsum. Plaster.	Iron Ore.	Lead-Silver. Ore.	Mer-cury.	Onyx.	Phos-phate Rock.	Salt.	Zinc Ore.
3...	138	488	150	29,750	473,569	120	.....	219	269,500	21,300	29,800
9...	200	472	200	31,800	550,921	389	.....	217	324,983	17,378	42,970
0...	93	.....	500	37,100	174,000	222	.....	228	319,422	18,325	30,281
1...	...	7,267	600	34,740	161,303	1,614	.....	294	265,000	18,518	26,913
2...	39	1,955	600	35,500	525,012	26	.....	150	305,174	27,263	33,139
3...	490	100	300	33,000	588,893	499	.....	67	320,834	26,329	43,313
4...	160	1,804	350	38,420	468,737	511	.....	121	343,317	18,563	47,192
5...	.....	1,784	.....	34,743	568,609	7,470	.....	270	334,784	26,986	67,922
6...	50	2,786	.....	27,950	779,826	11,246	.....	216	333,531	22,615	74,351
7...	799	16,259	.....	26,400	973,445	15,264	590	328	373,763	20,390	71,048
8...	190	3,330	.....	25,500	943,424	10,626	1,556	300	452,060	25,215	94,399
9...	163	67	.....	29,000	836,044	11,131	.....	200	345,385	17,817	81,852
0(*a*)	130	112	.....	48,500	1,064,909	12,652	50	123	412,319	21,470	94,445

(*a*) From *Statistique de l'Industrie Minérale.*

MINERAL PRODUCTION OF NEW CALEDONIA. (*a*)

(In metric tons.)

ear.	Chrome Iron. Ore.	Cobalt. Ore.	Copper Ore.	Nickel. Ore.	Year.	Chrome Iron Ore.	Cobalt Ore.	Copper Ore.	Nickel Ore.
0.....	10,474	2,438	2	100,319	1906...	84,211	2,600	207	118,890
1.....	17,451	3,123	6,349	132,814	1907...	57,367	29,800	437	119,000
2.....	10,281	7,512	3,720	129,653	1908...	25,371	2,360	(*b*) 10	108,000
3.....	21,437	8,292	10	77,360	1909...	40,000	548	9,600	86,000
4.....	42,197	8,964	*Nil.*	98,655	1910...	40,000	54	8,000	99,000
5.....	51,374	7,920	*Nil.*	125,289	1911(*c*).	82,806	*Nil.*	(*d*)	120,659

(*a*) From *Statistique de l'Industrie Minérale.* (*b*) From *Le Bulletin du Commerce* Nouméa. (*c*) ports *Echo des Mines.* (*d*) Copper matte worth $101,000.

MINERAL PRODUCTION OF TUNIS. (a)

(In metric tons.)

Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.	Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.
1900....	9,160	6,864	178,000	16,596	1906...	62,600	14,800	796,000	32,400
1901....	16,900	8,158	172,000	17,879	1907...	78,200	18,600	1,069,000	22,800
1902....	12,600	12,892	264,930	18,400	1908...	149,600	37,500	1,300,500	26,500
1903....	18,846	12,752	352,088	21,262	1909...	118,400	41,600	1,300,000	24,500
1904....	23,600	16,800	455,197	27,200	1910...	199,700	(c) 37,000	1,334,000	(c) 32,500
1905....	54,900	15,200	522,000	37,100	1911...	(b)	36,100	(c) 1,539,000	(c) 34,395

(a) From *Statistique de l'Industrie Minérale.* (b) Not reported. (c) From *l'Echo des Mines.*

MINERAL IMPORTS OF FRANCE. (a) (k)

(In metric tons or dollars. 5 f = $1.)

Year.	Alum.	Bitumen. (f)	Borax	Bromides.	Cement	Coal and coke.	Copper.		Copper.	
							Ore.	Ingot and Mfres.	Sulphate.	Oxide
1896....	41	30,954	255	13	14,395	10,180,449	8,584	46,830	33,803	22
1897....	54	29,931	264	18	15,141	10,457,255	11,960	54,460	30,132	29
1898....	27	20,385	139	30	11,290	10,445,090	8,779	52,976	30,897	52
1899....	34	30,770	123	46	13,640	11,896,030	8,517	58,419	21,733	36
1900....	23	39,598	111	10	13,612	14,601,981	9,766	61,638	22,820	84
1901....	39	28,888	128	3	16,232	13,925,623	13,383	47,035	15,313	162
1902....	36	26,053	141	3	15,720	13,137,720	17,862	54,484	22,273	111
1903....	138	27,573	312	9	21,152	14,029,687	9,796	59,126	25,428	129
1904....	370	17,178	3,113	17	21,702	13,936,475	9,942	69,183	30,856	142
1905....	63	24,606	1,736	31	21,954	13,910,523	14,252	70,101	23,805	57
1906....	105	99,336	189	93	24,974	17,848,284	11,932	64,590	15,358	97
1907....	.....	31,700	.....	.....	24,839	18,706,000	12,063	76,282	.....	.....
1908....	.....	48,000	.....	.....	31,550	18,563,000	15,300	86,985	.....	.....
1909....	.....	54,000	.....	.....	46,664	19,387,000	27,400	86,143	.....	.....
1910....	46	36,360	8,322	17	51,903	18,145,872	23,362	93,548	11,887	427
1911....	119	32,200	3,121	12	8,645	19,734,530	24,585	104,000	17,759	151

Year.	Cobalt Ore.	Iron.					Kaolin.	Lead.		
		Ore.	Pig.	Iron and Steel, Mfres. of.	Sulphate.	Oxide.		Ore.	Carbonate.	Pig, Scrap and Mfres.
1896....	.....	1,862,043	18,323	48,423	3,086	897	38,703	5,569	892	79,752
1897....	.....	2,137,860	35,633	60,804	1,353	1,125	42,384	13,981	1,327	86,589
1898....	.....	2,032,240	(b)	47,325	896	1,021	40,352	14,377	1,376	74,902
1899....	.....	1,950,665	(b)	64,178	1,698	1,037	36,904	12,637	2,029	67,149
1900....	.....	2,119,003	149,755	118,152	1,589	1,022	39,842	19,772	1,739	70,857
1901....	.....	1,662,875	61,085	77,742	45	1,001	41,972	15,430	1,789	59,051
1902....	.....	1,563,334	38,521	60,697	17	1,051	41,165	13,121	2,223	58,694
1903....	.....	1,832,820	121,726	119,799	36	1,207	47,534	20,172	2,040	75,416
1904....	.....	1,738,514	135,252	125,709	319	1,151	50,465	25,731	2,221	76,198
1905....	.....	2,151,954	122,102	150,480	709	1,330	52,603	35,103	2,306	73,938
1906....	.....	2,015,550	156,618	342,411	132	1,311	44,772	43,137	2,072	67,651
1907....	1,396	1,999,000	154,031	336,337	.....	.....	53,447	42,342	.....	53,359
1908....	1,500	1,454,000	168,810	373,624	.....	.....	58,909	40,700	.....	70,824
1909....	400	1,202,619	194,225	354,428	.....	.....	59,490	38,400	.....	68,332
1910....	2.6	1,318,520	234,631	378,227	2,243	1,235	62,920	28,950	3,259	64,517
1911....	0.3	1,350,794	49,612	77,000	4,454	1,498	66,500	47,340	3,095	120,000

Year.	Lime, Chloride of.	Manganese Ore.	Nickel.		Petroleum.	Phosphate Rock.	Platinum *Kg.*	Potassium.	
			Ore.	Metal.				Chloride.	Chromate. (*h*)
1896	2,033	61,600	15,756	425	272,693	256,888	2,117	11,499	2,838
1897	1,713	85,500	17,441	316	288,671	313,608	1,069	11,630	2,852
1898	1,288	100,243	24,935	330	291,961	336,842	505	10,929	2,890
1899	1,887	106,630	28,620	286	306,078	242,021	817	13,335	3,147
1900	1,215	120,790	17,687	299	302,482	283,921	2,398	13,524	3,293
1901	1,400	94,365	39,497	252	225,962	275,285	1,857	13,299	2,784
1902	2,130	85,629	58,374	301	148,170	302,898	2,940	10,802	2,861
1903	919	109,930	13,933	427	(*g*)476,230	343,012	3,764	12,275	2,760
1904	1,679	105,652	20,698	313	(*g*)435,730	419,720	5,650	14,734	2,618
1905	406	140,871	49,698	632	(*g*)512,727	447,738	4,023	21,819	2,619
1906	593	127,235	44,960	480	(*g*)213,462	533,213	5,708	26,523	3,024
1907	.....	192,448	45,892	979	(*g*)311,000	636,549	4,373	.....	.....
1908(*k*)..	.....	170,500	42,200	1,281	(*g*)300,000	767,424	3,955	.....	.....
1909(*k*)..	.....	177,314	12,200	.....	(*g*)300,000	645,178	5,914	.....	.....
1910	333	188,292	28,000	2,074	400,000	687,675	7,795	34,974	3,009
1911 ...	53	235,400	43,000	6,500	600,000	757,130	9,392	39,508	2,985

Year.	Potassium. (*Cont,d*).		Pyrites.	Quicksilver.		Sal-ammoniac.	Salt.	Sodium.	
	Nitrate.	Carbonate		Ore.	Metal.			Hydrate.	Nitrate.
1897	1,309	1,769	69,470	24	248	27,454	32,917	1,378	8,105,400
1898	1,008	2,418	71,569	19	221	20,426	35,863	1,772	8,026,400
1899	1,015	2,779	109,696	21	276	12,210	37,970	1,494	9,341,600
1900	1,928	2,768	156,825	22	161	15,205	32,045	1,062	11,995,820
1901	757	2,520	205,617	23	205	9,268	32,347	869	10,526,400
1902	1,547	1,539	170,783	24	224	15,446	32,505	643	9,372,600
1903	1,530	3,019	205,322	20	220	12,462	48,556	781	10,810,775
1904	2,117	3,781	230,097	22	208	13,744	46,232	1,068	9,074,859
1905	1,022	3,542	271,684	.....	228	11,639	45,241	860	11,336,752
1906	684	2,206	349,514	.....	242	18,146	38,361	614	13,678,848
1907	.....	.....	355,300	.....	216	.....	30,000	.....	.....
1908(*k*) .	.....	.....	348,300	.....	180	.....	33,000	.....	.....
1909(*k*) .	.....	.....	275,600	.....	191	.....	30,000	.....	.....
1910 .. .	132	2,579	309,400	.....	146	27,800	49,800	577	345,160
1911	174	4,452	318,530	.....	196	23,200	47,200	550	317,725

Year.	Sulphur.	Sulphuric Acid.	Superphosphate of Lime.	Tin.		Zinc.	
				Ore.	Metal.	Ore.	Metal.
1897.....	136,118	3,147	195,853	149	7,642	58,074	31,211
1898.....	130,289	4,666	178,569	357	9,247	60,481	32,342
1899.....	120,062	4,583	171,631	486	6,907	78,192	25,516
1900.....	133,531	4,254	143,437	512	7,324	66,178	33,144
1901.....	101,301	5,386	165,361	365	7,314	74,553	29,812
1902.....	85,839	7,793	116,093	748	8,575	69,451	36,564
1903.....	109,594	13,241	89,229	1,808	9,873	67,258	39,305
1904.....	148,547	11,212	72,921	1,344	9,352	88,083	35,737
1905.....	129,877	10,915	31,729	1,362	9,898	105,069	29,163
1906.....	131,678	5,268	44,502	1,038	7,687	106,307	26,960
1907.....	106,050	.....	.....	961	7,693	114,699	33,503
1908(*k*).....	195,000	.....	.....	1,000	8,482	137,900	40,312
1909(*k*).....	196,300	.....	.....	1,900	9,683	120,600	32,749
1910.....	201,500	14,640	132,530	1,650	8,517	193,360	19,237
1911.....	268,000	15,050	79,385	2,540	8,400	160,180	42,500

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (a) (k)
(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic.	Cement.	Coal.	Copper.		Gold. Kg. (d)
		Ore.	Metal.				Ore. (c)	Metal.	
1896	793	736	74	.....	242,247	1,044,820	1,261	10,494	2,193
1897	224	623	61	.....	244,504	1,142,195	2,000	12,667	3,335
1898	192	616	101	.....	241,150	1,320,616	1,783	14,350	1,812
1899	256	304	255	.....	244,480	1,229,090	2,078	17,949	2,622
1900	324	154	336	.....	232,577	1,201,210	9,197	16,791	883
1901	307	645	741	.....	242,010	908,583	16,066	14,776	1,589
1902	748	595	666	.....	210,590	910,760	20,489	14,423	1,517
1903	666	904	1,358	.....	233,835	2,238,735	12,487	11,403	3,139
1904	664	1,191	720	.....	260,686	2,384,928	14,258	12,663	1,537
1905	928	981	815	.....	275,503	(i)3,348,010	13,260	13,800	5,740
1906	1,522	3,541	871	.....	329,879	(i)1,448,000	8,056	6,130	11,727
1907	1,118	3,460	1,270	.....	366,624	(i)1,224,000	4,151	18,630	6,289
1908	1,332	3,300	2,129	8,600	342,131	(i)1,117,000	3,600	17,845	4,455
1909	4,425	4,000	2,408	3,267	316,486	(i)1,493,308	6,200	18,775	5,366
1910	4,027	4,583	2,163	2,411	315,020	(i)1,580,077	4,163	25,046	7,264
1911	4,181	3,991	1,945	832	333,043	1,634,100	8,335	22,600	

Year.	Iron.				Lead.		Manganese Ore.
	Ore.	Pig.	Bars.	Steel.	Ore.	Metal.	
1896	238,430	195,212	24,721	44,975	8,597	10,856	10,913
1897	299,589	108,645	39,894	45,809	12,007	10,364	19,464
1898	236,169	162,991	27,424	47,278	10,216	3,663	12,229
1899	291,346	153,792	29,112	33,584	3,909	1,163	12,289
1900	371,799	114,361	18,763	19,535	2,345	958	8,302
1901	258,925	96,463	25,220	56,347	3,490	718	5,289
1902	422,677	213,081	23,828	121,932	2,414	648	1,948
1903	714,173	196,444	40,533	215,737	2,313	13,048	717
1904	1,219,149	191,819	40,374	246,738	1,860	13,467	1,392
1905	1,355,932	218,227	67,240	343,612	3,064	12,903	662
1906	1,759,443	143,142	58,826	236,617	1,354	997	4,102
1907	2,147,000	249,708	84,557	291,434	1,210	1,912	5,107
1908	2,384,000	171,797	86,691	360,509	6,700	1,974	1,000
1909	3,907,338	161,058	96,999	318,925	2,800	2,458	1,149
1910	4,892,542	110,086	122,543	(k)302,168	5,855	3,004	722
1911	6,160,100	104,350	171,000	100,000	9,691	2,660	902

Year.	Nickel, Refined.	Phosphate Rock.	Plaster.	Pyrites.	Silver. Kg. (e)	Tin (Metal).	Zinc.	
							Ore.	Spelter, Sheets and Scrap.
1897	498	69,188	107,823	54,367	5,374	651	79,909	10,977
1898	526	93,742	106,790	60,406	1,886	587	60,664	16,995
1899	280	70,517	112,520	53,395	.....	666	76,104	14,958
1900	599	89,135	108,387	64,530	15,470	716	54,663	12,712
1901	1,031	81,405	101,063	52,952	16,745	438	42,995	15,022
1902	397	62,375	110,270	63,920	17,184	654	47,724	16,158
1903	720	72,252	131,245	119,173	43,690	1,994	62,731	12,657
1904	906	78,612	139,551	40,833	23,105	2,300	57,780	19,063
1905	1,583	55,240	124,561	21,257	66,904	2,611	72,512	17,802
1906	1,088	81,660	142,339	26,216	87,952	601	67,258	19,607
1907	1,414	100,508	137,356	24,417	58,199	729	54,316	21,928
1908	1,230	71,509	132,924	40,300	56,957	810	57,800	20,589
1909	1,940	47,611	120,701	40,200	56,549	886	60,000	24,729
1910	2,050	44,380	152,390	56,735	58,294	1,224	56,800	22,975
1911	880	31,245	149,806	68,167	17,256	1,520	23,000	25,500

(a) From *L'Economiste Francais* (representing the *Commerce Spécial*) except for 1903-06, inclusive, which are from *Tableau Général du Commerce et de la Navigation*. (b) Not reported. (c) Includes matte. (d) Gold and] platinum in sheets, leaves, threads or jewelry and crude platinum. (e) Silver in sheets, leaves, wire and jewelry. (f) Includes bitumen, bituminous schist and sand and asphaltic limestone. (g) Crude and refined. Transposition from hectoliters to tons was performed by assuming specific gravity of petroleum to be 0.9. (h) Includes chromate of soda. (i) Includes coke. (k) From *Statistique de l'Industrie Minérale* in the years 1908, 1909, 1910. Steel in 1910 not classified as before. 1911, from *l'Echo des Mines et Metallurgie*.

GERMANY

The mineral production and foreign commerce of the German Empire given in the following tables in metric tons unless otherwise specified, in dollars, on the basis of four marks to the dollar.

MINERAL PRODUCTION OF GERMANY. (*a*)

ear.	Alum.	Alumi-num. Sulphate	Arsenic.		Asphal-tum.	Boracite.	Cad-mium. *Kg.*	Coal.	
			Ore.	Salts.				Bituminous.	Lignitic.
....	4,069	35,366	3,527	2,679	67,649	230	14,943	96,309,652	31,648,898
....	3,358	37,693	3,834	2,423	74,770	183	13,608	101,639,753	34,204,666
....	4,355	44,372	4,379	2,415	89,685	232	13,553	109,290,237	40,498,019
....	4,145	46,807	4,035	2,549	90,193	184	13,144	108,539,444	44,479,970
....	4,108	47,905	3,959	2,828	88,374	196	12,625	107,473,933	43,126,281
....	3,934	49,727	4,369	2,768	87,454	159	16,565	116,637,765	45,819,488
....	3,850	55,881	4,390	2,829	91,736	135	25,245	120,815,503	48,635,080
....	4,127	52,892	4,913	2,535	103,006	183	24,568	121,298,607	52,512,062
....	4,494	55,969	6,249	3,052	117,413	161	21,486	137,117,926	56,419,567
....	4,200	59,473	4,878	2,904	126,649	114	32,949	143,185,691	62,546,671
....	3,802	53,958	6,065	2,822	89,009	128	32,995	147,671,149	67,615,200
....	4,179	56,096	6,150	2,911	77,537	149	37,187	148,788,050	68,657,606
....	4,406	58,349	6,151	3,066	81,208	167	41,100	152,827,777	69,473,883
....	3,076	.....	4,859	2,981	81,902	160	42,000	160,747,580	73,760,867

ear.	Cobalt, Nickel and Bismuth Ores.	Copper.				Gold.	Graphite.
		Ore.	Matte. (*b*)	Ingots.	Sulphate.		
.....	3,157	702,781	62	30,695	4,352	$1,891,974	4,593
.....	1,270	733,619	95	34,634	5,142	1,731,153	5,196
.....	4,495	747,749	4,207	30,929	5,076	2,030,200	9,248
.....	10,479	777,339	365	31,317	5,192	1,830,835	4,435
.....	12,433	761,921	447	30,578	4,997	1,770,361	5,023
.....	14,607	772,695	583	31,214	5,200	1,709,223	3,720
.....	14,016	798,214	641	30,264	6,584	1,819,538	3,784
.....	10,848	793,488	1,635	31,713	6,988	2,611,812	4,921
.....	.....	768,523	771	32,275	6,757	2,931,750	4,055
.....	2,809	771,227	527	31,946	5,284	3,111,379	4,033
.....	8,535	727,384	328	30,001	7,117	3,162,544	4,844
.....	10,388	798,618	2,242	31,126	6,211	3,366,986	6,774
.....	10,313	925,957	3,735	34,926	5,210	3,227,250	7,415
.....	.....	868,600	1,142	37,452	7,353	3,302,300	11,298

ear.	Iron and Steel.				Lead.		
	Iron Ore.	Pig Iron. (*c*)	Wrought iron and Steel.	Sulphate. (*d*)	Ore.	Pig.	Litharge.
....	15,901,263	7,312,766	6,941,278	10,422	149,311	132,742	3,857
....	17,989,635	8,153,133	7,532,524	10,931	144,370	129,225	3,562
....	18,964,294	8,520,540	7,377,275	10,913	148,257	121,513	3,088
....	16,570,182	7,880,087	7,033,438	11,148	153,341	123,098	4,101
....	17,963,591	8,529,900	8,317,231	.....	167,855	140,331	4,197
....	21,230,650	10,017,901	9,226,898	12,243	165,991	145,319	4,428
....	22,047,393	10,058,273	9,239,302	13,585	164,440	137,580	4,332
....	23,444,073	10,875,061	10,309,690	12,949	152,725	152,590	3,786
....	26,734,570	12,292,819	11,307,807	13,376	140,914	150,741	4,137
....	27,697,127	12,875,159	12,063,632	14,033	147,272	142,271	4,325
....	24,278,151	11,805,321	10,930,933	15,738	156,861	164,079	5,339
....	25,505,409	12,917,653	11,719,239	21,838	159,852	167,920	3,059
....	28,709,700	14,502,183	12,988,177	18,677	148,497	159,851	3,581
....	29,879,361	15,280,527	14,384,271	17,002	140,154	161,287	3,687

Year.	Magnesium Salts.		Manganese Ore.	Petroleum.	Potassium Salts.				
	Chloride.	Sulphate.			Chloride.	Kainite. (f)	Sulphate.	Potassium-Magnesium Sulphate.	Other than Kainite.
1898 ...	19,819	30,295	43,354	25,989	191,347	1,103,643	18,853	13,982	1,105,212
1899 ...	21,370	39,540	61,329	27,027	207,506	1,108,159	26,103	9,765	1,384,972
1900 ...	19,397	48,591	59,204	50,375	271,512	1,227,873	30,853	15,368	1,822,738
1901 ...	21,018	46,714	56,691	44,095	294,666	1,498,569	37,394	15,612	2,036,325
1902 ...	19,658	39,262	49,812	49,725	267,512	1,322,623	28,278	18,147	1,962,384
1903 ...	22,990	37,844	47,994	62,680	280,248	1,557,243	36,674	23,631	2,073,720
1904 ...	25,730	39,412	52,886	89,620	297,238	1,905,893	43,959	29,285	2,179,471
1905 ...	29,017	58,568	51,463	78,869	373,177	2,387,643	47,994	34,222	2,655,845
1906 ...	38,468	43,041	52,485	81,350	403,387	2,720,594	54,490	35,211	2,821,073
1907 ...	32,891	41,105	73,105	106,379	473,138	2,624,412	60,292	33,368	3,124,955
1908 ...	29,775	42,977	67,692	141,900	511,258	2,715,487	55,756	33,149	3,383,535
1909 ...	31,526	53,812	77,177	143,244	624,994	3,181,349	68,539	38,722	3,969,554
1910 ...	32,207	57,314	80,560	145,168	741,259	4,249,667	84,583	37,438	4,062,004
1911 ...	36,764	55,179	87,297	142,992	838,420	4,425,497	107,631	42,253	5,181,379

Year.	Pyrites.	Salt.		Silver and Gold Ore.	Silver. Kg.	Sodium Sulphate.	Sulphur.	Sulphuric Acid.
		Rock.	Evaporated.					
1898....	136,849	807,792	565,683	14,702	480,578	69,111	1,954	754,151
1899....	144,623	861,123	571,058	13,506	467,590	79,062	1,663	813,141
1900....	169,447	926,563	587,464	12,593	415,735	90,468	1,445	829,376
1901....	157,433	985,050	578,751	11,577	403,796	76,066	963	835,000
1902....	165,225	1,010,412	572,846	11,724	430,610	90,742	.....	894,409
1903....	170,867	1,095,541	598,394	11,467	396,253	83,087	219	928,190
1904....	174,782	1,079,868	621,064	10,405	389,827	75,171	209	963,384
1905....	185,368	1,165,495	612,062	10,286	399,775	68,454	205	1,228,211
1906....	196,971	1,235,041	635,171	8,066	393,442	81,175	178	1,335,128
1907....	196,351	1,285,137	665,547	8,280	386,933	80,347	176	1,402,398
1908....	219,456	1,331,984	665,651	7,653	407,185	72,667	811	1,391,653
1909....	198,688	1,370,668	647,939	7,510	400,562	71,813	1,185	1,434,709
1910....	215,708	1,424,064	669,120	6,646	420,003	84,787	1,272	1,616,336
1911....	217,459	1,436,492	645,991	4,890	439,580	82,664	1,251	1,534,455

Year.	Tin.			Uranium and Tungsten Ores.	Zinc.		
	Ore.	Block.	Chloride.		Ore.	Spelter.	Sulphate.
1898	51	993	.....	50	641,706	154,867	6,104
1899	72	1,481	.....	50	664,536	153,155	7,117
1900	80	2,031	(g) 143	43	639,215	155,790	6,027
1901	82	1,464	(g) 135	43	647,496	166,283	5,552
1902	104	2,779	.....	31	702,504	174,927	.....
1903	110	3,065	1,064	35	682,853	182,548	5,994
1904	99	4,216	816	23	715,732	193,058	6,185
1905	123	5,233	811	26	731,271	198,208	5,896
1906	.....	6,597	987	.....	704,590	205,691	6,092
1907	.....	5,838	1,812	3	698,425	208,195	5,145
1908	111	6,374	2,266	42	706,441	216,490	5,310
1909	124	8,994	3,247	98	723,565	219,766	5,574
1910	125	11,395	3,391	95	718,316	221,395	6,308
1911	.....	12,412	3,749	.....	699,970	235,776	6,703

(a) From the *Vierteljahrshefte zur Statistik des Deutschen Reichs.* Where gold is reported 1 mark = $0.238. (b) Includes black copper. (c) Includes ferromanganese and spiegeleisen. (d) Contains a small quantity of copper and iron sulphate mixed. (f) Compound of potassium chloride and magnesium sulphate. (g) Includes nickel sulphate.

MINERAL PRODUCTION OF BADEN. (a)
(In metric tons and dollars; 4 marks = $1.)

ar.	Aluminum Sulphate.	Barytes.	Coal.	Gypsum.	Manufacturers of Iron.		
					Cast, Foundry.	Steel.	Wrought.
7....	1,824	400	4,752	40,702	36,235	3,875	1,167
8....	2,051	1,100	4,133	28,037	39,988	3,875	1,167
9....	2,153	2,430	4,700	29,419	53,608	3,830	1,402
0....	2,286	2,970	4,930	26,381	50,102	3,532	1,364
1....	2,260	3,991	3,650	28,183	40,100	8,739	1,158
2....	2,374	6,234	2,078	33,150	40,973	12,663	1,052
3....	2,498	8,857	1,990	29,423	45,233	7,666	863
4....	2,392	9,078	1,485	26,984	64,320	7,687	783
5....	2,581	11,094	668	28,823	74,128	8,053	842
6....	2,583	11,984	1,000	25,643	81,387	11,068	466
7....	2,644	9,303	2,075	29,153	98,430	10,818	533
8....	2,524	(c) 8,554	2,473	35,217	83,724	10,430	602
9....	2,329	(c) 15,186	2,356	36,621	83,458	11,643	484
0....	2,141	(c) 13,832	1,650	41,078	95,795	12,134	239
1....	2,554	15,014	(b)	42,408	107,784	11,909	244

Year.	Lead Ores.	Salt.	Sulphuric Acid.	Tripoli.	Zinc Ore.
7.....	(b)	31,445	13,365	9	(b)
8.....	(b)	31,445	31,365	6	(b)
9.....	(b)	31,197	13,660	12	357
0.....	67	32,699	15,938	9	3,004
1.....	369	32,835	17,081	8	2,870
2.....	450	32,192	19,265	11	2,958
3.....	350	32,383	19,755	11	3,171
4.....	265	32,148	35,517	12	5,063
5.....	264	31,393	40,781	12	4,046
6.....	246	31,288	38,655	15	1,466
7.....	278	32,078	42,831	25	2,198
8.....	329	33,993	41,455	13	2,793
9.....	372	34,040	42,219	14	3,253
0.....	419	32,729	56,069	7	3,345
.....	191	32,118	47,796	14	3,067

(a) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem* schen *Staate.* (b) Not reported. (c) Includes fluorspar.

MINERAL PRODUCTION OF BAVARIA. (a)
(In metric tons; 4 marks = $1.)

ar.	Barytes.	Kaolin.	Coal.	Coal. (Lignite).	Copperas and Other Sulphate.	Emery.	Feldspar.	Fluor-spar.	Graphite.
6..	3,397	19,080	900,080	35,934	601	249	1,315	5,218	5,248
7..	3,365	24,086	917,022	39,043	981	217	1,689	4,904	3,861
8..	4,339	29,196	964,611	38,663	886	280	1,949	4,440	4,593
9..	6,214	25,822	1,004,421	35,736	900	399	287	3,631	5,196
0..	10,515	58,795	1,185,296	39,165	916	414	460	7,456	9,248
1..	8,711	35,450	1,203,792	25,224	590	366	788	5,220	4,435
2..	8,034	92,073	1,233,568	27,337	691	225	447	5,460	5,023
3..	8,642	88,140	1,356,556	25,189	814	220	1,060	3,410	3,719
4..	9,411	95,160	1,341,925	53,517	893	265	1,866	4,770	3,784
5..	10,030	99,910	1,317,951	154,128	844	255	1,710	4,413	4,921
6..	19,817	98,138	1,381,175	140,290	836	320	1,740	5,570	4,055
7..	21,500	115,387	1,495,895	286,256	850	326	2,125	4,780	4,033
8..	17,195	68,551	647,639	1,209,110	910	245	5,859	5,480	4,844
9..	17,920	187,312	694,191	1,242,088	1,094	305	3,151	5,580	6,774
0	24,711	107,660	713,994	1,299,970	1,333	270	2,620	5,132	7,415

Year.	Gypsum.	Iron.						Lithographic Limestone.
		Ore.	Bar.	Cast, 1st Fusion.	Cast, 2d Fusion.	Pig.	Steel.	
1897....	26,153	172,699	58,200	138	78,008	83,418	115,530	13,941
1898....	25,688	171,987	58,342	97	84,227	84,144	120,623	12,028
1899....	29,727	181,981	61,415	(b)	92,459	83,821	134,007	11,962
1900....	35,484	178,441	49,727	29	89,692	82,327	135,411	16,030
1901....	3,581	158,820	29,978	76	76,191	72,071	109,464	9,500
1902....	31,701	157,375	38,429	56	81,874	83,123	115,354	.....
1903....	30,894	162,500	36,853	41	89,804	90,168	127,141	9,890
1904....	22,766	180,342	37,780	40	108,025	92,200	125,483	13,836
1905....	46,247	182,389	36,459	24	112,875	94,242	134,755	11,360
1906....	50,763	203,596	38,508	.....	122,115	97,812	150,129	.....
1907....	48,975	277,280	36,883	*Nil.*	138,659	98,143	150,148	11,590
1908....	51,314	278,681	30,740	*Nil.*	128,234	131,404	176,085	9,858
1909....	51,630	279,514	33,448	*Nil.*	130,129	134,133	219,606	9,420
1910....	54,397	303,844	30,881	*Nil.*	148,261	133,679	249,198	9,790

(a) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (b) Not reported.

Year.	Marl. (For Cement).	Mineral Paint and Chalk.	Pyrites.	Rock Salt.	Soapstone.	Sodium Sulphate.	Sulphuric Acid.
1897....	97,831	8,673	2,211	1,161	2,464	2,318	7,041
1898....	110,757	8,748	2,304	736	1,912	2,332	103,385
1899....	220,716	9,287	2,516	802	2,197	1,570	123,273
1900....	180,032	11,507	2,120	1,298	1,977	1,821	123,910
1901....	76,663	84,929	2,649	1,319	2,291	1,893	115,775
1902....	178,301	13,947	2,635	832	.....	.....	.....
1903....	200,407	19,486	2,324	879	1,866	.....	.....
1904....	170,698	19,107	3,427	1,139	1,709	.....	.....
1905....	231,310	18,285	3,301	911	1,872	.....	.....
1906....	230,271	22,304	3,918	1,053	.....	.....	.....
1907....	230,583	21,219	5,085	1,393	1,999	1,439	161,868
1908....	307,820	21,310	4,037	1,285	2,199	1,743	149,079
1909....	276,974	21,692	2,952	1,860	2,329	1,265	178,371
1910....	273,727	22,357	4,466	1,192	3,083	1,416	171,131

MINERAL PRODUCTION OF PRUSSIA. (a)

(Metric tons; 4 marks = $1.)

Year.	Alum Shale.	Antimony and Alloys.	Arsenic Products.	Arsenic Ore.	Asphalt.	Boracite.	Cadmium. *Kg.*	Coal.
1897....	129	1,552	1,924	3,377	11,466	185	15,531	84,253,393
1898....	107	2,612	1,624	3,298	12,822	216	14,943	89,593,528
1899....	145	3,003	1,469	3,265	16,458	171	13,608	94,740,829
1900....	103	3,025	1,585	3,531	23,891	217	13,533	101,966,158
1901....	611	2,404	1,446	3,050	26,450	164	13,144	101,203,807
1902....	219	3,542	1,514	2,909	28,035	172	12,625	100,115,315
1903....	580	3,224	1,583	3,538	23,518	135	16,565	108,809,384
1904....	106	2,774	1,573	3,527	26,348	115	25,245	112,755,621
1905....	97	2,795	1,493	4,022	28,872	151	24,568	113,000,657
1906....	634	2,953	1,551	5,430	32,270	124	21,486	128,295,948
1907....	154	3,515	1,591	4,224	39,243	90	32,949	134,044,080
1908....	80	3,596	1,646	5,015	27,444	105	32,995	139,002,378
1909....	60	3,841	1,849	5,731	19,509	123	37,187	139,906,194
1910....	46	4,969	2,068	5,789	21,595	138	41,058	143,771,612

Year.	Coal. (Lignite.)	Cobalt Ore.	Cobalt Products.	Copper.	Copper and Iron Sulphate.	Copper Ore.	Copper Matte.
1897....	24,222,911	121	51	25,997	225	690,338	274
1898....	26,035,814	34	44	27,216	120	691,866	62
1899....	28,418,598	17	46	20,902	154	722,884	95
1900....	34,007,542	4	52	27,156	113	747,601	4,207
1901....	37,491,412	36	66	28,422	78	765,241	281
1902....	36,228,285	76	74	27,893	119	751,496	346
1903....	38,462,766	65	87	28,386	110	761,188	488
1904....	41,153,576	41	85	27,450	95	782,049	601
1905....	44,148,751	22	99	28,874	102	769,381	1,052
1906....	47,912,721	7	98	29,166	94	755,812	525
1907....	52,660,597	*Nil.*	109	28,945	64	755,203	499
1908....	55,456,860	*Nil.*	100	27,301	50	711,921	296
1909....	56,029,554	*Nil.*	93	28,523	55	798,448	1,935
1910....	56,694,291	*Nil.*	94	31,485	44	903,834	3,749

Year.	Copper Sulphate.	Epsom Salt.	Gold. *Kg.*	Iron.	Iron Ore.	Iron Sulphate.	Lead.
1897.....	2,689	2,248	1,087.1	4,892,059	4,183,536	9,064	108,880
1898.....	1,701	2,061	1,036.3	5,176,943	4,020,809	9,144	119,346
1899.....	1,586	1,793	1,016.4	5,644,614	4,295,575	10,186	116,995
1900.....	1,660	1,511	1,076.6	5,781,892	4,268,069	10,225	112,738
1901.....	1,951	1,952	1,087.1	5,315,628	3,831,670	10,239	113,939
1902.....	1,937	761	1,138.0	5,633,089	3,362,887	11,214	127,283
1903.....	2,254	421	949.5	6,614,768	3,786,743	11,086	133,405
1904.....	3,364	289	1,081.9	6,573,507	3,757,651	12,524	128,294
1905.....	3,065	338	1,034.9	7,106,975	4,130,210	12,075	143,270
1906.....	2,724	144	750.2	8,154,880	4,713,928	12,473	140,690
1907.....	2,129	263	771.0	8,266,300	5,077,773	13,014	132,366
1908.....	3,116	398	786.6	7,989,260	4,311,593	14,062	153,541
1909.....	2,500	395	588.2	8,410,824	4,389,950	18,295	156,533
1910.....	1,749	162	566.3	9,995,012	4,823,606	16,119	150,275

Year.	Lead ore.	Litharge.	Manganese Ore.	Nickel.	Nickel Ore.	Nickel Sulphate.	Ocher and Mineral Paints.
1897.....	133,158	1,999	45,254	898	204	167	2,400
1898.....	133,637	2,360	42,232	1,108	79	127	2,376
1899.....	128,942	2,482	60,379	1,115	91	123	2,770
1900.....	133,483	2,366	58,016	1,376	3,896	115	2,850
1901.....	139,285	2,885	55,866	1,660	9,922	120	2,800
1902.....	152,282	2,516	48,882	1,605	11,816	159	2,780
1903.....	151,746	2,710	47,110	1,945	14,058	173	2,850
1904.....	150,328	2,517	52,092	2,333	13,518	207	3,200
1905.....	138,928	2,272	51,048	2,631	10,432	220	3,170
1906.....	127,322	2,744	51,881	2,648	7,472	187	3,635
1907.....	133,528	2,959	72,442	2,093	7,557	189	3,707
1908.....	141,316	4,190	67,241	2,622	8,238	181	3,183
1909.....	158,811	2,365	76,741	3,186	10,095	163	3,435
1910.....	146,830	3,076	80,325	3,497	10,053	213	142

Year.	Petroleum.	Potassium salts.		Pyrites.
		Kainite.	All other.	
1897.....	2,600	716,348	640,236	121,766
1898.....	2,545	744,240	718,957	128,077
1899.....	3,405	744,657	941,055	134,564
1900.....	27,731	857,271	1,264,993	159,186
1901.....	24,098	1,068,237	1,131,703	148,457
1902.....	29,520	943,450	1,344,541	155,410
1903.....	41,733	1,118,270	1,344,038	159,234
1904.....	67,604	1,261,930	1,447,323	163,209
1905.....	57,741	1,580,530	1,734,033	174,641
1906.....	59,196	1,923,088	1,937,181	186,849
1907.....	80,255	1,839,409	2,070,978	184,962
1908.....	113,002	2,037,203	2,192,188	204,992
1909.....	113,518	2,431,401	2,436,319	189,773
1910.....	110,996	3,119,400	2,524,565	202,648

Year.	Silver. *Kg.*	Silver and Gold Ores.	Sulphur.	Sulphuric Acid.	Tin.
1897.....	289,960	6	2,091	484,289	912
1898.....	291,969	43	1,757	531,838	979
1899.....	293,858	7	1,419	573,733	1,461
1900.....	266,577	1	1,207	593,109	2,010
1901.....	246,286	6	772	609,041	1,443
1902.....	273,901	17	250	677,798	2,753
1903.....	255,722	13	16	724,784	3,042
1904.....	252,020	8	16	868,424	4,193
1905.....	266,072	4	14	921,219	5,196
1906.....	264,427	239	16	980,188	6,570
1907.....	249,348	34	7	1,004,599	5,819
1908.....	274,154	7	706	997,931	5,330
1909.....	271,779	2	1,096	1,006,787	8,943
1910.....	284,004	0.2	1,239	1,164,015	11,345

(a) From *Zeitschrift fur das Berg, Hutten, und Salinenwesen.*

MINERAL IMPORTS OF GERMA

Year.	Aluminum Refined and Crude.	Ammonium Sulphate.	Antimony.	Antimony and Arsenic Ores.	Asbestos, Crude.	As
1899 ...	.....	28,868	.....	.....	.....	6
1900 ...	943	23,105	1,461	1,291	6,850	8
1901 ...	1,090	44,408	1,494	1,098	5,500	6
1902 ...	1,100	42,252	1,495	1,231	3,415	8
1903 ...	1,155	35,168	2,281	1,741	5,727	9
1904 ...	2,422	35,166	2,003	1,687	5,251	8
1905 ...	3,252	48,005	1,680	567	7,830	
1906 ...	3,886	35,366	2,044	2,417	9,828	1
1907 ...	3,974	33,522	2,496	4,913	11,096	
1908 ...	3,204	47,265	2,670	2,073	10,034	
1909 ...	8,696	58,132	2,719	2,912	11,928	
1910 ...	9,892	31,400	2,982	3,347	11,729	
1911 ...	10,454	24,463	3,607	6,429	2,334	13

Year.	Borax.	Bauxite.	Calcium Carbide.	Cement.	Chalk (*d*), Crude White.	Chrome Ore.	Coal.		Coke.
							Bitum., Arthracite Cannel.	Lignitic.	
1899 ...	.....	.....	.....	63,388	.....	.....	6,220,489	8,616,751	462,577
1900 ...	2,403	29,383	7,703	79,303	(*d*)63,929	18,728	7,384,049	7,960,313	512,690
1901 ...	2,537	24,113	9,526	87,262	(*d*)29,611	18,222	6,297,389	8,108,943	400,197
1902 ...	2,057	26,698	11,287	52,018	(*d*)26,408	10,152	6,425,658	7,882,010	382,488
1903 ...	2,567	22,316	14,081	49,870	(*d*)33,362	13,919	6,766,513	7,962,123	432,819
1904 ...	2,603	27,849	14,840	60,188	(*d*)32,581	18,132	7,299,042	7,669,099	550,302
1905 ...	2,802	39,137	17,256	148,118	(*d*)35,529	11,998	9,399,693	7,945,261	713,619
1906 ...	3,044	43,177	22,819	233,119	18,871	17,124	9,253,711	8,430,441	565,561
1907 ...	2,014	59,989	25,834	241,475	16,035	19,508	13,729,849	8,963,103	584,220
1908 ...	1,903	48,064	29,024	168,504	17,606	16,974	11,661,503	8,581,966	575,091
1909 ...	2,550	45,543	26,956	224,178	15,924	22,018	12,198,634	8,166,479	673,012
1910 ...	3,181	56,287	30,712	242,663	17,700	24,470	11,195,593	7,397,719	622,452
1911 ...	5,358	37,155	36,994	253,023	23,717	16,022	10,913,948	7,069,064	598,958

Year.	Peat and Peat Coke.	Briquettes.	Cobalt and Nickel Ore.	Copper.				Copperas	Cryolite.
				Ore. and Matte.	Ingots.	Bars, Wire and Sheets.	Sulphate.		
1899 ...	.....	.....	.....	.....	70,091	610	.....	.....	.....
1900 ...	19,807	137,153	13,032	10,930	83,503	906	2,369	**752**	**1,460**
1901 ...	15,102	92,037	12,186	4,614	58,620	786	1,211	**501**	**1,249**
1902 ...	16,696	81,854	14,630	14,630	76,050	540	2,499	**807**	**1,332**
1903 ...	14,640	84,635	36,927	13,714	83,261	568	1,691	**778**	**1,082**
1904 ...	9,071	125,477	14,555	7,949	110,231	719	1,735	**765**	**1,139**
1905 ...	11,439	191,753	39,590	10,137	102,218	927	2,180	**666**	**1,143**
1906 ...	19,428	162,650	22,557	9,941	126,071	409	1,702	**621**	(*k*)
1907 ...	15,238	195,403	29,296	19,295	124,116	772	4,519	**1,165**	(*k*)
1908 ...	15,266	192,391	17,402	17,456	157,669	952	5,078	**7,234**	(*k*)
1909 ...	13,208	211,058	10,186	26,488	154,673	416	6,550	**5,954**	(*k*)
1910 ...	16,188	241,267	9,937	22,194	181,551	403	3,952	**4,336**	(*k*)
1911 ...	14,517	210,933	14,897	23,327	191,590	520	4,145	**3,323**	(*k*)

Year.	Gold, Silver and Platinum Ores.	Graphite	Gypsum.	Iodine.	Iron.		Lead.		
					Ore.	Pig.	Ore.	Pig and Scrap.	Lead White.
1899 ...	7,597	23,400	.....	191	4,165,372	612,652	.....	55,635	703
1900 ...	9,153	22,495	7,571	236	4,107,840	726,712	51,338	70,252	698
1901 ...	8,764	17,374	7,622	266	4,370,022	267,503	100,196	52,886	423
1902 ...	6,585	19,392	8,177	220	3,957,403	143,040	71,078	39,006	357
1903 ...	4,386	20,953	8,328	320	5,225,336	158,347	67,573	52,440	442
1904 ...	5,960	23,533	9,550	272	6,061,127	178,256	83,807	61,388	622
1905 ...	6,225	26,143	11,247	377	6,085,196	158,700	92,667	78,528	2,488
1906 ...	4,819	28,175	11,062	297	7,629,730	409,083	90,027	71,191	2,342
1907 ...	3,601	29,405	14,662	147	8,476,076	443,624	137,861	75,200	3,037
1908 ...	1,922	34,491	14,599	194	7,732,949	252,779	133,597	77,218	3,558
1909 ...	1,759	29,191	11,285	369	8,366,599	134,230	111,017	76,930	2,890
1910 ...	2,552	30,733	10,873	363	9,816,822	136,330	112,151	81,541	2,780
1911 ...	2,947	32,812	11,973	303	10,812,595	129,850	143,598	100,515	3,938

Year.	Magnesite.	Manganese Ore.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum products.		Phosphorus.
						Illuminating Oil.	Lubricating Oil.	
1899....	.....	196,825	.....	1,391	.....	963,943	106,624	....
1900....	13,920	204,420	12,107	1,712	3,457	989,361	124,505	381
1901....	8,897	222,010	9,403	1,947	1,981	985,904	118,999	313
1902....	12,237	204,647	7,719	1,458	1,585	1,006,829	125,667	350
1903....	14,958	223,709	9,888	1,507	1,663	1,067,697	147,837	222
1904....	15,877	255,760	10,494	1,712	1,300	1,076,324	142,929	220
1905....	19,459	262,311	11,473	1,955	1,114	1,070,252	143,926	198
1906....	25,527	331,171	3,960	3,478	1,303	984,134	180,989	208
1907....	30,857	393,327	2,166	2,182	1,653	1,115,205	226,609	165
1908....	28,305	334,133	1,635	3,058	1,447	1,123,632	216,887	141
1909....	29,994	384,445	13,804	3,745	1,447	1,085,839	216,987	179
1910....	40,218	487,872	15,100	4,606	1,729	1,135,886	230,516	169
1911....	47,930	420,709	15.703	2,598	1,591	955,482	260,242	200

Year.	Phosphate Rock.	Potassium salts.							Pumicestone (g)	Pyrites.
		Chloride.	Cyanide. (f)	Iodide.	Nitrate.	Carbonate.	Hydroxide.	Sulphate.		
1899....	407,457	443	3	9	1,785	1,737	.....	533	.....	437,732
1900....	320,138	484	2	10	2,047	1,522	283	856	2,154	457,679
1901....	351,155	462	2	1,529	1,758	1,529	165	680	2,336	488,633
1902....	430,043	261	3	10	1,889	2,112	42	266	2,070	482,095
1903....	461,092	40	3	8	2,163	1,850	52	81	2,697	519,317
1904....	508,634	47	2	10	2,349	1,955	61	121	3,000	503,503
1905....	501,048	223	3	30	2,156	1,693	24	131	3,240	552,184
1906....	531,195	181	3	18	1,918	2,099	44	257	5,463	579,355
1907....	579,505	1,615	1	8	1,815	2,304	92	141	5,443	742,526
1908....	736,127	49	4	7	2,200	1,773	50	169	6,154	659,871
1909....	663,400	55	2	4	2,853	1,750	64	101	6,639	691,213
1910....	723,271	72	2	8	1,979	2,366	63	66	7,446	792,735
1911....	831,027	1,160	33	13	2,114	2,616	67	61	9,659	862,214

Year.	Quicksilver.	Salt.	Slag and Slag Wool.	Sodium Salts.			Strontianite. (n)
				Soda, Calcined.	Nitrate (Chile Saltpeter.)	Sulphate and Sulphite.	
1899	572	22,040	892,764	515	526,944	.....	.....
1900	555	21,738	974,947	373	484,544	9,450	8,701
1901	651	23,901	733,931	178	529,568	7,921	19,739
1902.....	648	26,404	831,282	121	467,024	7,308	34,035
1903	674	20,118	877,394	114	467,130	6,058	24,183
1904	691	18,743	846,738	179	506,172	9,598	18,055
1905	729	20,726	888,665	143	540,916	4,752	13,720
1906	698	16,997	813,388	189	593,218	7,405	5,212
1907.....	831	23,109	568,046	257	591,131	10,446	5,595
1908	648	24,975	562,853	293	604,457	4,404	4,211
1909	723	19,319	492,771	181	665,450	9,214	4,277
1910	836	30,443	766,320	105	749,945	9,302	4,852
1911	919	29,067	685,943	559	730,939	5,976	3,251

ear.	Sulphur.	Sulphuric Acid.	Super-phosphate.	Tin, Crude.	Zinc.			
					Ore.	Spelter.	Drawn or Rolled.	Zinc-white, Zinc-gray, Lithophone.
9....	31,196	.....	.....	12,253	57,880	23,691	95	4,226
0....	40,689	20,634	72,062	12,454	68,982	24,263	145	4,884
1....	32,750	18,502	107,365	12,910	75,533	21,250	306	3,673
2....	32,798	22,205	109,374	13,760	61,407	25,946	134	3,986
3....	41,545	13,418	82,740	13,925	67,156	25,749	237	4,667
4....	41,030	16,087	91,288	14,352	93,515	26,389	151	6,461
5....	39,989	33,837	109,666	13,501	126,577	29,583	54	7,802
6....	41,390	74,536	76,384	14,098	178,953	39,314	97	9,140
7....	44,700	59,753	62,877	12,814	184,703	28,459	134	10,189
8....	44,066	61,391	71,879	14,039	199,840	32,622	286	7,080
9....	42,941	74,384	80,512	13,537	201,110	44,514	99	7,002
0....	46,796	86,743	78,873	14,297	240,584	39,328	246	9,239
1....	46,054	99,653	71,119	14,500	262,399	48,355	467	7,697

MINERAL EXPORTS OF GERMANY. (*a*)

ear.	Aluminum Refined and crude.	Aluminum Wares, etc.	Aluminum Sulphate.	Ammonium.		Antimony and Arsenic Ores.	Antimony.	
				Carbonate and Chloride.	Sulphate.		Metallic.	Salts.
9....	.....	2,312	.....	.....	1,553	....	....	.....
0....	269	2,398	29,372	3,196	2,431	284	131	786
1....	282	2,270	31,171	3,196	9,842	283	76	826
2....	410	2,608	34,005	3,351	5,744	410	105	954
3....	353	2,865	28,513	2,778	5,592	427	83	873
4....	407	3,077	29,311	3,106	10,696	486	250	964
5....	1,192	3,476	34,776	3,579	27,589	287	218	1,097
6....	1,111	1,321	25,937	3,555	37,288	548	221	997
7....	1,119	1,142	24,759	3,118	57,439	930	255	1,168
8....	590	642	22,376	1,161	73,186	588	146	1,030
9....	492	1,273	58,723	1,093	58,722	577	160	1,090
0....	616	1,755	93,069	1,500	93,069	571	198	1,047
1....	768	1,331	30,107	4,895	74,410	612	298	1,208

ear.	Arsenic.		Asbestos.	Barytes. (*b*)	Barium.		Bauxite.	Borax.	Bromine.
	Metallic.	White, etc.	Crude.		Chloride and Salts of.	White.			
9....	.....	.....	.....	.....	.....	.....	.....	.....	.....
0....	14	1,573	496	59,012	5,927	2,717	44	2,894	191
1....	28	1,534	638	67,526	6,803	2,765	137	2,563	228
2....	46	2,036	709	56,026	7,358	2,922	32	2,836	153
3....	32	1,903	513	72,455	8,417	3,187	19	2,779	155
4....	50	1,956	738	69,564	8,596	3,777	21	2,741	208
5....	40	1,753	1,173	81,134	9,550	4,382	6	2,720	156
6....	.....	2,282	1,938	90,819	6,541	10,721	398	2,795	172
7....	(*m*)45	1,733	1,707	111,209	4,189	8,454	517	3,049	118
8....	(*m*)65	1,956	1,345	91,111	3,389	5,190	783	2,379	227
9....	(*m*)54	1,012	1,740	90,555	5,340	4,888	1,116	2,735	206
0....	(*m*)57	1,507	1,537	114,264	6,403	5,528	863	2,916	225
1....	71	1,973	1,503	128,452	6,180	6,834	960	3,199	229

Year.	Coke.	Peat and Peat Coke.	Briquets.	Cobalt and Nickel Ore.	Copper.			
					Ore and Matte.	Bars, Sheets and Wire.	Ingots.	Sulphat
1899....	2,137,985	.....	.....	...	.....	.....	7,061	.....
1900....	2,229,188	8,849	550,222	186	25,686	9,787	5,505	1,881
1901 ..	2,096,931	11,588	529,765	96	26,678	7,700	5,097	1,942
1902....	2,182,383	13,410	697,799	3	17,031	10,599	4,678	1,398
1903....	2,523,351	16,986	895,145	1	15,986	10,715	4,333	1,880
1904....	2,716,855	14,830	917,526	83	19,235	12,594	4,223	2,251
1905....	2,761,080	16,009	936,694	107	28,908	10,006	5,958	2,180
1906....	3,415,347	15,680	1,095,029	(i)	6,414	10,728	7,241	3,018
1907....	3,792,580	25,746	1,260,135	(i)	20,950	13,411	6,113	2,606
1908....	3,577,496	26,817	1,493,054	(i)	21,729	17,209	6,868	2,984
1909....	3,444,791	23,579	1,620,460	(i)	22,437	15,395	6,495	2,290
1910....	4,125,898	20,360	1,988,177	(i)	23,751	19,351	7,654	2,108
1911....	4,555,477	35,855	2,477,492	(i)	27,396	19,669	7,106	3,246

Year.	Bromine Salts.	Calcium.		Cement.	Chalk. Crude White.	Chromium.		Coal.	
		Carbide.	Chloride.			Ore.	Alum.	Bituminous Anthracite and Cannel.	Lignite
1899 ...	.....	.....	.....	580,255	.....	.....	.....	13,943,174	20,923
1900 ...	255	224	1,315	600,386	(d)11,860	427	1,192	15,275,805	32,795
1901 ...	249	275	888	560,612	(d)14,134	581	1,299	15,269,267	21,738
1902 ...	357	126	1,346	699,378	(d) 8,475	846	1,758	16,101,141	21,786
1903 ...	435	335	1,831	742,381	(d)12,211	37	1,921	17,389,934	22,696
1904 ...	411	608	2,381	635,248	(d)11,359	47	2,432	17,996,726	22,133
1905 ...	634	709	2,831	675,664	(d)13,081	43	2,507	18,156,998	20,178
1906 ...	643	545	(l)	736,579	4,287	(h) 36	2,942	19,550,964	18,759
1907 ...	655	918	(l)	692,982	2,919	(h) 149	3,110	20,056,503	22,065
1908 ...	506	844	(l)	528,847	2,108	(h) 110	3,215	21,190,777	27,877
1909 ...	486	968	(l)	611,893	4,395	(h)5,023	3,023	23,350,705	20,515
1910 ...	323	1,482	(l)	725,356	3,359	386	3,998	24,257,651	62,443
1911 ...	.....	2,120	(l)	845,850	1,537	1,795	2,815	27,412,218	58,871

Year.	Copperas.	Cryolite.	Fluorspar.	Graphite.	Gypsum.	Iodine.	Iron.	
							Ore.	Pig.
1899.....	.....	.....	.....	2,703	.....	26	3,119,878	182,081
1900.....	3,829	315	12,749	2,068	39,933	29	3,247,888	128,409
1901.....	4,125	367	13,436	1,667	40,397	27	2,389,870	150,440
1902.....	4,360	486	14,177	1,691	42,859	24	2,868,068	374,359
1903.....	3,986	349	13,028	1,810	51,874	29	3,343,510	418,072
1904.....	3,514	310	13,540	1,815	55,043	30	3,440,846	225,987
1905.....	4,495	286	15,019	1,971	52,886	27	3,698,563	380,874
1906.....	4,712	(k)	15,493	2,013	63,516	46	3,851,791	479,772
1907.....	6,212	(k)	16,624	2,176	70,737	44	3,904,400	273,170
1908.....	4,393	(k)	14,925	2,469	60,992	51	3,067,737	257,849
1909.....	2,232	(k)	14,534	2,377	63,220	59	2,825,007	671,043
1910.....	2,929	(k)	17,988	3,424	89,590	70	2,952,634	786,534
1911.....	4,151	(k)	23,073	3,822	102,754	73	2,581,698	829,588

Year.	Lead.					Lime, Chloride of.	Magnesite.	Magnesium Chloride.	Manganese Ore.
	Ore.	Pig and Scrap.	Litharge.	White.	Red.				
1899	.....	24,491	.....	16,360	.....	.....	.....	.....	7,040
1900	1,309	18,825	3,577	15,126	6,603	25,954	2,392	13,375	2,454
1901	891	20,820	4,876	16,966	7,776	32,705	2,485	16,102	5,584
1902	2,024	23,100	4,072	19,070	8,372	29,694	2,955	14,757	4,528
1903	1,270	30,243	5,175	20,765	7,617	28,849	2,812	17,008	11,138
1904	1,312	23,169	5,410	16,638	7,544	30,078	1,917	16,706	5,536
1905	1,496	32,515	4,466	16,478	8,902	30,667	2,552	21,673	4,116
1906	1,915	27,067	2,493	14,022	9,450	29,485	2,843	26,708	2,555
1907	1,296	38,259	4,470	13,651	9,371	24,946	3,264	29,566	3,554
1908	1,189	29,967	5,242	13,733	9,602	23,895	4,021	27,525	2,333
1909	2,556	31,674	4,750	10,583	9,058	27,314	3,702	31,334	4,488
1910	2,382	31,025	4,865	13,595	10,444	24,716	5,399	35,352	4,557
1911	3,746	32,063	5,436	14,962	11,321	27,107	4,496	43,896	9,615

Year.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum Products. (f)		Phosphorus.	Phosphate Rock.
				Illuminating Oil.	Lubricating Oil.		
1899	.....	295	.....	.....	.....	.....	2,504
1900	13,958	268	1,592	843	1,455	170	1,123
1901	12,671	390	1,700	655	963	149	2,260
1902	14,392	689	1,856	824	1,177	260	1,103
1903	15,161	700	2.027	701	1,975	286	4,342
1904	16,395	1,203	2,447	760	1,763	236	3,222
1905	17,603	1,034	2,757	7,286	1,746	228	3,720
1906	4,290	954	509	673	9,982	228	5,484
1907	4,097	930	692	770	10,552	165	1,494
1908	15,675	1,349	921	1,008	10,852	160	1,196
1909	18,130	1,606	1,300	588	11,621	168	5,429
1910	19,355	1,381	1,642	472	17,141	160	5,044
1911	22,426	1,592	1,295	274	18,905	192	10,591

Year.	Potassium Salts.						Potassium and Potassium-Magnesium Sulphate.	Pumice Stone. (g)	Pyrites.
	Carbonate	Cyanide. (f)	Chloride.	Hydroxide.	Iodide.	Nitrate.			
1899	11,917	1,645	101,045	.....	145	15,146	.....	.....	16,985
1900	15,761	1,338	114,469	15,379	138	14,744	38,125	561	24,936
1901	15,567	2,089	118,959	14,892	145	13,439	37,216	699	23,680
1902	14,041	3,257	106,925	13,804	152	9,734	40,487	691	35,370
1903	13,121	2,017	125,302	13,006	154	9,671	56,455	794	32,611
1904	10,777	3,290	140,765	24,963	174	10,405	64,400	943	30,666
1905	11,963	4,005	156,440	22,246	170	12,140	67,286	939	35,195
1906	12,543	5,049	171,994	21,772	168	11,564	54,557	1,578	35,829
1907	13,314	5,210	173,638	20,254	146	12,668	128,344	2,590	24,183
1908	13,009	4,887	174,345	25,048	127	10,643	181,975	6,055	16,384
1909	13,797	6,283	216,286	27,095	122	12,498	201,393	9,140	11,564
1910	13,099	6,329	266,783	29,094	124	14,728	273,614	7,643	9,871
1911	14,584	6,554	329,751	28,921	133	16,430	392,129	22,075	11,015

Year.	Quicksilver.	Salt.	Slag and Slag Wool.	So		
				Bicarbonate.	Carbonate.	Hydride.
1899...	23	241,036	25,565	.....	.....	.....
1900...	23	236,291	32,494	1,314	1,392	1,91
1901...	27	286,424	27,269	1,086	1,382	4,92
1902...	109	328,324	22,726	954	2,449	5,65
1903...	62	399,184	14,674	1,016	2,982	5,88
1904...	43	347,351	38,587	1,524	3,050	5,08
1905...	48	284,203	28,032	1,881	4,113	5,92
1906...	21	97,878	49,912	2,120	5,860	6,10
1907...	26	92,288	46,680	1,764	2,680	7,46
1908...	26	318,395	74,821	1,713	3,842	7,62
1909...	29	364,107	61,674	1,292	2,968	8,31
1910...	31	370,074	58,832	1,370	5,994	9,29
1911...	36	374,633	88,423	1,598	1,782	10,63

Year.	Sodium and Potassium Salts.		Stassfurt Salts.	Stron
	Chromates.	Sulphides.		Carbonate (n)
1899.....	.....	.....	367,828	.....
1900.....	3,741	2,461	468,277	74
1901.....	2,791	2,763	592,347	384
1902.....	2,656	4,565	499,220	762
1903.....	2,977	5,845	501,385	819
1904.....	2,272	5,489	631,762	613
1905.....	2,133	6,569	852,454	613
1906.....	2,877	6,730	831,293	1,726
1907.....	3,016	8,103	839,889	1,462
1908.....	4,402	6,536	818,677	1,494
1909.....	4,789	7,596	946,457	2,636
1910.....	4,246	8,679	1,181,208	2,456
1911.....	5,193	7,837	1,154,974	595

Year.	Super-Phosphate.	Tin, Crude.	Zinc.		
			Ore.	Spelter and Scrap.	Dr R
1898.....	.....	874	30,408	51,324	14
1899.....	.....	1,121	25,192	46,334	18
1900.....	77,118	1,626	34,941	51,899	16
1901.....	79,190	1,683	14,002	54,490	16
1902.....	77,818	2,271	46,965	70,292	17
1903.....	99,672	2,581	40,458	67,057	15
1904.....	129,925	2,965	40,488	70,063	17
1905.....	115,886	3,259	38,972	67,675	18
1906.....	104,713	4,845	42,546	69,142	17
1907.....	115,049	4,244	34,863	93,649	21
1908.....	125,464	3,707	39,450	75,290	18
1909.....	168,988	5,431	51,994	82,225	18
1910.....	211,812	7,530	59,440	8,803	26
1911.....	221,521	7,582	48,998	81,042	36

(*a*) From *Statistisches Jahrbuch für das Deutsches Reich.* (*b*) I
cipitated chalk. (*e*) Not reported. (*f*) Includes sodium cyan
cludes nickel ore. (*i*) Included under chromium ore. (*k*) Incl
under magnesium chloride. (*m*) Includes all alkali metals. (*n*)

GREECE

The statistics of mineral production in Greece, according to the latest available reports, are summarized in the following table:

MINERAL PRODUCTION OF GREECE. (a)
(In metric tons or dollars; 1 drachma = 20 cents.)

Year.	Chrome Ore.	Emery.	Gypsum.	Iron Ore.	Iron Ore. Manganiferous.	Lead Soft.	Lead Ore. Argentiferous.	Lead. Argentiferous.	Lead. Fume.	Lignite.
1897...	563	3,024	51	260,828	182,850	520	2,815	15,946	2,785	20,018
1898...	1,367	3,932	83	287,100	213,938	305	(b)	18,888	2,655	17,310
1899...	4,386	4,360	81	331,030	294,320	291	(b)	18,768	2,584	12,150
1900...	5,600	6,328	129	279,880	243,920	245	878	16,150	2,045	12,940
1901...	4,580	5,691	671	278,640	196,152	(b)	(b)	17,644	5,292	9,726
1902...	11,680	4,727	172	364,340	170,040	(b)	430	14,048	1,647	6,500
1903...	8,478	5,586	94	531,804	152,740	(b)	(b)	12,361	(b)	8,687
1904...	6,530	6,182	117	422,159	108,319	(b)	(b)	15,186	(b)	13,500
1905...	8,900	6,972	57	465,622	89,687	(b)	(b)	13,729	(b)	11,757
1906...	11,530	7,718	85	680,620	96,382	(b)	(b)	12,308	(b)	11,582
1907...	11,730	10,652	105	768,863	92,970	(b)	(b)	13,814	(b)	11,719
1908...	4,350	7,471	61	531,368	65,757	(b)	(b)	15,892	(b)	8,786
1909...	9,600	8,193	208	475,616	54,926	(b)	(b)	14,948	(b)	3,873
1910...	9,450	12,939	243	535,482	50,015	(c)	8,355	16,710	(b)	1,500

Year.	Magnesite.			Manganese Ore.	Puzzolan.	Sea Salt.	Sulphur.	Zinc Ore.	
	Crude.	Bricks.	Calcined.					Blende.	Calamine, Calcined.
1897....	11,311	826	686	11,868	42,600	20,421	358	3,118	22,817
1898....	14,829	516	129	14,097	70,700	25,250	135	1,139	30,906
1899....	17,184	542	3,087	17,600	46,375	37,125	1,150	1,137	21,770
1900....	17,277	534	807	8,050	49,426	22,411	891	(b)	18,751
1901....	20,348	500	2,009	14,166	80,169	23,079	3,212	454	17,764
1902....	23,020	935	4,730	14,960	32,514	25,200	1,391	(b)	18,670
1903....	28,415	(b)	(b)	9,340	40,978	26,000	1,266	(b)	12,350
1904....	9,133	(b)	(b)	8,549	44,644	27,000	1,225	(b)	19,913
1905....	37,063	(b)	(b)	8,171	41,900	25,201	1,126	(b)	22,562
1906....	40,584	(b)	(b)	10,040	30,622	25,167	(b)	(b)	26,258
1907....	55,816	(b)	(b)	11,139	39,637	26,966	(b)	(b)	30,346
1908....	63,079	(b)	(b)	10,750	52,312	23,988	(b)	28,739	24,101
1909....	56,797	294	16,609	5,374	46,238	29,448	(b)	41,656	30,159
1910....	18,073	295	19,982	41	31,609	25,978	(b)	(f)5,845	28,898

(a) Statistics up to 1903 communicated by E. Grohmann, Seriphos. Statistics for 1903 and subsequent years from *Statistique du Mouvement Minier de Grèce*. The figures represent sales or shipments. (b) Not reported. (c) No sales, but production of lead ore is reported at 240,013 tons in 1909 and 185,207 tons in 1910. (d) Contains 9 per cent. lead and 11 per cent. zinc. (f) Production in 1910 was 37,108 tons.

INDIA

The official statistics of mineral productio
marized in the subjoined table:

MINERAL PRODUCTION OF I

(In metric tons or dollars; £1

Year.	Amber.	Coal.	Chromite.	Diamonds Carats.	Gold. (c)
1897	(b)	4,128,330	.....	.....	$8,041,055
1898	$5,080	4,681,927	.....	.....	7,798,709
1899	755	5,174,752	.....	.....	8,357,087
1900	515	6,222,591	.....	.....	9,205,518
1901	55	6,741,899	.....	.....	9,394,723
1902	2,160	7,543,272	.....	.....	9,611,985
1903	2,070	7,557,400	248	.....	11,203,926
1904	4,190	8,348,561	3,596	286.5	11,513,340
1905	4,725	8,552,422	2,708	172.4	11,760,957
1906	3,545	9,939,782	4,375	305.9	10,852,546
1907	1,925	11,326,254	18,597	268.0	10,348,795
1908	1,820	12,974,588	4,821	140.8	10,597,404
1909	1,435	12,060,580	9,398	147.8	10,728,900
1910	1,415	12,240,775	1,765	77.7	10,717,300
1911	.....	.....	.....	.....	10,672,359

Year.	Manganese Ore.	Mica.	Petroleum *Gallons.*	Rubies.
1897	74,862	652	19,128,828	$200,613
1898	61,419	527	22,234,438	289,750
1899	88,524	497	32,934,007	454,240
1900	129,865	1,025	37,729,211	486,630
1901	122,831	1,505	50,075,117	522,380
1902	160,311	806	56,607,688	434,475
1903	174,563	1,002	87,859,069	444,095
1904	152,601	828	118,491,382	453,060
1905	250,788	1,172	144,798,444	.....
1906	579,231	2,458	140,553,122	465,115
1907	916,770	2,652	152,045,677	491,290
1908	685,135	2,720	176,646,320	239,770
1909	652,958	1,672	233,678,087	293,200
1910	813,721	1,153	214,829,647	294,245

(a) Records of the Geological Survey of India. (b) Not
duction of iron ore in Bengal only. (c) Exports in cwt. of 112 l

ITALY

The following tables itemize the statistics of the production and the foreign commerce of mineral and metallurgical products in Italy:

MINERAL PRODUCTION AND REFINED PRODUCTS OF ITALY. (*a*)
(In metric tons or dollars; 5 lire = $1.)

Year.	Alum.	Aluminum Sulphate.	Alunite.	Antimony.	Antimony Ore.	Asphalt, Mastic and Bitumen.	Asphaltic Rock.	Barytes.
1897.....	1,030	2,310	6,500	404	2,150	18,644	55,339	(*b*)
1898.....	1,165	2,915	7,000	380	1,931	17,813	93,750	12,400
1899.....	945	2,330	5,800	581	3,791	41,732	81,987	12,545
1900.....	1,097	2,403	5,200	1,174	7,609	33,127	101,738	14,003
1901.....	1,075	2,260	4,900	1,721	8,818	31,814	104,111	13,245
1902.....	.....	.....	8,200	1,574	6,116	33,684	64,245	.....
1903.....	.....	.....	8,100	905	6,927	35,757	89,078	.....
1904.....	2,490	2,210	8,000	836	5,712	34,227	111,390	250
1905.....	2,975	2,740	8,500	327	5,083	26,838	106,586	590
1906.....	2,878	2,800	7,500	537	5,704	34,386	130,225	800
1907.....	3,175	3,010	7,600	610	7,892	38,568	161,126	1,720
1908.....	2,875	2,100	6,165	345	2,821	34,761	134,163	1,895
1909.....	2,580	2,750	5,636	59	1,077	39,165	111,067	1,950
1910.....	2,510	2,773	6,081	*Nil*	2,194	41,705	162,212	14,420
1911.....	2,725	2,493	6,100	*Nil*	2,441	50,179	188,133	13,620

Year.	Borax, Refined.	Boric Acid.		Coal. (*c*)	Coal. (Briquettes.)	Coke.	Copper.		
		Crude.	Refined.				Ore.	Ingot, etc.	Sulphate.
1897.....	990	2,704	260	314,222	549,050	430,617	93,377	2,980	5,337
1898.....	702	2,650	166	341,327	594,500	469,228	95,128	3,230	6,364
1899.....	709	2,674	129	388,534	566,000	485,951	94,764	3,032	7,795
1900.....	858	2,491	283	479,896	703,740	487,831	95,644	2,797	13,191
1901.....	544	2,558	347	425,614	754,800	490,803	107,750	3,097	15,374
1902.....	.....	2,763	.....	414,569	.....	.....	101,142	.....	14,601
1903.....	.....	2,583	.....	346,887	724,993	554,559	114,823	.....	18,164
1904.....	569	2,624	314	362,151	903,610	607,297	157,503	2,313	17,237
1905.....	1,007	2,700	749	412,916	842,250	627,984	149,035	1,175	26,212
1906.....	1,062	2,561	562	473,293	829,277	672,689	147,137	.....	34,270
1907.....	880	2,305	466	453,137	787,087	717,704	167,619	17,491	45,263
1908.....	1,024	2,520	429	480,029	822,699	813,842	106,629	18,280	42,598
1909.....	1,110	2,431	578	555,073	924,479	999,381	90,272	20,005	28,551
1910.....	912	2,502	695	562,154	918,055	1,160,543	68,369	22,467	36,236
1911.....	738	2,648	444	555,137	794,206	363,493	68,136	22,908	44,626

Year.	Gold.		Graphite.	Iron and Steel.				
	Ore.	Bullion.		Ore.	Pig.	Bar, sheet, Pipe, Wire, etc.	Tin Plate.	Steel.
1897.....	10,723	$ 209,998	5,650	200,709	8,393	149,944	6,500	63,940
1898.....	9,549	124,869	6,435	190,110	12,387	167,499	7,200	87,467
1899.....	11,859	75,294	9,990	236,549	19,218	197,730	8,000	108,501
1900.....	5,840	38,212	9,720	247,278	23,990	190,518	10,000	115,887
1901.....	890	2,725	10,313	232,299	15,819	180,729	7,550	123,310
1902.....	1,215	.....	9,210	240,705	.....	.....	.....	.....
1903.....	5,734	41,933	7,920	374,790	90,744	177,392	11,275	154,134
1904.....	6,746	43,063	9,765	409,460	112,598	181,385	16,655	177,086
1905.....	1,200	9,169	10,572	366,616	181,248	205,915	18,560	244,793
1906.....	6,543	47,321	10,805	384,217	180,940	236,946	16,350	332,924
1907.....	13,475	33,582	10,989	517,952	148,996	248,157	24,423	346,749
1908.....	14,671	39,930	12,914	539,120	158,100	302,509	28,277	437,674
1909.....	2,890	6,562	11,583	505,095	254,904	281,098	35,880	608,795
1910.....	2,147	10,047	12,510	551,259	399,700	311,210	42,670	670,983
1911.....	2,080	36,550	12,621	373,786	302,931	303,223	25,662	697,958

Year.	Lead.		Manganese Ore.	Manganiferous Iron Ore.	Marble.	Petroleum. Crude.	Refined Petroleum, Benzene, etc.	Pumice Stone.
	Ore.	Pig.						
1897.....	36,200	22,407	1,634	21,262	236,958	1,932	3,392	(a)
1898.....	33,930	24,543	3,002	11,150	271,725	2,015	5,040	2,700
1899.....	31,046	20,543	4,356	29,874	313,744	2,242	5,384	7,500
1900.....	35,103	23,763	6,014	26,800	310,336	1,683	6,077	7,000
1901.....	43,449	25,796	2,181	24,290	334,146	2,246	4,211	8,300
1902.....	42,330	26,494	2,477	23,113	.....	2,633	.....	.....
1903.....	42,443	22,126	1,930	4,735	.....	2,486	4,577	.....
1904.....	42,846	23,475	2,836	2,836	390,118	3,543	6,388	11,000
1905.....	39,030	19,077	5,384	5,384	389,869	6,122	9,924	11,200
1906.....	40,945	21,268	3,060	20,500	430,202	7,452	10,954	18,300
1907.....	(d)43,037	22,978	3,654	18,874	434,612	8,326	10,556	11,300
1908.....	46,649	26,003	2,750	17,812	425,600	7,088	10,876	15,000
1909.....	37,945	22,133	4,700	25,830	391,295	5,895	11,077	10,000
1910.....	36,840	14,495	4,200	25,700	427,274	7,069	12,149	12,000
1911.....	39,008	16,684	3,515	6,482	497,661	10,390	15,569	16,430

Year.	Pyrites. (Cupriferous in part.)	Quicksilver.		Salt.			Silver.	
		Ore.	Metal.	Brine.	Rock.	Sea.	Ore.	Bullion. (j)
1897.....	58,320	20,659	192	11,725	19,801	429,253	405	43,373
1898.....	67,191	19,201	173	11,546	18,199	451,426	435	43,457
1899.....	76,538	29,322	205	11,021	18,721	363,826	540	32,945
1900.....	71,616	33,930	260	10,890	18,331	338,034	584	31,189
1901.....	89,376	38,614	278	10,690	23,054	401,443	511	32,484
1902.....	93,177	44,261	.....	10,581	23,677	424,239	421	.....
1903.....	101,455	55,528	312	10,962	25,911	451,633	405	26,530
1904.....	112,004	60,403	352	11,878	18,638	433,810	143	24,940
1905.....	117,667	63,378	369	12,756	19,669	405,274	170	20,211
1906.....	122,364	80,638	417	13,751	19,007	496,872	48	20,502
1907.....	126,925	76,561	434	19,238	31,540	454,454	62	21,202
1908.....	131,721	82,534	684	15,180	24,033	473,857	53	20,740
1909.....	149,084	97,592	771	15,081	28,026	421,362	44	20,534
1910.....	165,688	87,129	894	16,600	39,197	447,440	32	14,257
1911.....	165,273	97,803	955	17,251	43,763	460,439	24	12,141

Year.	Sulphur.			Talc. Ground.	Zinc.	
	Crude (Fused).	Ground.	Refined.		Ore.	Spelter.
1897.....	496,658	69,178	85,872	(b)	122,214	250
1898.....	502,351	146,001	99,494	12,760	132,099	250
1899.....	563,697	161,509	110,213	11,000	150,629	231
1900.....	544,119	167,466	157,957	14,415	139,679	547
1901.....	563,096	171,252	141,431	11,770	135,784	511
1902.....	510,333	.....	.....	.....	131,965	.....
1903.....	553,751	139,376	139,464	6,300	157,521	126
1904.....	527,563	189,266	163,695	6,740	148,365	189
1905.....	568,927	180,676	180,774	6,626	147,834	1
1906.....	499,814	176,476	170,990	7,894	155,751	90
1907.....	426,972	151,338	160,617	8,850	160,517	168
1908.....	445,312	160,603	156,995	9,410	(e)152,254	(k)
1909.....	435,060	132,531	144,579	9,530	(g)130,890	(k)
1910.....	(i)430,366	171,570	169,093	11,580	(g)146,307	(k)
1911.....	414,671	176,538	166,802	14,156	139,719	(k)

(a) From *Rivista del Servizio Minerario*. (b) Not reported. (c) Includes anthracite, lignite, fossil wood and bituminous schist. (d) Does not include 680 tons lead and zinc ore. (e) Includes 560 tons lead-zinc ore. (g) Includes 290 and 300 tons lead-zinc ore,. 1909 and 1910. (k) Includes 19,460 from foreign ores. (i) Besides 21,297 tons ore sold. Natural.

MINERAL IMPORTS OF ITALY. (a)

(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony.	Arsenic. *Kg.*	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.
1897.....	66	2,604	619	1,632	578	253	16,680	28,937
1898.....	58	700	1,186	1,150	860	147	12,029	18,252
1899.....	64	600	1,675	1,473	936	123	14,391	13,738
1900.....	37	900	1,645	1,933	859	122	15,494	18,436
1901.....	49	1,800	2,019	1,450	825	232	14,872	20,731
1902.....	80	1,200	1,536	1,020	1,170	516	13,732	15,216
1903.....	98	4,400	1,691	1,567	1,099	504	15,547	10,063
1904.....	131	3,700	2,174	2,604	1,875	271	15,260	6,891
1905.....	117	3,400	1,806	3,252	1,444	112	15,797	5,556
1906.....	50	5,300	2,171	2,854	1,400	163	18,937	7,714
1907.....	163	3,100	3,110	3,661	1,540	307	29,024	6,156
1908.....	153	2,800	2,548	3,730	1,523	333	28,935	7,210
1909.....	293	4,800	2,285	3,826	2,094	386	25,250	11,120
1910.....	342	5,300	2,051	3,365	1,777	363	22,183	9,007
1911.....	358	5,500	3,892	4,239	2,024	741	24,467	7,942

Year.	Coal.	Copper Ore.	Copper. Cement.	Copper, Brass and Bronze.	Copper and Iron Sulphates.	Gold, unrefined. *Kg.*	Graphite.
1897.....	4,259,643	1,611	1,049	7,999	28,878	807	315
1898.....	4,431,524	5,471	2,040	7,433	25,560	507	382
1899.....	4,859,556	2,777	1,328	7,334	27,408	326	608
1900.....	4,947,180	5,290	1,298	9,249	32,127	309	982
1901.....	4,838,994	11,047	1,987	8,659	32,058	494	102
1902.....	5,406,069	9,422	2,299	10,865	25,107	479	60
1903.....	5,546,823	9,459	649	9,588	24,566	1,396	63
1904.....	5,904,578	8,104	309	15,198	37,298	1,961	52
1905.....	6,437,539	6,879	486	18,188	30,684	5,768	107
1906.....	7,673,435	9,363	802	21,458	25,060	4,571	361
1907.....	8,300,439	18,023	888	28,937	15,939	4,443	267
1908.....	8,452,320	14,784	344	28,025	25,037	1,909	383
1909.....	9,303,506	11,303	630	22,391	9,137	5,490	141
1910.....	9,338,752	7,550	456	30,339	13,599	6,105	229
1911.....	9,595,882	6,607	278	38,650	37,929	5,743	217

Year.	Iron.		Iron and Steel Scrap.	Lead.		Lead Oxide and Carbonate.	Mineral Paints.
	Ore.	Pig.		Ore. (c)	Metal and Alloys in Pigs.		
1897.....	5,831	156,019	130,938	14,854	1,178	580	888
1898.....	8,723	169,059	138,426	10,947	1,431	647	692
1899.....	20,799	191,613	245,616	7,476	3,990	662	958
1900.....	19,205	160,686	197,415	9,134	3,248	557	958
1901.....	4,054	159,972	148,305	9,063	2,926	815	865
1902.....	4,314	155,143	198,914	1,680	7,563	846	670
1903.....	5,937	126,756	206,036	689	5,398	768	859
1904.....	4,390	149,130	246,359	2,187	4,541	871	940
1905.....	4,745	136,077	276,311	465	6,764	686	974
1906.....	6,452	168,985	344,977	4,526	10,958	984	964
1907.....	22,046	231,042	362,567	4,342	9,231	953	1,119
1908.....	31,090	254,239	326,119	5,620	11,742	1,474	982
1909.....	28,150	246,730	416,354	3,003	10,011	1,132	967
1910.....	17,673	204,854	386,604	1,426	14,674	1,225	1,257
1911.....	50,553	232,811	392,703	6,042	20,187	1,078	1,296

Year.	Nickel Alloys and Manufactures.	Petroleum.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Potassium Sulphate.	Quicksilver.	Silver, Unrefined in Bars. Kg.	Slag.
1897.....	432	68,973	(b)	11,012	562	30	2,434	37,201
1898.....	258	70,654	65,126	11,047	928	39	991	51,199
1899.....	250	71,391	116,283	12,370	1,297	62	1,782	56,549
1900.....	232	73,089	140,281	14,077	1,670	49	2,678	32,254
1901.....	476	69,298	142,108	14,693	1,411	36	4,391	7,312
1902.....	561	68,781	159,341	17,617	1,566	57	8,768	5,634
1903.....	525	68,220	172,328	17,528	1,353	28	12,541	8,849
1904.....	652	69,233	217,162	14,846	1,663	25	15,885	3,821
1905.....	574	66,493	240,144	17,752	1,804	57	20,697	72,785
1906.....	717	64,541	307,762	16,718	1,534	12	20,410	88,118
1907.....	725	72,715	384,896	15,225	3,866	11	21,829	5,378
1908.....	1,079	82,373	531,921	14,962	4,891	10	32,303	1,122
1909.....	540	88,929	478,199	15,547	5,333	2.2	39,208	878
1910.....	1,311	84,748	422,714	16,983	7,753	2.5	37,757	3,153
1119.....	1,584	119,245	479,043	16,325	9,493	2.0	35,439	952

Year.	Sodium Salts.		Sod. and Pot. Nitrates, Refined.	Tin.		Zinc.			
	Carbonate.	Nitrate (Crude).		Block.	Mnfres.	Ore.	Oxide.	Spelter and Old.	Mnfres.
1897....	20,721	16,400	917	1,520	81	(b)	570	3,278	3,556
1898....	20,845	19,961	702	1,722	109	216	573	2,813	3,200
1899....	22,654	22,385	671	1,240	96	(b)	804	3,498	3,221
1900....	23,215	27,706	511	1,643	56	85	1,034	3,627	3,543
1901....	21,956	40,498	315	1,858	91	23	813	3,991	4,079
1902....	26,133	24 483	314	2,114	110	131	904	3,805	4,167
1903....	24,753	43,480	638	2,288	130	46	1,416	4,551	4,461
1904....	27,747	32,283	613	2,170	150	362	1,124	5,202	4,168
1905....	29,066	46,517	689	2,304	103	14	1,246	5,997	4,701
1906....	31,170	32,508	395	3,361	167	2,042	1,920	6,835	4,421
1907....	35,538	41,457	668	2,771	183	11	1,962	8,152	5,407
1908....	38,268	60,784	428	2,602	187	7	2,026	9,339	5,112
1909....	38,252	43,658	532	2,555	191	13	1,571	9,222	4,572
1910....	45,174	61,192	1,083	2,760	238	18	2,001	8,971	5,416
1911....	45,596	59,561	417	2,524	211	428	2,100	10,740	5,892

MINERAL EXPORTS OF ITALY. (a)

(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.	Coal.
1897.....	271	170	15,310	143	1,618	5,330	7,556	23,191
1898.....	338	208	19,465	70	2,167	5,192	6,744	17,749
1899.....	240	245	26,402	45	2,872	5,462	5,386	20,803
1900.....	467	261	24,287	40	2,114	6,860	2,980	23,928
1901.....	765	302	21,856	32	2,190	8,463	3,428	25,594
1902.....	359	144	20,884	91	1,847	7,930	4,215	33,374
1903.....	314	222	24,303	35	901	6,325	3,802	29,219
1904.....	107	163	14,880	70	1,122	7,810	4,089	35,149
1905.....	132	236	23,740	162	2,255	8,445	5,007	38,555
1906.....	208	205	27,176	147	2,777	6,774	4,194	31,666
1907.....	115	142	26,036	152	1,330	4,477	3,118	40,789
1908.....	10	193	24,158	724	1,005	5,439	3,224	46,774
1909.....	8	527	21,978	125	1,704	7,534	2,533	51,343
1910.....	2	485	26,125	150	1,536	7,011	3,525	48,144
1911.....	.....	517	28,455	353	1,346	10,742	5,764	40,800

Year.	Copper. Ore.	Copper and Iron Sulphate.	Gold. Unrefined. *Kg.*	Graphite.	Iron.	
					Ore.	Pig.
.....	2,408	18	1,381	4,164	207,619	498
.....	2,356	25	1,739	5,145	217,556	840
.....	1,148	20	1,162	8,114	234,515	378
.....	1,179	60	2,763	7,820	170,286	329
.....	9	20	2,955	7,169	121,592	311
.....	11	39	733	7,098	209,070	395
.....	15	44	1,291	7,068	98,319	810
.....	43	29	1,494	7,433	2,577	229
.....	77	249	1,731	6,811	11,358	1,395
.....	189	102	1,476	4,904	1,833	254
.....	179	835	802	7,474	26,000	121
.....	188	721	4,739	7,009	35,653	176
.....	233	1,211	8,517	8,125	46	209
.....	969	741	9,056	7,647	9,892	330
.....	130	157	9,127	7,633	22,851	289

r.	Lead.			Mineral Paints.	Phosphate Rock.	Quick-Silver.	Salt.	Silver, Unrefined. *Kg.*
	Ore.	Lead Alloys in Pigs.	Oxide and Carbonate.					
...	4,747	2,790	461	2,318	(*b*)	236	176,520	50,503
...	4,492	5,870	414	2,884	(*b*)	244	126,860	68,607
...	3,129	2,497	389	2,784	(*b*)	223	114,050	32,432
...	3,741	5,018	367	2,977	1,726	259	112,900	25,310
...	3,977	4,463	410	2,913	1,290	301	114,210	42,325
...	3,354	5,650	404	2,953	894	215	145,190	20,427
...	5,041	2,911	426	3,305	2,942	222	144,910	9,486
...	5,524	1,954	347	3,231	2,812	266	130,940	24,165
...	4,311	976	310	3,632	3,519	243	116,040	25,947
...	8,356	2,005	315	4,502	1,652	278	126,199	18,262
...	3,213	1,548	240	4,602	4,560	350	99,191	18,164
...	2,041	1,243	219	3,319	2,271	565	85,489	26,138
...	1,037	782	211	4,289	2,979	714	103,895	34,470
...	4,122	933	197	3,905	3,627	781	122,404	16,721
...	15,791	637	196	4,047	5,427	1,037	118,961	14,247

.	Slag.	Sodium Salts.			Sulphur.	Tin.		Zinc.		
		Carbonate	Nitrate. (Crude.)	Sod. and Pot. Nitrates, Refined.		Block.	Manufactures	Ore.	Oxide.	Spelter and Scrap.
..	8,847	275	151	344	358,932	29	109	133,125	189	309
..	6,861	391	79	256	405,823	34	177	130,064	110	156
..	4,898	438	136	124	424,018	69	176	140,107	123	227
..	4,222	486	58	129	479,139	147	153	111,870	102	359
..	3,261	377	116	59	414,018	202	187	103,020	140	349
..	3,615	446	346	259	439,242	236	174	114,894	122	338
..	4,929	482	781	492	461,289	173	180	116,449	116	591
..	4,458	376	363	230	437,067	171	151	126,393	483	263
..	9,844	214	424	159	381,128	285	107	117,810	173	434
..	8,990	253	80	133	336,339	303	81	144,244	687	639
..	10,934	200	138	102	297,378	434	117	142,271	727	1,182
..	12,122	583	37	57	330,093	180	173	122,456	395	984
..	17,163	517	464	163	329,233	85	126	123,936	212	983
..	10,488	321	98	45	344,513	164	245	127,315	957	845
..	8,274	180	168	15	408,592	166	319	133,471	1,577	616

From *Statistica del Commercio speciale di Importazione e di Esportazione.* (*b*) Not reported. (*c*) es argentiferous lead ore.

JAPAN

The total mineral production of the Japanese Empire, according to the latest available returns, is shown in the following table, in metric tons, unless otherwise specified:

MINERAL PRODUCTION OF JAPAN. (*a*).

Year.	Antimony.		Arsenic. *Kg.*	Coal.	Copper.	Gold. *Kg.*	Graphite	Iron. Pig.	Lead.
	Ore.	Metal.							
1895....	1,061	641	7,343	4,770,313	19,103	935	77	25,863	1,978
1896....	828	517	6,043	5,100,005	20,114	964	215	27,420	1,958
1897....	348	823	13,039	5,147,103	20,425	1,037	391	28,040	1,737
1898....	1,005	235	7,129	6,643,047	21,023	1,159	347	23,611	1,703
1899....	712	229	5,077	6,668,608	19,421	1,675	53	23,066	1,963
1900....	81	349	4,669	7,370,667	24,317	2,124	94	24,841	1,878
1901....	118	429	10,312	8,884,812	27,392	2,475	88	29,449	1,803
1902....	88	528	12,188	9,588,910	29,034	2,975	97	32,130	1,644
1903....	153	434	6,000	10,088,845	33,245	3,140	114	33,870	1,728
1904....	104	321	4,000	10,723,796	33,187	2,765	216	38,143	1,803
1905....	96	190	8,333	11,955,946	35,944	3,048	209	53,210	2,272
1906....	97	627	5,250	13,468,529	36,963	2,873	177	57,373	4,305
1907(*d*) .	(*b*)	248	7,491	13,764,731	40,183	2,938	103	44,447	3,079
1908(*d*) .	(*b*)	198	19,838	14,767,638	41,399	3,598	177	42,007	2,910
1909....	10	157	8,061	14,810,412	45,117	3,932	121	46,627	3,337
1910....	8	120	11,463	15,433,621	48,545	4,368	145	58,043	3,845

Year.	Manganese. Ore.	Petroleum. *Gallons.*	Phosphates.	Pyrite.	Quicksilver. *Kg.*	Salt. *Hectoliters.*	Silver. *Kg.*	Sulphur.	Tin.
1895....	17,142	7,118,962	(*b*)	(*b*)	481	(*b*)	74,815	15,557	48.3
1896....	17,967	(*c*)7,440,206	(*b*)	(*b*)	1,762	(*b*)	64,303	12,540	50.0
1897....	15,448	9,179,474	(*b*)	7,626	2,678	(*b*)	54,289	13,582	47.6
1898....	11,497	11,145,457	(*b*)	8,726	1,399	11,482,422	60,436	10,321	42.7
1899....	11,336	18,844,034	(*b*)	8,376	.....	10,483,082	56,161	10,237	18.5
1900....	15,831	30,470,068	(*b*)	16,166	270	11,890,361	58,799	14,439	12.3
1901....	16,270	39,056,820	(*b*)	17,589	750	12,463,771	54,739	16,548	14.1
1902....	10,844	34,850,129	196	18,580	1,418	11,042,192	57,635	18,287	18.6
1903....	5,616	50,724,174	191	16,149	206	6,574,890	58,704	22,914	19.0
1904....	4,324	51,573,754	13	24,886	*Nil.*	7,019,650	61,339	25,587	25.0
1905....	14,017	47,132,800	1,519	25,569	349	(*b*)	82,886	24,652	26.0
1906....	54,339	60,005,957	3,037	36,038	336	(*b*)	76,247	27,589	77.0
1907(*d*) .	20,586	60,110,558	1,721	56,166	456	(*b*)	95,600	33,329	31.8
1908(*d*) .	11,130	65,165,860	740	33,867	804	(*b*)	123,180	33,149	25.7
1909....	8,708	65,789,858	3,781	27,066	335	(*b*)	127,917	36,317	22.8
1910....	11,120	63,843,601	1,042	78,418	407	(*b*)	141,613	43,155	23.

(*a*) From *Résumé Statistique de l'Empire du Japon*, Tokio. (*b*) Not reported. (*c*) Crude petroleum. (*d*) From reports of the Japanese Department of Agriculture and Commerce.

MEXICO

Owing to the incompleteness of the Mexican statistics of production, we are unable to give any satisfactory table. Exports may, however, be taken as indicating the condition of the mining industry. We owe the statistics for 1908, 1909 and 1910 together with a complete revision of this table, to the courtesy of Don Miguel M. Irigoyen, chief of the Section of Statistics, Secretaria de Hacienda y Credito Publico.

MINERAL EXPORTS OF MEXICO. (a)
(In metric tons or Mexican dollars.)

Year.	Antimony.		Coal.	Copper.		Gold.				
	Metal.	Ore.		Ore.	Ingot.	Ore.	Bullion.	Specie.	Cyanide.	Sulphide.
1896....	.....	3,261	75,541	144	20,659	$ 206,874	$5,533,789	$261,078	$161,784	$44,890
1897....	.....	5,873	105,298	1,094	16,858	365,226	6,220,765	202,223	226,986	33,916
1898....	.....	5,932	118,553	13,146	10,362	1,037,202	6,493,735	367,704	294,730	64,061
1899....	.....	10,382	113,192	223	25,293	335,849	7,017,286	183,474	115,961	266,782
1900....	.....	2,313	38,676	408	27,970	306,392	7,435,864	192,456	128,675	177,193
1901....	.....	5,103	17,281	5,576	33,818	284,722	8,324,681	210,431	178,803	81,744
1902....	1,218	1,280	3,406	6,101	63,609	303,979	9,079,371	129,899	78,295	40,658
1903....	2,304	7,302	1,840	10,912	51,716	264,503	9,693,692	54,636	85,465	124,020
1904 (c).	1,694	81	125	48,365	57,338	537,290	10,867,272	172,532	79,129	176,090
1905....	1,978	57	497	92,540	56,634	1,513,344	29,636,117	106,470	397,814	138,033
1906....	2,418	178	91	73,193	46,767	5,369,173	21,072,014	37,746	337,294	180,348
1907....	4,615	681	1,532	115,245	51,519	3,033,090	19,653,362	5,023,404	417,162	497,893
1908....	4,046	36	719	70,900	26,214	2,746,289	30,101,546	42,389	144,959	334,944
1909....	4,115	31	235	136,031	32,580	2,346,279	40,725,976	23,678	85,110	250,741
1910....	3,730	6	120	129,568	54,061	2,526,444	46,189,613	37,723	46,803	168,162
1911....	4,131	121	101	109,627	60,493	1,945,660	48,113,427	8,153,293	38,031	141,640

Year.	Graphite.	Gypsum.	Lead.		Silver.					
			Ore.	Base Bullion.	Ore.	Bullion.	Specie.	Sulphide.	Cyanide.	Slag.
1896...	795	2,050	167	48,663	$ 9,971,053	$28,565,843	$18,737,331	$1,495,306	$38,049	$64,121
1897...	759	2,095	2	60,029	11,401,176	35,775,125	21,925,347	1,663,581	123,246	39,800
1898...	1,365	1,650	(b)	60,918	11,048,358	37,137,599	16,588,789	1,663,501	257,342	46,488
1899...	2,305	1,050	1	67,441	10,766,099	37,585,911	5,580,834	1,929,085	76,942	4,819
1900...	2,561	1,600	468	74,944	12,495,524	41,468,745	22,679,655	1,893,646	67,607	87,880
1901...	762	800	(b)	79,097	9,615,939	36,348,374	12,038,158	2,141,685	259,282	93,543
1902...	1,434	(b)	118	107,366	4,108,088	45,796,576	17,753,526	1,978,919	108,344	132,093
1903...	1,404	(b)	11	100,532	11,781,048	48,276,797	16,167,673	1,642,627	135,561	289,900
1904 (c)	970	(b)	1	95,010	11,000,869	45,430,020	7,251,132	1,392,356	171,452	202,594
1905...	970	27	1	101,196	8,505,834	63,564,789	20,335,297	736,228	438,094	29,012
1906...	3,915	(b)	(b)	73,699	9,619,763	63,057,152	42,390,357	595,112	434,885	(b)
1907...	3,202	(b)	11	76,158	11,396,844	68,187,169	23,848,571	785,116	483,638	(b)
1908...	1,076	(b)	26	127,010	11,230,372	63,298,659	60,405	791,698	68,849	(b)
1909...	1,704	(b)	1	118,186	8,057,189	66,789,716	91,006	557,520	78,688	(b)
1910...	2,332	1	355	120,662	6,793,896	70,272,894	72,891	208,300	52,053	(b)
1911...	2,974	(b)	397	124,605	6,766,644	78,515,378	8,901	235,410	159,170	*None*

(a) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. (b) Not reported. (c) Figures for 1904 are from *Anuario Estadistico de la Republica Mexicana* for 1904.

NORWAY

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (a)

(In metric tons or dollars; 1 Krone = 27 cents.)

Year.	Apatite. (b)	Chrome Ore.	Copper.		Feldspar.	Gold.	Iron.		
			Ore.	Ingot.			Ore.	Pig and Cast.	Bars and Steel.
1899 ...	1,500	41	43,358	1,209	19,260	$2,700	4,576	406	666
1900 ...	300	165	46,588	1,280	17,609	2,430	17,925	444	614
1901 ...	738	85	40,726	1,073	18,323	2,700	42,252	261	376
1902 ...	2,295	22	40,499	1,347	19,591	36,990	53,675	527	461
1903 ...	1,795	*Nil.*	35,417	1,382	18,590	8,370	53,475	509	442
1904. ..	1,456	154	36,891	1,342	20,835	*Nil.*	45,328	350	395
1905 ...	2,522	*Nil.*	37,045	1,153	22,508	*Nil.*	46,582	474	253
1906. ..	3,482	*Nil.*	32,203	1,333	23,896	5,400	109,259	257	317
1907 ...	1,830	107	39,887	1,517	32,970	*Nil.*	140,804	*Nil.*	283
1908 ...	1,771	*Nil.*	33,688	1,806	34,437	*Nil.*	119,656	.....	.....
1909 ...	1,364	*Nil.*	42,612	1,594	36,439	8,000	40,389	.....	.....
1910 ...	703	.....	46,308	1,814	39,507	5,000	102,447	.....	.....
1911 ...	500	.....	.....	1,650	34,864	.....	.....	.....	.....

Year	Molybdenite.	Nickel.		Pyrites, Iron, and Copper.	Rutile.	Silver.		Zinc Ore. (c)
		Ore.	Metal.			Ore and Native Silver.	Metal. *Kg.*	
1899.....	.....	220	5	95,636	30	429	4,600	279
1900.....	.....	1,888	13	98,945	40	475	4,600	204
1901.....	.....	2,018	40	101,894	55	519	5,680	90
1902.....	20	4,040	60	121,247	*Nil.*	471	6,220	30
1903.....	31	5,670	75	129,939	25	481	7,269	335
1904.....	30	5,352	73	133,603	25	1,297	8,064	42
1905.....	46	5,477	77	162,012	35	1,570	7,100	4,241
1906.....	1,026	6,081	81	197,886	55	1,565	6,370	3,308
1907.....	30	5,781	81	236,038	55	1,756	6,700	400
1908.....	35	5,190	62	269,129	83	2,262	7,470	2,435
1909.....	30	5,770	69.8	282,606	.....	2,729	7,780	983
1910.....	.....	19,639	172	329,642	116	.....	.....	2,219
1911(e) ..	.....	25,200	700	350,000	.....	.....	.....	.....

(a) *Tabeller vedkommendo Norges Bergværkdsdrift, Statistisk Aarbok for Kongeriket Norge.* (b) Exports which represent production. (c) Includes lead ore. (e) Estimate.

MINERAL IMPORTS OF NORWAY. (a)

(In metric tons.)

Year.	Borax. *Kg.*	Cement and Hydraulic Lime.	Coke, Coal and Cinders. *Hectoliters.*	Copper and Brass.		Iron and Steel.	
				Plates and Bars.	Wares.	Pig.	Bars, Hoops, etc. Wrought Iron.
1899.....	62,060	33,652	18,475,996	1,000	1,120	21,445	25,379
1900.....	71,124	24,511	19,002,026	696	1,164	20,844	23,010
1901.....	68,000	20,993	17,665,349	1,018	761	19,112	20,672
1902.....	(c)	18,984	19,338,615	1,118	(C)	18,969	26,685
1903.....	(c)	17,906	20,086,974	899	309	20,652	21,977
1904.....	54,953	12,845	21,049,128	688	866	18,891	24,094
1905.....	(c)	13,797	20,973,608	882	1,146	20,828	27,740
1906.....	63,000	11,676	21,478,000	906	783	20,197	26,015
1907.....	79,810	16,647	24,274,260	(f) 954	1,107	23,345	32,764
1908.....	87,255	44,991	(e)2,073,907	(f)1,013	1,157	26,106	31,849
1909	79,786	57,768	(e)2,095,611	(f) 869	1,095	23,167	31,215
1910.....	105,453	20,100	(e)2,159,717	1,118	1,406	28,034	34,623
1911.....	100,493	17,892	2,187,246	1,229	1,780	26,569	36,172

Year.	Iron and Steel—Continued.						Lead in Pigs and Sheets.
	Anchors, Cables and Chains.	Rails.	Nails, Spikes and Screws.	Steel.	Sheets and Plates.	Other Manufactures.	
1899	1,394	8,137	1,529	2,652	32,192	21,400	869
1900	1,203	11,952	1,219	2,085	29,318	17,493	670
1901	1,708	22,959	1,808	1,905	31,184	18,372	590
1902	2,103	15,316	2,205	1,754	36,288	22,069	(c)
1903	1,807	4,631	1,261	1,958	42,098	18,855	311
1904	2,109	5,814	1,071	1,610	42,013	5,462	498
1905	2,224	6,566	1,222	1,436	42,203	44,414	448
1906.....	2,585	8,086	1,012	2,018	48,969	45,959	727
1907	2,653	6,989	991	1,592	47,758	48,965	887
1908	2,535	12,180	1,032	1,628	49,569	52,594	529
1909	1,750	9,051	1,049	1,911	46,195	55,198	536
1910	2,131	5,512	852	3,203	56,898	60,282	838
1911	2,327	15,312	1,129	3,908	65,847	70,049	939

Year.	Lead White and Zinc Oxide.	Petroleum and Paraffin	Potash.	Salt.	Saltpeter.	Soda.	Sulphur. (b)	Tin in Blocks, etc.	Zinc in Bars, Plates, etc.
1899 ...	1,296	42,182	802	134,583	278	4,555	10,734	546	1,509
1900 ...	1,216	39,657	638	143,365	356	4,576	14,827	149	1,254
1901 ...	1,321	47,011	518	127,607	208	5,220	11,149	141	1,027
1902 ...	(c)	(c)	(c)	141,415	315	(c)	(c)	(c)	1,104
1903 ...	(c)	58,822	457	143,110	245	4,200	8,829	106	1,015
1904 ...	1,898	50,543	477	153,699	321	3,197	12,181	176	940
1905 ...	1,309	43,860	393	137,800	1,048	3,704	10,240	134	967
1906 ...	1,149	41,546	396	167,300	776	4,334	11,465	261	2,791
1907 ...	1,245	44,231	588	163,458	1,004	5,819	11,412	332	3,549
1908 ...	1,201	64,680	504	177,349	935	7,850	12,281	323	1,418
1909 ...	1,304	63,042	475	164,195	572	8,357	10,293	280	1,336
1910 ...	1,277	63,976	536	182,288	546	9,270	10,279	358	5,891
1911 ...	1,465	71,496	524	192,354	940	9,431	8,782	353	9,173

MINERAL EXPORTS OF NORWAY. (a)
(In metric tons.)

Year.	Apatite.	Copper.			Feldspar.	Iodine. *Kg.*	Iron.
		Ore.	Ingot.	Scrap.			Ore.
1899	1,500	7,198	1,785	1,038	19,260	16,180	12,517
1900	300	5,756	1,891	1,168	17,609	11,210	27,158
1901	738	6,041	1,465	774	(*d*)18,423	10,000	39,173
1902	2,295	4,848	1,913	(*c*)	(*d*)19,611	.....	48,775
1903	1,795	3,448	1,930	888	(*d*)18,640	11,417	41,575
1904	1,456	2,673	1,124	785	20,835	9,414	45,434
1905	2,522	3,393	958	968	20,696	12,000	60,558
1906	3,482	84	875	964	19,669	13,248	81,398
1907	1,830	1,581	1,033	1,644	29,399	13,780	132,593
1908	1,771	156	1,260	385	29,896	11,097	110,425
1909	1,365	3,245	1,335	543	33,692	13,620	38,933
1910	703	241	1,377	583	39,507	13,114	88,715
1911	500	924	1,091	575	34,864	16,982	180,932

Year.	Iron—Continued.				Nickel Ore.	Pyrites.	Silver Ore.
	Pig and Scrap.	Bars and Hoops.	Nails and Spikes.	Steel.			
1899	6,085	337	6,089	377	63	83,912	14
1900	8,141	135	5,643	220	272	84,604	90
1901	3,250	370	6,001	179	55	104,151	6
1902	7,359	166	6,431	240	1	105,980	*Nil.*
1903	6,350	10	6,504	200	*Nil.*	118,148	*Nil.*
1904	10,152	13	7,477	167	30	116,550	*Nil.*
1905	9,920	34	8,725	88	220	147,155	*Nil.*
1906	7,362	8	6,786	21	*Nil.*	164,119	*Nil.*
1907	4,652	7	5,879	31	11	187,983	*Nil.*
1908	6,787	2	4,839	17	*Nil.*	218,851	*Nil.*
1909	11,429	24	5,187	11	3	216,767	*Nil.*
1910	13,455	5	6,878	34	.....	343,073	*Nil.*
1911	12,569	10	5,692	271	.....	343,850	457

(a) From *Tabeller vedkommende Norges Bergvaerksdrift und Tabeller vedkommende Norges Handel.* (b) Includes flowers of sulphur. (c) Returns not available. (d) Includes a small quantity of fluorspar. (e) Metric tons. (f) Includes a quantity of sheet aluminum.

PORTUGAL

ıe subjoined table reports the mineral production of Portugal:

MINERAL PRODUCTION OF PORTUGAL. (a)

(In metric tons.)

ıar.	Antimony Ore.	Arsenic Ore.	Coal (Anthracite.) (c)	Copper.			
				Copper-Iron Pyrite. (e)	Other Ores.	Cement.	Matte.
.....	418	524	7,996	206,738	241	3,304	(b)
.....	245	751	10,250	302,686	290	3,145	(b)
.....	59	1,083	11,930	347,234	408	2,521	(b)
.....	38	1,031	24,066	402,870	(b)	2,948	(b)
.....	(b)	527	16,000	443,397	(b)	2,061	(b)
.....	68	736	11,000	413,714	655	2,205	(b)
.....	83	698	8,063	376,177	527	2,448	(b)
.....	31	1,370	12,805	383,581	297	(b)	(b)
.....	84	1,562	11,449	352,479	210	2,148	(b)
.....	481	1,322	6,762	350,746	196	3,634	(b)
.....	383	1,538	8,824	241,771	2,478	2,942	298
.....	76	1,655	4,614	81,417	15,455	3,041	564
.....	6	1,420	6,274	12,337	1,321	3,037	309
.....	25	974	8,149	17,630	2,326	3,768	230

;r.	Gold. *Kg.*	Iron Ore.	Lead Ore. (Galena)	Manganese Ore.	Sulphur Ore.	Tin Ore and Metal.	Tungsten Ore.
....	(d) 17.0	(b)	2,180	1,652	(c)	9	29
....	(d) 6.8	2,519	3,242	907	(c)	102	59
....	(d) 13.0	15,078	3,468	2,949	(c)	30	55
....	2.6	19,803	3,620	1,970	(c)	81	49
....	2.0	21,599	445	904	(c)	31	90
....	2.0	19,914	1,651	(b)	(c)	24	234
....	1.3	15,200	830	30	(c)	(b)	228
....	*Nil.*	12,488	291	(b)	(c)	51	290
....	*Nil.*	3,200	50	(b)	(c)	20	358
....	*Nil.*	(b)	511	22	(c)	22	570
....	10.5	(b)	510	1,374	123,393	35	226
....	57.0	(b)	481	(b)	24,522	28	106
....	25.0	(b)	736	(b)	272,398	18	673
....	4.1	3,360	919	(b)	293,745	9	943

From reports specially furnished *The Mineral Industry* by the Chief of the Department of Mines Ministerio das Obras Publicas except for 1904 to 1906 inclusive, which are from official Govern- reports. The mineral production of the country is identical with exports except in the case of (b) Not reported. (c) Consumed in the country. (d) Metric tons of ore. (e) Previous to he figures for "Sulphur Ore" (largely pyrite) were included under "Copper-Iron Pyrite."

RHODESIA

The statistics of the mineral production of Rhodesia for the last 12 years are given in the accompanying table.

MINERAL AND METALLURGICAL PRODUCTION OF RHODESIA (a).

Year.	Gold. Ozs.	Value.	Silver. Ozs.	Lead. Tons. (b)	Coal. Tons. (b)	Year.	Gold. Ozs.	Value.	Silver. Ozs.	Lead. Tons. (b)	Coal. Tons. (b)
1900....	85,367	$1,498,100	951	.....	.....	1906..	551,894	$9,647,581	110,575	652	103,803
1901....	172,035	2,966,490	3,132	.....	.....	1907..	612,052	10,589,385	147,324	756	115,073
1902....	194,170	3,339,286	3,445	.....	.....	1908..	606,962	12,276,394	283,424	1,069	164,114
1903....	231,872	4,022,756	20,715	.....	.....	1909..	623,388	12,985,430	262,133	965	170,893
1904....	267,737	4,711,016	70,146	455	59,678	1910..	609,956	12,657,791	217,633	749	180,088
1905....	407,048	7,046,692	89,278	570	97,191	1911..	628,521	12,991,500	187,641	639	212,529

(a) From report of Colonel Seely, Under-Secretary of State for the colonies. (b) Long tons.

RUSSIA

The mineral and metallurgical production of Russia, according to official statistics especially reported to THE MINERAL INDUSTRY, is given in the subjoined tables.

MINERAL AND METALLURGICAL PRODUCTION OF RUSSIA. (*a*)

(In metric tons; one metric ton = 61.05 poods.)

Year.	Asbestos.	Chrome Ore.	Coal.	Copper.	Gold. (*b*)	Pig-iron.	Lead.	Manganese Ore.
1896..	1,275	6,682	9,377,560	5,832	$21,667,269	1,620,814	261.5	191,645
1897..	1,016	13,433	11,202,750	6,940	22,194,664	1,880,130	450.1	263,115
1898..	1,665	15,466	12,307,463	7,290	22,195,208	2,241,293	241.2	329,276
1899..	2,693	19,146	13,974,376	7,533	22,396,315	2,708,752	321.8	659,302
1900..	3,845	18,233	16,156,055	8,258	22,369,864	2,933,786	220.7	802,236
1901..	4,398	22,169	16,526,652	8,467	22,763,967	2,866,779	156.0	522,395
1902..	4,508	19,656	16,465,852	8,817	22,258,343	2,598,086	225.3	536,519
1903..	5,264	16,421	17,868,515	9,232	24,147,222	2,487,783	106.3	414,334
1904..	7,502	26,575	19,608,631	9,835	24,627,537	2,972,115	90.3	430,090
1905..	5,896	27,051	18,727,766	8,515	20,521,587	2,628,101	700.2	508,635
1906..	9,197	16,969	21,593,158	9,296	20,020,862	2,694,895	906.8	1,015,686
1907..	9,398	(*g*)26,357	25,741,321	15,930	26,518,253	3,041,570	520.0	(*g*)1,003,528
1908..	10,911	(*g*)10,950	(*g*)25,903,560	17,118	33,143,810	2,818,450	522.5	362,303
1909..	(*h*)13,129	.....	24,532,349	19,656	31,889,956	2,889,353	.....	574,938
1910..	(*h*)10,936	.....	(*f*)23,105,628	(*f*)22,619	(*f*) 35,566,045	(*f*)3,042,046	.....	668,050
1911..	.....	.....	(*f*)16,678,280	(*f*)25,097	(*f*) 34,550,609	(*f*)3,588,449	.....	.....

Year.	Petroleum.	Phosphate Rock.	Platinum. (*Kg.*)	Pyrites.	Quicksilver.	Salt.	Silver. (*Kg.*)	Sulphur.	Zinc.
1896...	6,371,826	3,776	4,930	11,550	491	1,346,118	7,808	437	6,257
1897...	6,945,127	5,917	5,601	19,380	616	1,561,895	4,779	574	5,874
1898...	8,009,828	1,867	6,016	24,570	362	1,505,602	5,143	1,018	5,664
1899...	8,517,608	16,863	5,962	23,250	362	1,679,726	4,419	451	6,326
1900...	9,844,390	25,663	5,089	23,154	141	1,968,007	2,293	1,587	5,963
1901...	10,925,471	21,276	6,371	30,732	363	1,705,924	1,088	2,489	6,104
1902...	10,445,536	13,709	6,135	26,465	416	1,847,021	1,200	1,800	8,264
1903...	9,759,214	14,635	6,009	22,780	362	1,658,938	1,152	281	9,894
1904...	10,058,968	20,282	5,016	31,667	332	1,908,275	726	16	10,612
1905...	7,505,637	20,585	5,250	30,689	318	1,844,678	2,965	16	7,911
1906...	8,167,934	18,970	5,776	20,660	210	1,730,934	430	39	9,602
1907...	9,098,931	(*g*)15,457	5,903	18,316	(*g*)132	1,873,171	7,843	(*d*)57	10,409
1908...	8,732,301	(*g*)14,976	4,883	56,345	147	1,879,717	9,595	(*d*)85	9,960
1909...	8,961,507	.....	5,120	.....	.....	2,259,690	4,109	.....	7,644
1910...	(*e*)4,108,108	.....	5,471	.....	.....	2,047,980	.....	.....	(*f*)8,631
1911...	.....	.....	(*f*)5,432	.....	.....	.....	.....	.....	(*f*)9,652

(*a*) From official sources. (*b*) The value of gold is taken at $20.67 per ounce. (*c*) Not reported. (*d*) Includes sulphide ore. (*e*) Six months. (*f*) Statistics of Association of Russian Manufacturers and Traders. (*g*) St. Petersburg Scientific Committee. (*h*) The Russian Year Book, Dr. Kennard, London.

SOUTH AMERICA

The following tables itemize the statistics of the production and the foreign commerce, or both, of mineral and metallurgical products of South American countries so far as available. No statistics later than those given in the tables have been published.

MINERAL AND METAL PRODUCTION OF BOLIVIA. (*a*)

(In metric tons.)

Year.	Antimony Ore.	Bismuth.		Cobalt Ore.	Copper. (*c*)	Gold. (*b*)	Silver. (*d*)	Tin Ore.	Tungsten Ore.
		Metal.	Ore.						
1905	17	592	.....	.....	6,708	15,044	8,266	26,428	(*e*) 75
1906	.....	231	.....	.....	4,347	17,403	.....	29,374	(*e*) 75
1907	2,279	249	153.379	.....	3,469	3,551	3,696	27,668	(*e*)500
1908	784	259	160.305	.....	3,027	21,617	156.5	29,938	187
1909	.....	(*g*) 89	236.762	.....	3,097	.....	155.4	35,566	168
1910	.....	(*g*) 97	311.060	.....	3,212	.....	143	38,548	.....
1911	.....	.....	1,208.689	.....	1,670	6,499	64.9	(*f*)19,053	.....

(*a*) From a British Consular report. (*b*) Reduced to U. S. currency. (*c*) Includes ingots, precipitate, matte and ore. (*d*) Includes ingots, ore and sulphide. (*e*) U. S. Geol. Surv. (*f*) First half of year.

MINERAL EXPORTS OF BRAZIL. (*a*)

(In metric tons or dollars.) (*d*)

Year.	Agate.	Carbonado.	Copper Ore.	Diamonds.	Gold.	Manganese Ore.	Mica and Talc.	Monazite.	Platinum. (Grams.)	Precious Stones. (*b*)	Rock Crystal.
1903 ..	74	$66,888	316	$62,248	$684,389	161,926	7.0	3,299	1,315	$8,247	.....
1904 ..	54	32,063	610	34,975	611,198	208,260	14.0	4,860	2,122	12,505	35
1905 ..	83	113,157	658	142,459	647,581	224,377	1.0	4,437	72,000	88,463	23
1906 ..	121	319,743	1,484	340,137	771,611	121,331	$6,123	4,351	*Nil.*	141,395	37
1907(*e*)	53	363,200	(*c*)	110,457	1,976,000	236,778	8,176	4,438	*Nil.*	33,335	37
1908(*e*)	79	126,380	(*c*)	97,090	2,187,950	166,122	5,540	4,965	*Nil.*	109,150	36
1909(*e*)	.....	177,572	(*c*)	58,862	2,252,900	240,774	.....	6,462	*Nil.*	56,338	.....
1910(*e*)	34	13,848	(*c*)	14,366	1,954,674	253,953	.....	5,437	*Nil.*	745	.....
1911(*e*)	112	63,400	1	103,567	2,850,370	173,941	9	371	*Nil.*	135,220	24

(*a*) As reported by the *Brazilian Review*. (*b*) Other than carbonado and diamonds. (*c*) Statistics not available. (*d*) The par exchange value of the *Mil Reis* in 1907 was $0.546 U.S. gold. Common exchange value was in 1902, $4.155; in 1903, $4.134; in 1904, $4.146; in 1905, $3.153; in 1906, $3.103, and in 1907, $3.301. (*e*) Reported by U. S. Consul at Rio de Janeiro.

MINERAL PRODUCTION OF CHILE (a).
(In metric tons.)

Year.	Borax.	Coal.	Cobalt Ore.	Copper.	Gold. Kg.	Guano.	Iodine.	Salt.	Silver, Kg.	Sodium Nitrate.	Sulphur.
1896..		(c)	(d)	23,649	1,634	(f)	(e)	2,434	150,480	1,158,088	940
1897..		(c)	(d)	21,128	1,538	(f)	(e)	5,867	140,732	1,148,696	664
1898..	86,892	(c)	(d)	26,331	2,037	(f)	(e)	6,684	131,995	1,283,563	1,256
1899..	(b)	(c)	(d)	25,719	2,060	(f)	274	9,937	129,503	1,389,823	989
1900..		(c)	(d)	25,715	1,975	(f)	302	9,879	73,071	1,460,100	2,472
1901..		(c)	(d)	30,155	1,100	(f)	269	10,099	70.237	1,273,800	2,516
1902..		(c)	(d)	27,066	1,286	(f)	242	9,532	57,418	1,400,408	2,636
1903..	16,879	827,112	290	29,923	994	11,134	387	16,264	28,552	1,444,920	3,560
1904..	16,733	751,628	125	31,025	1,135	2,669	461	17,674	28,501	1,487,598	3,594
1905..	19,612	793,927	28	29,126	1,055	19,380	564	12,108	16,315	1,669,806	3,470
1906..	28,996	932,488	0.19	25,829	1,135	4,709	331	17,116	21,216	1,822,144	4,598
1907..	28,374	832,612	.....	28,863	1,907	7,518	4,202	18,982	28,280	1,846,036	2,905
1908..	35,039	939,836	.....	42,097	1,189	871	330	1,626	52,435	1,970,974	2,705
1909..	32,218	898,971	.....	42,726	1,268	10,692	474	2,046	44,283	2,101,513	4,508

(a) From *Estadistica Minera de Chile.* (b) The combined output of the years 1894 to 1902 inclusive. (c) The combined output of Chile up to the end of 1902 is estimated at 20,650,000 tons. (d) The combined output of Chile up to the end of 1902 is estimated at 5941 tons. (e) Not reported. (f) The combined output of Chile up to the end of 1902 is estimated at 163,704 tons, valued at 5,041.560 pesos ($1,840,169).

MINERAL AND METAL PRODUCTION OF PERU. (a)
(In metric tons.)

Year.	Bismuth.	Borate.	Coal. (b)	Copper.	Gold. Kg.	Lead.	Nickel. Kg.	Petroleum.	Quicksilver. Kg.	Silver. Kg.	Salt.	Sulphur.
1903...	.....	2,466	36,920	9,497	1,078.3	1,302	.....	37,079	.....	170,800	17,637	.....
1904...	.....	2,675	59,920	9,504	601.4	2,209	.....	38,683	.....	145,165	18,545	.....
1905...	12	1,954	75,338	12,213	776.6	1,476	1,778	49,700	1,554	191,476	21,039	.....
1906...	.....	2,598	79,969	13,474	1,247.0	2,568	.....	70,832	2,304	230,300	20,226	1,830
1907...	48	2,451	185,565	20,681	777.6	5,525	.....	100,184	1,500	207,810	21,592	1,880
1908...	9	2,870	311,122	19,854	774.0	2,633	.....	125,948	1,822	195,404	21,899	.....
1909...	30	2,715	321,502	20,068	554.0	2,093	.....	188,125	350	207,656	22,715	.....
1910...	24	2,351	307,320	27,376	708.0	1,866	.....	167,712	350	252,565	17,594	.....

(a) Reported by the Cuerpo de Ingenieros de Minas del Peru, in its *Boletin.* (b) Includes asphaltum and bituminous schist.

SPAIN

The following tables record the mineral and metal production of Spain, as reported by official authorities:

MINERAL PRODUCTION OF SPAIN. (*a*)
(In metric tons.)

Year.	Aluminous Earths.	Antimony Ore.	Arsenic.	Asphaltum.	Asphalt. Rock.	Barytes.	Cement, Hydraulic.	Coal.	
								Anthracite.	Bituminous.
1897...	409	354	244	1,878	1,656	429	159,439	8,758	2,010,960
1898...	505	130	111	2,354	2,383	364	164,862	20,105	2,414,127
1899...	685	50	101	2,646	2,542	887	165,645	34,842	2,565,437
1900...	420	30	150	2,331	4,193	833	185,811	68,427	2,514,545
1901...	305	10	120	4,182	3,956	1,067	189,909	85,266	2,566,591
1902...	337	67	*Nil.*	6,034	6,301	642	201,856	109,298	2,614,010
1903...	381	42	1,088	4,675	6,277	507	245,294	108,959	2,587,652
1904...	925	245	400	3,463	3,761	453	286,737	163,275	2,903,771
1905...	221	77	1,140	5,805	5,725	290	296,605	159,517	2,912,466
1906...	386	180	1,114	6,229	7,794	330	299,294	113,747	3,095,043
1907...	1,209	205	1,500	8,643	8,219	314	329,926	164,498	3,531,337
1908...	60	124	2,004	9,231	12,373	334	343,001	188,463	3,696,653
1909(*f*)	35	(*b*)	506	6,582	5,284	422	472,909	198,302	3,662,573
1910...	400	15	444	8,473	7,795	476	484,161	211,958	3,600,056

Year.	Coal (*Continued*).		Coke.	Copper Ore.		Copper.			Fluorspar.
	Lignitic.	Briquets.		Argentiferous.	Pyritic.	Fine.	Matte.	Precipitate.	
1897...	54,232	332,272	755,394	(*c*) 18,488	2,161,182	7	16,120	29,652	2
1898...	66,422	369,418	768,151	203	2,299,444	593	16,024	29,703	5
1899...	70,901	348,838	341,443	1,103	2,443,044	4	15,755	41,927	310
1900...	91,133	341,156	381,000	2,006	2,714,714	5	18,159	29,652	4
1901...	95,867	338,684	455,586	(*b*)	2,672,365	79	15,634	28,433	(*b*)
1902...	84,242	331,957	404,503	878	2,617,776	(*b*)	.....	36,045	93
1903...	104,232	339,120	433,780	3,056	2,796,733	(*b*)	.....	27,448	4,000
1904...	100,773	307,630	432,726	(*b*)	2,624,512	(*b*)	8,117	29,494	(*b*)
1905...	168,994	290,830	448,073	(*b*)	2,621,054	(*b*)	8,243	17,988	(*b*)
1906...	189,048	311,328	435,808	(*b*)	2,888,778	(*b*)	9,068	19,200	70
1907...	191,001	355,718	476,360	(*b*)	3,182,645	(*b*)	9,886	20,887	270
1908...	223,160	296,216	477,059	(*b*)	2,985,779	(*b*)	205	19,599	253
1909(*f*)	265,019	(*b*)	500,909	(*b*)	2,955,254	(*b*)	2,077	16,641	246
1910...	245,518	474,891	521,078	(*b*)	3,231,418	85	2,684	14,056	180

Year.	Iron Ore.		Iron and Steel.			Kaolin (China Clay).	Lead (Argentiferous).	
	Argentiferous.	Non-Argentiferous.	Pig.	Wrought Iron.	Iron and Steel Worked.		Ore.	Metal.
1897...	5,559	7,419,768	146,940	80,894	66,007	6,294	186,692	91,258
1898...	24,490	7,197,047	113,492	65,900	50,362	5,445	244,068	88,981
1899...	17,139	9,397,738	113,071	40,332	112,982	2,790	184,906	70,874
1900...	26,348	8,675,749	91,126	54,307	144,355	3,794	182,016	74,341
1901...	27,726	7,906,517	135,600	47,085	121,023	2,220	207,188	73,895
1902...	24,361	7,904,555	330,747	.....	163,564	3,412	227,645	74,370
1903...	90,996	8,304,153	380,284	.....	199,642	2,578	179,858	56,687
1904...	122,109	7,964,748	283,819	50,858	186,705	1,700	177,104	57,956
1905...	152,027	9,007,245	305,462	11,366	223,545	720	160,381	56,361
1906...	126,445	9,448,533	315,309	6,035	274,280	610	158,425	53,856
1907...	(*b*)	9,896,178	355,240	14,767	310,125	640	165,289	51,430
1908...	(*b*)	9,271,592	403,554	21,807	262,843	1,370	165,382	53,741
1909(*f*)	3,813	8,786,021	428,622	24,187	218,410	570	161,496	43,552
1910...	46,161	8,666,795	373,322	59,133	201,798	1,496	150,592	38,548

Year.	Lead (Non-Argentiferous).		Manganese Ore.	Mineral Paints (Ocher).	Phosphate Rock.	Pyrites (Iron).	Pyrites (Arsenical).	Quicksilver.	
	Ore.	Metal.						Ore.	Metal.
1897 ...	110,496	75,112	100,566	200	2,084	100,000	(b)	32,378	1,728
1898 ...	150,472	78,370	102,228	200	4,500	70,265	230	31,361	1,691
1899 ...	123,753	91,730	104,974	100	3,510	107,386	(b)	32,144	1,361
1900 ...	131,437	98,189	112,897	58	4,170	34,638	515	30,216	1,095
1901 ...	174,376	95,399	60,325	164	4,220	33,953	1,328	23,367	754
1902 ...	100,403	103,190	46,069	(b)	1,150	145,173	5,648	26,037	1,425
1903 ...	108,660	118,422	26,194	(b)	1,124	155,739	7,996	30,370	968
1904 ...	93,230	127,906	18,732	(b)	3,305	161,841	3,510	27,185	1,130
1905 ...	105,113	129,332	26,020	(b)	1,370	179,079	4,790	26,485	853
1906 ...	105,095	131,614	62,822	164	1,300	189,243	2,434	28,965	1,568
1907 ...	103,632	135,066	41,504	114	3,547	225,830	3,423	28,789	1,212
1908 ...	126,667	134,321	16,745	400	4,483	263,451	5,533	42,210	1,068
1909(f).	137,050	136,441	7,827	418	1,387	258,931	3,235	37,398	1,393
1910 ...	216,738	151,975	8,607	759	2,840	294,184	1,461	22,714	1,114

Year.	Salt.	Silver.		Soapstone.	Sulphur.		Tin Ore. Dressed.	Topaz *Kg.*	Tungsten Ore.	Zinc.		
		Ore.	Metal. *Kg.*		Crude Rock.	Refined.				Ore.	Spelter.	Sheets.
1897..	508,606	982	71,168	3,601	18,805	3,500	(e)2,378	44	10	73,848	3,907	2,337
1898..	479,358	967	76,295	2,613	34,943	3,100	4	90	37	99,836	4,300	1,731
1899..	598,108	764	88,409	4,814	58,922	1,100	57	44	151	119,710	4,100	2,084
1900..	450,041	742	140,457	8,109	64,364	750	47	95	1,958	86,158	2,855	2,756
1901..	345,063	391	94,977	4,880	49,856	610	115	310	6	119,708	2,573	2,781
1902..	426,434	175	96,975	542	15,442	450	12,762	*Nil.*	11	127,618	5,569	(b)
1903..	427,394	231	112,978	3,725	38,573	1,608	330	90	*Nil.*	154,126	5,134	(b)
1904..	543,658	303	117,418	5,165	40,389	605	229	....	60	156,329	5,887	2,913
1905..	493,451	540	123,607	4,364	38,153	610	209	....	375	160,561	6,184	2,936
1906..	541,978	470	126,424	3,609	28,965	700	86	171	430	170,384	6,209	2,639
1907..	605,895	702	127,435	13,875	27,054	3,612	315	266	386	191,853	6,144	2,485
1908..	822,677	441	129,881	4,730	13,872	2,988	838	(b)	226	156,233	6,357	2,693
1909(f	800,703	388	143,403	5,583	21,750	3,429	(e)1,555	(b)	129	163,522	9,625	(b)
1910..	678,057	217	129,157	4,665	30,113	3,834	35	149	153	156,113	8,557	2,722

(a) Figures are from *Estadistica Minera de España*, except for 1896 and 1898, which were from the official *Reports of the Junta Superior Facultativa de Minas* Madrid. (b) Not reported. (c) Represents non-argentiferous copper ore. (e) Undressed tin ore. (f) *Revista Minera.* June 24, 1911.

SWEDEN

The official statistics of mineral production, imports and exports, a summarized in the following tables:

MINERAL PRODUCTION OF SWEDEN. (a)
(In metric tons.)

Year.	Alum.	Coal.	Copper.			Feldspar.	Graphite.	Gold. Kg
			Ore.	Ingot.	Sulphate.			
1898	153	236,277	23,335	235	1,165	20,737	50	125.9
1899	164	239,344	22,334	179	1,287	16,017	35	106.2
1900	167	252,320	22,725	136	1,265	15,228	85	88.5
1901	121	271,509	23,660	137	1,224	13,502	56	62.7
1902	132	304,733	30,095	178	1,257	17,960	63	94.3
1903	140	320,390	36,687	776	1,171	19,392	25	50.6
1904	125	320,984	36,834	533	1,248	18,021	55	60.9
1905	139	322,384	39,255	1,385	1,029	19,224	40	55.0
1906	167	296,980	19,655	1,209	562	21,014	37	20.3
1907	131	305,338	21,957	1,577	782	20,244	33	28.1
1908	138	305,206	21,371	2,808	731	17,494	66	20.3
1909	132	246,808	9,562	2,375	628	15,772	26	14.1
1910	182	302,786	3,638	3,111	20	21,591	49	2.

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate.	Bessemer.	Basic.	Crucible
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434,606	497,727	195,331	328,999	105	91,898	179,357	1,223
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(b)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,878	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	252,832	1,162
1905	4,364,833	539,437	182,640	356,898	144	78,204	288,675	1,319
1906	4,501,656	604,789	178,298	381,118	170	84,633	311,435	1,457
1907	4,478,917	615,778	174,405	403,994	159	77,036	341,893	1,282
1908	4,712,494	567,821	152,256	363,408	277	81,054	355,394	1,169
1909	3,885,046	444,764	120,669	292,478	182	63,351	248,757	927
1910	5,549,917	603,929	151,713	204,955	236	97,503	372,451	2,215

Year.	Lead.	Manganese Ore.	Pyrites.	Silver-lead Ore.	Silver. Kg.	Sulphur.	Zinc Ore.
1898	1,559	2,358	386	6,743	2,033	50	61,627
1899	1,606	2,622	150	5,730	2,290	(b)	65,189
1900	1,424	2,651	179	5,300	1,927	70	61,044
1901	988	2,271	*Nil.*	11,366	1,557	(b)	48,930
1902	843	2,850	*Nil.*	9,378	1,365	74	48,783
1903	678	2,244	7,793	9,792	1,005	(b)	62,927
1904	589	2,297	15,957	8,187	651	35	57,634
1905	576	1,992	20,762	8,397	606	0	56,885
1906	753	2,680	21,827	1,938	938	0	52,552
1907	813	4,334	27,113	1,987	929	0	50,894
1908	277	4,616	29,569	2,058	630	0	40,077
1909	166	5,212	16,104	1,721	512	0	43,760
1910	355	5,762	25,445	2,700	30	0	49,453

(a) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen.* (b) Not reported.

MINERAL IMPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1 krone = 27 cents).

Year.	Asbestos. (*c*)	Asphalt.	Barytes.	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement.	Chalk, White, Unground. *Hectoliters.*	Coal.
1898....	112	5,409	299	196	75	5,401	1,656	7,016	2,392,451
1899....	567	6,286	292	190	65	4,914	1,363	16,079	3,047,618
1900....	763	5,676	411	194	66	6,084	1,941	12,059	3,033,885
1901....	178	4,524	295	253	68	6,602	2,868	13,569	2,793,309
1902....	213	5,779	.....	242	71	7,278	9,822	11,583	2,911,286
1903....	217	5,957	.....	240	71	7,419	11,145	41,868	3,192,990
1904....	356	6,243	.....	299	77	10,128	10,526	10,115	3,367,826
1905....	140	4,760	264	294	82	18,788	10,999	13,305	3,297,485
1906....	287	7,134	559	321	79	9,908	13,136	10,777	3,718,884
1907....	672	8,213	610	490	85	6,784	17,801	(*g*) 860	4,146,785
1908....	505	6,368	514	347	71	11,499	6,158	(*g*) 419	4,427,507
1909....	383	7,922	529	365	76	10,280	12,944	(*g*) 512	4,084,055
1910....	671	7,406	625	379	92	10,913	15,438	(*g*) 473	4,180,250
1911....	1,011	6,803	607	381	76	16,819	24,830	(*g*) 605	3,964,870

Year.	Copper, also Alloys of Copper.	Emery.	Graphite.	Gypsum	Iron (crude).	Lead.	Litharge.	Phosphorus. *Kg.*	Platinum. *Kg.*
1898....	5,227	131	167	7,979	76,832	2,139	160	66,466	49
1899....	4,740	125	162	6,457	68,909	2,125	177	59,989	99
1900....	4,745	136	213	6,794	82,957	2,067	148	67,557	59
1901....	5,153	169	180	6,589	66,131	1,991	165	70,672	172
1902....	6,890	147	(*b*)	6,754	43,828	2,509	172	68,441	130
1903....	6,109	132	(*b*)	8,795	49,411	2,644	237	112,659	116
1904....	7,367	221	(*b*)	8,868	90,102	2,849	213	47,421	84
1905....	6,481	271	(*b*)	11,270	87,843	2,823	205	69,526	105
1906....	8,899	284	(*b*)	13,496	108,193	3,457	255	79,048	133
1907....	(*c*) 14,213	336	375	15,037	115,186	3,384	210	77,936	109
1908....	(*c*) 12,392	308	540	11,644	109,841	3,964	248	107,301	117
1909....	(*c*) 11,000	428	443	14,212	99,519	3,222	217	88,241	72
1910....	(*c*) 13,297	438	398	15,816	117,180	3,453	267	99,145	77
1911....	14,014	459	398	14,505	114,859	2,901	270	77,837	103

Year.	Potassium.				Quicksilver. *Kg.*	Salt.		Silver.	
	Chloride.	Cyanide *Kg.*	Hydrate.	Carbonate.		Crude.	Refined.	Bullion and Mfres. *Kg.*	Specie.
1898.....	259	2,604	1,451	1,112	2,631	85,246	2,188	21,696	$191,766
1899.....	225	2,313	1,266	1,231	4,210	98,417	3,166	11,565	156,707
1900.....	364	2,221	1,915	1,257	3,629	70,302	3,098	11,559	62,315
1901.....	260	2,658	1,435	1,266	5,958	79,038	3,072	7,476	78,416
1902.....	222	2,950	1,720	1,238	4,866	82,439	3,037	4,853	74,826
1903.....	245	3,294	2,034	1,150	5,043	88,139	3,419	11,259	90,366
1904.....	214	3,237	2,234	1,184	5,768	84,237	4,615	19,034	86,891
1905.....	1,296	3,437	2,251	1,133	4,609	87,677	3,889	11,067	82,620
1906.....	1,986	4,106	2,486	1,082	5,535	88,341	3,700	15,253	93,990
1907.....	1,840	4,150	2,484	1,269	8,930	(*h*) 835,190	18,821	26,334	52,074
1908.....	2,190	3,563	2,835	1,209	7,299	(*h*) 903,633	24,394	20,149	93,669
1909.....	1,809	3,808	2,627	1,312	6,077	(*h*) 766,195	24,143	17,315	154,265
1910.....	2,335	4,272	2,767	1,346	7,079	(*h*) 766,512	22,425	21,162	75,604
1911.....	1,288	4,993	3,737	1,268	4,164	(*h*) 736,935	26,026	15,175	134,407

Year	Sodium.				Sulphur.	Sulphuric Acid.	Tin.		
	Carbonate.	Hydrate.	Nitrate. (*d*)	Sulphate (*e*).			Salts-*Kg.*	Block.	Zinc.
1899....	13,323	929	15,006	15,140	13,505	2,558	5,404	486	2,829
1900....	12,680	1,038	14,245	15,590	20,152	2,472	3,243	630	2,912
1901....	13,669	800	17,614	15,494	20,715	1,950	2,334	541	2,900
1902....	.....	1,623	15,553	18,924	23,002	1,887	1,652	644	3,255
1903....	.....	1,426	20,616	16,120	24,577	2,620	1,467	655	3,312
1904....	11,898	2,112	19,776	17,596	18,248	2,001	1,460	719	3,705
1905....	13,592	1,489	23,183	17,115	18,631	3,424	1,722	597	3,780
1906....	14,974	1,478	27,174	19,948	22,745	2,535	1,102	819	4,484
1907....	14,970	1,628	26,181	21,486	25,456	2,628	6,117	891	5,407
1908....	14,149	1,256	27,631	18,717	30,806	3,073	2,817	808	4,626
1909....	13,490	1,119	28,849	20,226	26,836	1,955	1,357	794	5,294
1910....	15,594	1,260	32,315	22,429	38,530	2,707	1,161	764	5,843
1911....	15,777	2,501	31,324	27,482	34,622	861	915	866	5,951

MINERAL EXPORTS OF SWEDEN. (*a*)
(In metric tons or dollars; 1krone = 27 cents.)

Year.	Alum.	Ammonium Sulphate	Antimony, Crude-*Kg.*	Asbestos. *Kg.*	Cement.	Coal. (Anthracite.)	Copper.		Graphite.
							Ore.	Copper and Alloys.	
1897 ...	54	180	800	1,348	27,112	74	(*b*)	933	7
1898 ...	32	36	4,700	1,055	28,676	496	1,102	1,346	9
1899 ...	26	2	2,600	2,812	31,101	762	315	1,230	17
1900 ...	24	2	4,600	2,436	42,564	1,108	448	2,012	18
1901 ...	56	156	1,800	2,179	17,794	716	602	1,243	19
1902 ...	20	174	4,090	1,864	19,499	866	845	1,516	5
1903 ...	22	*Nil.*	3,473	15,357	21,319	509	1,555	1,858	9
1904 ...	9	219	3,810	16,339	27,509	605	749	1,396	(*b*)
1905 ...	12	445	3,147	2,386	38,504	425	2,137	2,654	(*b*)
1906 ...	11	30	4,584	1,510	45,960	1,352	1,841	2,662	(*b*)
1907 ...	7	*Nil.*	4,485	2,167	18,053	2,925	882	2,762	8
1908 ...	9	202	4,188	1,612	34,164	1,293	1,114	3,299	18
1909 ...	5	331	6,536	1,093	33,197	771	723	3,264	7
1910 ...	11	704	4,555	1,242	73,351	776	878	2,580	7
1911 ...	2	253	7,320	760	108,653	1,045	1,114	2,500	12

Year.	Gypsum and Mfres.	Iron and Steel.		Lead and Mfres.	Peat.	Phosphorus. *Kg.*	Potassium Chloride.
		Ore.	Unwrought.				
1897.....	9	1,400,801	279,525	1,473	1,816	1,627	463
1898.....	27	1,439,860	301,192	570	1,616	4,085	506
1899.....	8	1,628,011	320,742	818	1,979	1,890	335
1900.....	10	1,619,902	304,175	1,209	3,843	879	931
1901.....	55	1,761,257	268,143	1,028	3,064	1,254	708
1902.....	117	1,729,000	(*f*) 73,403	546	3,620	1,290	1,114
1903.....	119	2,828,000	(*f*) 70,788	333	3,217	300	790
1904.....	162	3,065,522	(*f*) 88,124	275	4,212	1,994	1,266
1905.....	156	3,316,626	(*f*)120,987	512	5,157	34,388	1,499
1906.....	6	3,661,218	(*f*)112,719	531	6,531	700	(*b*)
1907.....	16	3,521,717	201,643	519	6,524	(*b*)	(*b*)
1908.....	37	3,654,268	159,095	496	5,559	400	1,363
1909.....	21	3,204,522	161,757	319	9,999	1,305	1,400
1910.....	41	4,434,805	213,861	557	9,754	265	1,383
1911.....	13	5,086,898	226,044	1,181	10,371	595	1,564

Year.	Salt. Refined *Kg.*	Silver. Bullion. *Kg.*	Soda. (Carbonate.)	Sulphur.	Tin.		Zinc.	
					Block and Scrap.	Mfres-*Kg.*	Ore.	Crude and Mfrs.
1897....	1,424	329	686	11	25.6	7,113	44,425	135
1898....	216	130	509	11	20.8	1,263	49,597	185
1899....	110	367	227	68	8.8	1,033	45,635	157
1900....	407	296	238	20	21.5	1,521	40,879	157
1901....	1,556	179	237	12	20.4	8,110	41,248	101
1902....	1,945	110	621	147	25.5	1,603	43,813	63
1903....	*Nil.*	484	10	217	43.3	3,893	45,389	351
1904....	1,883	115	45	4	45.6	3,479	44,259	332
1905....	*Nil.*	10	403	4	33.9	654	51,765	295
1906....	8,652	77	463	12	51.0	353	45,380	410
1907....	4,452	160	39	1	67.9	2,518	41,236	529
1908....	14,300	136	114	7	53.9	274	38,543	908
1909....	6,422	437	27	5	42.5	276	38,865	1,200
1910....	1,326	814	31	6	51.1	356	40,405	1,278
1911....	3,450	1,090	40	7	66.0	152	41,854	2,158

(*a*) From *Bidrag till Sveriges Officiella Statistik* and *Sveriges Utförsel och Införsel.* (*b*) Not reported. (*c*) Includes crude and manufactures. (*d*) Includes a small quantity of potassium nitrate. (*e*) Includes sodium bisulphate. (*f*) Includes only crude or ballast iron. (*g*) Metric tons. (*h*) Hectoliters.

UNITED KINGDOM

The statistics of the mineral production, imports and exports, according to official reports, are given in the subjoined tables.

MINERAL AND METALLURGICAL PRODUCTION OF THE UNITED KINGDOM. (a)
(In metric tons.)

Year.	Alum Shale.	Arsenious Acid.	Arsenical Pyrites.	Barium Minerals.	Bauxite.	Chalk.	Clay. (e)	Coal.
1899	5,913	3,890	13,735	25,059	8,137	4,752,982	15,305,895	223,616,279
1900	1,329	4,146	9,727	29,937	5,871	4,444,765	14,279,181	228,772,886
1901	4,019	3,416	2,620	26,844	10,357	4,399,043	14,393,196	222,614,981
1902	5,755	2,165	842	23,986	9,192	4,466,004	15,549,002	230,728,562
1903	3,337	916	58	24,659	6,226	4,541,494	16,460,526	234,019,821
1904	6,636	992	44	26,748	8,839	4,509,768	16,210,734	236,130,373
1905	7,245	1,552	651	29,528	7,417	4,608,153	15,376,910	239,906,999
1906	9,605	1,625	650	36,319	6,760	4,825,299	12,459,213	255,067,622
1907	10,063	1,523	1,800	42,648	7,658	4,855,857	15,065,141	272,097,858
1908	5,459	2,007	3,270	39,572	11,904	4,329,983	14,638,710	265,726,332
1909	9,266	2,926	182	42,436	9,652	4,508,136	14,293,598	268,007,890
1910	6,781	2,187	958	45,384	4,208	4,705,766	(h)14,316,469	268,677,182
1911	.....	2,178	1,189	43,232	6,103	.....	(i)541,242	276,242,579

Year.	Copper.		Fluorspar.	Gold.		Gravel and Sand.	Gypsum.	Bog Ore (Ireland) (c)
	Ore and Precipitate.	Fine.		Ore.	Bullion. *Kg.*			
1899	8,452	647	796	3,096	103.5	1,800,208	215,974	4,390
1900	9,643	777	1,472	21,135	437.6	1,867,211	211,436	4,221
1901	6,903	541	4,232	16,641	194.5	1,990,926	204,045	2,649
1902	6,210	490	6,388	30,432	130.0	2,100,829	228,264	4,983
1903	6,977	545	12,102	29,057	171.0	2,281,689	223,426	4,156
1904	5,552	501	18,450	23,574	610.7	2,275,426	237,749	4,616
1905	7,267	727	40,079	16,237	169.0	2,277,486	259,596	3,256
1906	7,882	(b)	42,521	17,662	(b)	2,404,857	228,627	5,512
1907	6,867	677	50,257	13,186	59.4	2,438,798	239,285	6,391
1908	5,528	588	35,257	7,237	28.5	2,228,245	231,980	4,364
1909	3,777	442	43,165	5,627	37.6	2,199,583	242,832	2,719
1910	4,050	456	62,607	6,252	75.5	2,235,160	259,661	(b)
1911	3,296	.....	32,094	2,796	(b)	.....	236,241	(b)

Year.	Iron.		Lead.		Manganese Ore.	Mineral Paints.	Oil Shale.	Phosphate of Lime.
	Ore.	Pig.	Ore.	Pig.				
1899	14,692,711	4,992,468	31,494	23,929	422	16,575	2,246,197	1,469
1900	14,257,344	4,743,172	32,487	24,762	1,384	15,448	2,318,736	630
1901	12,475,700	4,158,745	33,084	20,361	1,673	14,780	2,392,812	71
1902	13,641,459	4,470,420	25,000	17,988	1,299	17,235	2,141,355	87
1903	13,935,748	4,573,202	26,993	20,278	831	14,377	2,041,851	71
1904	13,994,670	4,596,803	26,796	20,155	8,896	16,307	2,370,391	59
1905	14,824,183	(f)9,746,221	28,091	20,977	14,582	16,468	2,536,784	Nil.
1906	15,748,412	(f)9,999,211	30,710	22,693	23,126	14,437	2,586,851	Nil.
1907	15,983,310	(f)9,850,953	33,053	24,853	16,356	14,927	2,732,968	33
1908	15,272,273	4,925,250	29,718	21,336	6,409	15,643	2,938,456	9
1909	15,220,408	(f)9,818,916	30,221	22,823	2,812	16,575	3,014,678	4
1910	15,470,392	(f)10,380,213	28,992	21,867	5,554	16,780	3,180,520	(b)
1911	(i)9,863,987	(f)9,874,621	24,247	.....	5,067	(i)5,681	.....	(b)

ar.	Pyrites.	Salt.	Silica (chert and flint).	Silver. *Kg.*	Stone.			
					Igneous Rock.	Limestone. (*d*)	Sandstone.	Slate.
.....	12,426	1,945,531	69,955	5,969	4,785,284	12,499,736	5,296,026	650,077
.....	12,484	1,873,601	78,971	5,964	4,709,997	12,099,940	5,101,868	595,428
.....	10,405	1,812,180	132,700	5,452	5,131,787	11,363,202	5,199,234	496,756
.....	9,315	1,893,881	100,938	4,560	5,554,696	12,368,196	5,571,121	525,665
.....	9,794	1,917,274	74,355	5,440	5,512,605	12,419,120	5,496,312	540,143
.....	10,452	1,921,899	66,300	4,967	6,084,642	12,235,825	5,391,265	572,181
.....	12,381	1,920,149	71,808	5,212	6,052,210	12,701,808	5,729,799	523,892
.....	11,318	1,996,593	69,300	(*b*)	6,264,402	12,962,725	5,345,328	500,546
.....	10,358	2,016,409	54,523	4,780	5,765,262	12,709,288	5,092,246	450,651
.....	9,600	1,873,555	64,813	4,207	6,211,860	11,977,007	5,105,481	420,979
.....	8,964	1,851,999	53,063	4,421	6,384,144	12,000,790	4,673,839	408,639
.....	10,393	2,083,550	75,089	4,251	6,714,774	21,713,572	4,456,679	423,005
.....	9,666	(*i*)213,460	(*i*)5,150	(*b*)	(*i*)	(*i*)	(*i*)	(*i*)

Year.	Strontium Sulphate.	Tin.		Tungsten Ore.	Uranium Ore.	Zinc.	
		Ore Dressed.	Block.			Ore.	Spelter.
.....	12,831	6,494	4,077	96	7	23,505	8,837
.....	9,270	6,911	4,337	9	42	25,070	9,214
.....	16,923	7,407	4,634	21	80	23,967	8,555
.....	32,799	7,681	4,462	9	53	25,462	9,275
.....	23,209	7,500	4,351	276	6	25,287	9,430
.....	18,460	6,849	4,198	164	*Nil.*	28,097	10,427
.....	14,523	7,316	4,540	174	105	24,025	9,023
.....	14,338	6,376	(*b*)	267	11	23,189	(*b*)
.....	10,917	7,192	4,478	327	72	20,402	7,222
.....	16,733	8,137	5,133	237	72	15,469	5,926
.....	14,267	8,422	5,282	382	6	10,061	5,282
.....	4,838	7,693	4,874	275	76	11,418	3,879
.....	.....	8,331	(*b*)	264	68	17,943	(*b*)

:) From *Mineral Statistics of the United Kingdom*. (*b*) Not reported. (*c*) Bog ore, which is mined and, is an ore of iron, used principally for purifying gas. (*d*) Does not include chalk. (*e*) Includes clay, potters' clay, and fuller's earth. (*f*) Includes production from imported ore. (*g*) Estimated. :ludes shale. (*i*) Partial figures available only.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*)

(In metric tons or dollars; £1 = $5.)

Alkali.	Asphaltum.	Borax.	Coal, Coke and Pat. Fuel.	Copper.			Iron and Steel.		
				Ore.	Regulus and Precipitate.	Wrought, Unwrought and Old.	Iron Ore.	Pig Iron.	Scrap.
16,360	53,061	15,667	10,112	102,365	89,123	72,223	6,398,639	178,199	31,687
(*c*)13,429	74,694	15,710	7,685	102,503	93,338	68,809	5,637,670	198,536	44,721
(*c*)26,292	65,896	13,390	3,331	90,007	74,684	92,349	6,542,793	226,708	39,584
(*c*)14,321	(*b*)	11,959	3,535	85,644	77,884	64,591	6,417,188	132,364	17,051
(*c*)14,325	(*b*)	16,012	2,812	80,771	67,739	90,717	6,198,368	132,494	19,326
(*c*)16,593	(*b*)	11,552	49,277	94,198	70,235	71,294	7,172,171	128,183	23,569
(*c*)14,070	(*b*)	16,955	49,269	97,789	76,073	75,487	7,634,839	90,674	36,559
(*c*)15,579	(*b*)	17,551	19,136	105,409	73,101	89,312	7,764,589	104,950	27,404
(*c*)13,732	(*b*)	(*n*)19,424	3,904	111,897	71,120	124,226	6,154,733	69,930	24,653
(*c*)10,857	(*b*)	(*n*)16,349	6,419	(*l*)90,579	(*l*)66,337	(*l*)133,758	(*l*)6,429,996	(*l*)111,300	(*l*)33,475
(*c*) 8,996	(*o*)14,415	5,554	36,615	99,755	70,081	89,279	(*l*)7,133,483	(*l*)173,568	70,176
(*c*) 9,463	(*o*)20,866	7,530	29,605	87,986	66,371	101,059	(*l*)6,448,634	(*l*)180,644	51,002

Year.	Iron and Steel. (*Continued*).								Lead.	
	Puddled and Wrought	Sheets and Plates.	Rails.	Strips and Wire Rods.	Nails, Screws, Rivets, Bolts.	Steel Ingots, Blooms, Billets, etc.	Steel Bars, Shapes, Beams, Pillars.	Mnfrs. Unenumerated. (*h*)	Ore.	Pig Sh
1899..	(*f*)	(*f*)	(*g*)	(*f*)	(*f*)	78,257	(*g*)	$39,527,075	30,263	2
1900..	189,891	(*f*)	38,636	(*f*)	(*f*)	182,210	94,667	17,861,660	21,566	1
1901..	102,811	(*f*)	55,809	(*f*)	(*f*)	185,810	124,648	18,252,435	29,944	2
1902..	178,425	(*f*)	48,942	(*f*)	45,095	285,494	129,743	14,424,835	25,838	2
1903..	196,084	73,079	74,939	35,574	51,888	278,441	343,259	9,007,990	18,923	2
1904..	109,289	69,552	40,438	38,214	50,649	531,069	219,510	8,630,380	8,748	2
1905..	110,576	69,831	34,439	60,318	55,331	613,612	148,995	8,783,895	(*i*) 27,649	2
1906..	111,062	83,747	11,900	61,288	57,071	493,805	149,363	7,824,405	(*i*) 30,795	2
1907..	83,145	57,280	19,33	56,110	51,863	332,442	90,327	3,731,275	(*i*) 13,609	2
1908 (*i*)	85,270	56,652	31,393	109,313	54,251	437,238	231,859	3,034,895	23,861	2
1909..	(*l*) 95,553	(*l*) 68,067	(*l*) 22,430	(*l*) 132,543	(*l*) 59,197	(*l*) 392,008	(*l*) 264,909	2,819,365	(*b*)	(*l*) 2
1910..	(*l*) 98,859	88,245	16,074	163,681	58,977	361,401	349,662	3,239,715	(*b*)	2
1911..	(*l*) 19,688	115,349	36,588	173,764	62,314	512,841	500,643	(*p*) 3,654,825	(*b*)	2

Year.	Manganese Ore.	Mica, Sheet.	Mica and Talc.	Paraffin.	Petroleum. *Liters.*	Phosphate Rock.	Platinum, Wrought and Unwrought. *Kg.*	Potassium Nitrate.	Pyr Iro Co
1899.....	261,740	519	6,025	54,712	908,107,248	426,830	5,404	12,635	71
1900.....	270,098	469	7,952	50,033	965,167,850	361,309	5,027	12,798	78
1901.....	195,736	(*b*)	7,117	42,643	960,650,967	360,568	4,917	12,115	66
1902.....	237,066	1,078	6,127	52,023	1,078,095,152	370,697	3,027	11,526	63
1903.....	235,574	(*b*)	(*b*)	49,163	1,299,570,625	398,997	(*b*)	9,425	74
1904.....	208,458	(*b*)	(*b*)	42,882	1,373,488,176	425,978	(*b*)	12,277	73
1905.....	289,827	(*b*)	(*b*)	41,247	1,364,301,583	427,762	(*b*)	8,260	70
1906.....	314,016	(*b*)	(*b*)	44,673	1,130,667,737	450,058	(*b*)	10,125	77
1907.....	513,750	(*b*)	(*b*)	46,542	1,382,595,355	512,601	(*b*)	10,719	78
1908 (*i*)....	349,694	(*b*)	(*b*)	(*n*) 46,504	(*n*) 1,300,726,576	537,628	(*b*)	(*n*) 11,913	77
1909.....	335,813	(*b*)	(*b*)	(*n*) 53,289	(*n*) 1,355,562,599	459,058	(*b*)	(*n*) 10,704	(*j*) 80
1910.....	489,948	(*b*)	(*b*)	(*n*) 54,073	(*n*) 1,308,396,240	462,905	(*b*)	11,558	82
1911.....	346,676	(*b*)	(*b*)	51,456	1,672,293,900	501,332	(*b*)	11,376	86

Year.	Quicksilver.	Silver Ore. (*d*)	Sodium Nitrate.	Sulphur.	Tin.		Zinc.		
					Ore.	Block, Ingot, Bars or Slabs.	Ore.	Spelter.	Mi
1899.....	1,759	$ 5,162,750	163,387	21,906	6,324	27,608	38,143	71,068	
1900.....	1,113	5,154,430	143,461	22,993	7,449	33,648	42,755	61,504	
1901.....	1,202	5,309,920	108,822	22,440	10,690	35,397	38,660	68,633	
1902.....	1,129	5,383,515	116,791	23,863	12,255	37,713	45,312	89,688	
1903.....	1,187	6,596,045	118,582	21,313	12,473	36,076	41,009	86,539	
1904.....	1,130	8,271,480	122,454	17,629	15,734	39,932	54,438	90,088	
1905.....	1,158	10,426,570	106,107	18,163	(*b*)	40,391	23,909	92,261	
1906.....	1,320	10,532,020	110,222	22,704	21,003	44,306	(*i*) 22,824	95,203	
1907.....	1,341	11,224,650	115,716	15,730	21,205	44,505	(*i*) 66,076	90,756	
1908(*i*).....	1,483	10,743,400	(*n*) 148,056	(*n*) 19,580	25,414	48,496	61,661	91,548	
1909.....	(*m*) 1,469	10,409,800	(*n*) 91,650	21,211	(*m*) 24,471	(*l*) 42,393	(*b*)	104,217	
1910.....	1,517	9,311,665	128,528	20,496	26,490	47,028	(*b*)	123,061	
1911.....	1,584	8,652,850	130,549	21,269	29,300	46,643	(*b*)	117,077	

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported. (*c*) sified as soda compounds since 1901. (*d*) Includes the value of silver in argentiferous ore and meta Includes puddled iron. (*f*) Not separately enumerated. (*g*) Returns not available. (*h*) Prior to many manufactures were not reported separately. (*i*) From *Mines and Quarries.* (*l*) From *Iron an Trades Review.* (*m*) *Min. Journ.* (*n*) *Oil, Paint and Drug Reporter.* (*o*) From Trinidad. (*p*) The besides, not here given, pipes, castings and forgings, amounting to 51,000 tons.

MINERAL EXPORTS OF THE UNITED KINGDOM—DOMESTIC PRODUCTS. (*a*)
(In metric tons or dollars; £1 = $5.)

Year.	Bleaching Materials.	Cement.	Coal.	Coke.	Patent Fuel.	Supplied to Steamers.	Coal Products (*c*).
1899	(*b*)	359,273	41,938,217	881,172	(*b*)	12,422,429	7,712,965
1900	57,478	365,742	46,845,739	1,001,131	(*b*)	11,940,353	9,058,220
1901	46,912	318,216	42,547,114	820,594	1,098,459	13,804,222	5,756,265
1902	40,939	308,104	43,849,591	669,664	1,067,060	15,390,485	5,991,025
1903	49,415	406,388	45,669,258	728,957	970,449	17,068,646	7,290,825
1904	35,289	390,736	46,995,636	779,060	1,257,589	17,465,954	6,879,400
1905	42,526	463,863	48,236,334	786,498	1,126,190	17,674,484	6,742,455
1906	45,510	668,461	56,489,367	828,266	1,399,244	18,887,656	7,226,790
1907	48,856	777,741	64,621,743	997,170	1,504,661	18,917,660	7,726,685
1908(*k*)	(*n*) 39,120	607,950	63,551,057	1,212,184	1,463,557	19,786,734	(*n*) 7,711,360
1909	(*n*) 46,321	607,825	(*l*) 64,089,132	(*l*) 1,180,218	(*l*) 1,479,142	(*l*) 20,029,427	(*n*) 7,276,050
1910	51,188	747,627	63,081,948	979,526	1,494,397	19,839,123	(*n*) 8,398,405
1911	46,418	727,416	65,636,084	1,081,930	1,638,625	19,573,379	.....

Year.	Copper.				Iron.				
	Ingot.	Mixed or Yellow Metal.	Mfres	Sulphate.	Ore.	Pig.	Scrap.	Cast Iron and Mfres.	Wrought Iron, Shapes and Mfres.
1899	32,449	7,038	11,231	40,822	(*d*)	(*e*) 1,401,365	118,262	(*b*)	161,679
1900	18,300	8,940	10,765	43,601	(*d*)	(*e*) 1,450,365	96,567	(*b*)	159,677
1901	26,935	9,252	11,156	36,601	(*d*)	(*e*) 852,609	86,559	(*b*)	119,962
1902	21,658	13,314	14,075	43,995	4,062	1,120,207	104,890	(*b*)	(*b*)
1903	23,723	14,425	16,975	54,307	4,534	1,082,426	143,929	62,249	217,139
1904	14,791	16,704	18,467	71,367	6,706	823,909	166,010	49,004	173,233
1905	21,232	9,959	22,128	55,219	14,664	997,601	151,619	49,193	186,340
1906	19,778	7,149	16,195	43,670	13,415	1,670,753	180,547	54,876	203,521
1907	25,652	7,994	16,676	46,049	15,538	1,978,350	162,295	43,218	215,159
1908(*k*)	14,869	15,546	19,796	(*m*) 72,429	4,478	1,317,330	135,115	48,336	174,539
1909	12,465	(*m*) 11,381	17,248	(*m*) 45,547	(*l*) 5,185	(*l*) 1,026,517	(*l*) 165,183	(*l*) 44,406	(*l*) 171,698
1910	11,898	(*m*) 15,970	12,551	43,391	(*l*) 7,464	(*l*) 1,062,444	(*l*) 150,507	(*l*) 57,148	(*q*) 143,707
1911	10,981	15,087	24,665	81,112	(*l*) 6,802	(*l*) 1,082,970	(*l*) 148,001	(*l*) 70,618	(*q*) 140,253

Year.	Iron. (*Continued.*)								
	Rails.	Wire and Mfrs., of.	Plates and Sheets.	Galvanized Sheets.	Black Plates for Tinning.	Tinned Plates.	Steel Ingots, Billets, Blooms, etc.	Steel Shapes, Beams and Pillars.	Total Iron and Steel, and Mfrs. of.
1899	(*f*) 601,266	50,041	111,773	242,167	86,936	260,735	333,837	(*b*)	3,777,098
1900	379,939	39,104	39,157	251,203	66,810	278,338	313,383	(*b*)	3,602,083
1901	474,073	48,107	36,418	254,290	52,217	275,661	217,236	(*b*)	2,944,083
1902	(*b*)	(*b*)	(*b*)	336,572	58,245	317,201	306,152	(*b*)	3,529,223
1903	613,741	60,800	165,672	357,665	66,279	297,485	13,427	159,330	3,621,635
1904	533,895	61,894	154,774	391,608	63,467	365,262	4,324	176,232	3,315,047
1905	555,390	82,519	207,866	413,533	69,937	360,630	8,735	219,491	3,781,059
1906	470,652	96,641	279,459	450,221	66,749	381,421	11,924	311,231	4,763,868
1907	595,272	103,100	305,399	476,838	72,675	411,814	13,705	344,135	5,249,028
1908(*k*)	589,525	95,801	210,526	396,366	62,079	409,335	2,452	278,792	4,162,065
1909	(*l*) 589,499	(*l*) 107,406	(*l*) 169,918	(*l*) 502,791	(*l*) 61,025	(*l*) 446,781	(*l*) 3,217	(*l*) 309,555	4,278,172
1910	490,068	(*l*) 124,703	(*l*) 197,858	(*l*) 606,530	(*l*) 56,691	(*l*) 490,733	(*l*) 3,340	(*q*) 230,850	4,661,647
1911	501,384	(*l*) 129,252	(*l*) 203,850	(*l*) 627,469	(*l*) 68,137	(*l*) 492,129	(*f*) 3,687	(*q*) 231,924	4,591,641

Year.	Lead. Pig and Mfres.	Salt.	Sodium.				Tin. Block	Zinc.		
			Soda Ash.	Carbonate and Bicarbonate.	Hydrate.	Sulphate.		Ore.	Spelter.	Mfres.
1899 ..	40,923	638,213	(g)193,492	(h)	(h)	(h)	4,785	8,171	5,492	1,249
1900 ..	36,576	556,704	(g)185,783	(h)	(h)	(h)	5,713	13,913	7,136	1,159
1901 ..	38,166	627,078	58,412	22,161	50,624	26,057	5,584	13,981	7,512	1,256
1902 ..	33,537	624,752	59,894	24,654	61,658	35,672	6,210	16,717	6,756	1,345
1903 ..	36,152	594,300	58,605	23,574	59,725	45,630	6,349	15,659	8,192	(i)
1904 ..	35,600	632,605	61,327	25,252	61,985	40,324	5,953	14,606	7,930	(i)
1905 ..	42,265	588,389	67,678	28,425	68,675	33,681	7,741	(b)	7,451	(i)
1906 ..	45,612	629,658	86,232	26,970	72,218	44,448	8,631	(b)	7,962	(i)
1907 ..	44,068	592,989	91,120	29,539	70,432	45,898	8,808	11,511	6,666	(i)
1908(k)	50,221	532,101	(n) 83,713	(n)20,805	(n) 73,769	(n)22,974	9,486	3,833	8,537	(i)
1909 ..	(m) 46,399	523,690	(n) 97,480	(n)23,378	(n) 80,439	(n)35,587	11,373	(b)	(m) 8,684	(i)
1910 ..	47,588	571,845	110,188	23,295	84,291	58,960	12,576	(b)	9,247	(i)
1911 ..	44,733	614,572	125,945	26,353	82,356	59,382	11,736	(b)	9,736.	(i)

(a) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (b) Not reported. (c) Including naphtha, paraffin, paraffin oil and petroleum. (d) Previous reports not available. (e) Includes puddled iron. (f) Includes railroad material of all kinds. (g) Includes all soda compounds; not sep rate; enumerated previous to 1901. (h) Included under soda ash. (i) Included under spelter. (k) From *Mines and Quarries.* (l) From *Iron and Coal Trades Review.* (m) *Mining Journal.* (n) From *Oil Paint and Drug Reporter.* (q) Bars only.

UNITED STATES

Of the following tables, the first records the imports of foreign mineral and metal products into the United States, whether dutiable or duty free; the second shows the exports of materials produced in the United States: and the third reports the re-exports of products of foreign origin. These statistics are as reported by the Bureau of Statistics of the Department of Commerce and Labor, and special acknowledgment is due to Hon. O. P. Austin, chief of the bureau, for furnishing the figures for many substances which are not reported in the Monthly Summary. The complete statement of production in the United States is given on an early page in this volume.

IMPORTS. (a)

Year.	Aluminum. Crude. Lb.	Kg.	Value.	Value per Lb.	Ammonium Sulphate. Lb.	Metric Tons.	Value.	Value per Lb.
1902	745,217	338,028	$215,032	0.290	35,535,558	16,119	$858,036	$0.024
1903	498,655	226,190	139,298	0.279	29,104,817	13,199	565,230	0.026
1904	515,416	234,293	128,350	0.249	39,859,690	18,077	1,058,981	0.027
1905	530,429	240,284	106,108	0.200	15,512,399	7,038	416,048	0.027
1906	770,713	349,195	154,292	0.200	31,797,291	14,423	894,663	0.028
1907	872,474	395,754	181,352	0.208	70,440,992	31,960	1,828,236	0.026
1908	465,317	210,785	80,268	0.173	76,475,104	34,698	1,982,830	0.026
1909	5,109,843	2,317,784	745,963	0.145	85,829,334	38,932	2,114,694	0.024
1910	(o)12,271,277	5,567,730	1,844,830	0.151	184,686,534	83,795	4,668,820	0.025
1911	4,173,308	1,893,000	598,272	0.143	189,265,797	85,850	5,066,470	0.027

Year.	Antimony. Lb.	Metric Tons.	Value.	Value per Lb.	Antimony Ore. Lb.	Metric Tons.	Value.	Value per Lb.
1902	5,742,703	2,605	$347,899	$0.061	1,639,043	743	$29,476	$0.018
1903	5,125,515	2,325	279,957	0.054	2,673,142	1,213	51,489	0.019
1904	4,056,299	1,840	235,401	0.058	2,487,602	1,129	50,362	0.020
1905	5,737,891	2,603	431,774	0.075	1,976,694	897	52,868	0.027
1906	7,900,194	3,583	1,417,816	0.179	2,247,131	1,019	128,347	0.057
1907	8,662,683	3,928	1,423,276	0.164	2,780,186	1,261	180,903	0.065
1908	8,114,651	3,954	572,979	0.071	3,280,922	1,488	106,930	0.033
1909	9,652,568	4,335	619,179	0.064	3,386,708	1,575	94,249	0.027
1910	9,899,514	4,487	572,031	0.058	*Nil.*	.....	.....	.....
1911	10,957,844	4,970	573,564	0.052	*Nil.*	.....	.....	.....

Year.	Asbestos. Crude Value.	Mfd. Value.	Total Value.	Asphaltum. Long Tons.	Metric Tons.	Value.	Value per L. T.
1902	$729,421	$33,013	$762,434	139,944	142,183	$439,570	$3.09
1903	657,269	32,058	689,327	167,554	170,235	514,051	3.06
1904	700,572	51,290	751,862	119,575	121,489	510,524	4.27
1905	776,362	70,117	846,749	86,748	88,136	382,667	4.41
1906	536,500	.....	.....	97,274	98,830	388,010	3.93
1907	1,104,109	200,371	1,304,480	127,902	129,948	518,074	4.05
1908	1,068,342	147,548	1,215,869	131,862	133,971	587,698	4.45
1909	993,254	240,381	1,233,635	132,807	134,939	646,655	4.87
1910	1,235,170	308,078	1,543,248	166,379	169,041	776,283	4.66
1911	1,413,541	161,442	1,574,983	174,234	177,030	778,185	4.47

Year.	Arsenic.(b)				Barytes.				Bauxite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1902....	(f)	.....	.....	.....	(f)	.....	.....	.....	15,790	16,043	$54,410	$3.45
1903....	7,391,566	3,241	$256,097	$0.036	6,344	6,446	$22,777	$3.59	14,889	15,127	49,684	3.34
1904....	6,391,566	2,900	226,481	0.036	6,689	6,796	27,463	4.11	15,475	15,723	49,577	3.20
1905....	6,444,083	2,924	219,198	0.034	7,879	8,005	36,796	4.67	11,726	11,914	46,517	3.96
1906....	7,639,507	3,464	336,609	0.044	4,293	4,362	37,296	8.69	17,809	18,094	63,221	3.55
1907....	9,922,870	4,500	553,440	0.056	28,350	28,804	174,225	6.15	25,065	25,466	93,208	3.71
1908....	9,592,881	4,558	417,137	0.056	n)12,196	12,390	58,822	4.83	21,679	22,033	87,823	4.05
1909....	7,183,644	3,259	272,493	0.038	n)13,091	13,301	54,707	4.19	18,989	18,689	83,956	4.46
1910....	8,204,123	3,722	250,115	0.035	n)16,960	17,226	48,457	2.86	14,038	14,258	65,743	4.68
1911....	5,404,263	2,451	159,626	0.030	20,860	21,195	58,726	2.82	43,222	43,916	164,301	3.80

Year.	Chrome Ore.				Bismuth.				Coal, Anthracite.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value	Value per L.T.
1902 .	39,570	40,203	$582,597	$14.72	.....	.....	.....	.....	73,006	74,174	$323,517	$4.43
1903 .	22,932	23,299	302,025	13.17	147,324	66,826	$235,199	$1.59	151,023	153,439	675,623	4.47
1904 .	24,227	24,615	348,527	14.38	147,712	67,002	268,837	1.82	72,526	73,686	220,665	3.04
1905 .	54,434	55,305	725,301	13.32	148,589	67,459	318,007	2.14	34,262	34,810	107,394	3.13
1906 .	43,441	44,136	557,594	12.84	254,733	115,000	318,452	1.25	32,357	32,875	105,190	3.25
1907 .	41,999	42,671	491,925	11.71	259,881	117,882	325,015	1.25	9,896	10,054	40,966	4.14
1908 .	27,876	28,320	345,960	12.40	164,793	73,002	257,397	1.56	16,483	16,747	73,777	4.47
1909 .	39,820	40,459	460,758	11.57	183,413	83,195	286,516	1.56	4,709	4,785	19,438	4.13
1910 .	38,579	39,196	415,768	10.77	198,174	89,916	332,674	1.67	8,196	8,327	42,234	5.14
1911 .	37,529	38,131	407,958	10.87	172,093	78,061	311,771	1.81	2,463	2,502	12,550	5.10

Year.	Chloride of Lime.				Cement.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Barrels. (c)	Metric Tons.	Value.	Value per Bbl.
1902.....	112,374,478	50,973	$1,456,435	$0.013	1,994,790	361,932	$2,581,883	$1.29
1903.....	113,285,240	51,586	912,843	0.008	2,317,951	420,569	3,027,111	1.30
1904.....	87,909,168	39,876	707,174	0.008	1,046,404	189,910	1,382,913	1.32
1905.....	104,919,462	47,604	843,285	0.008	846,577	153,644	1,102,041	1.30
1906.....	105,221,371	47,718	863,490	0.008	2,205,712	400,115	2,950,268	1.33
1907.....	112,090,783	50,833	939,248	0.008	2,006,228	363,929	2,637,424	1.31
1908.....	74,602,059	33,848	621,713	0.008	839,246	152,313	1,189,560	1.42
1909.....	91,390,004	41,454	743,636	0.008	431,785	78,342	642,397	1.49
1910.....	101,029,345	45,839	797,260	0.008	292,314	53,053	396,428	1.36
1911.....	82,895,472	37,601	667,804	0.008	163,802	29,720	254,258	1.55

Year.	Coal, Bituminous.				Total Coal.		Coke.			
	Long Tons.	Metric Tons	Value.	Value per L.T.	Long Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1902....	2,478,375	2,518,029	$7,012,674	$2.84	2,551,381	$7,339,791	107,437	109,156	$423,774	$4.03
1903....	3,295,379	3,348,105	9,329,221	2.83	3,446,402	10,004,844	127,479	129,519	437,625	3.43
1904....	1,556,149	1,581,047	3,915,613	2.52	1,628 675	4,136,278	161,476	164,060	648,520	4.01
1905....	1,618,581	1,644,478	3,908,877	2.42	1,652,843	4,016,271	181,376	184,278	796,544	4.39
1906....	1,712,150	1,739,544	4,129,555	2.41	1,744,507	4,234,745	114,703	116,538	558,419	4.87
1907....	2,116,122	2,149,980	5,398,167	2.55	2,126,018	5,439,133	132,536	134,656	594,137	4.48
1908....	1,487,816	1,511,621	4,059,786	2.73	1,504,299	4,133,563	129,591	131,624	603,964	4.66
1909....	1,257,629	1,277,814	3,597,991	2.56	1,262,338	3,617,429	170,671	173,410	735,253	4.31
1910....	1,991,943	2,023,914	5,381,062	2.69	2,000,270	5,423,296	156,417	158,920	625,619	4.00
1911....	1,238,808	1,258,691	3,611,518	2.92	1,241,271	3,624,068	69,515	70,631	254,938	3.67

Year.	Cobalt Oxide.				Copper, Ore and Matte.			
	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
2.....	79,984	36,281	$151,115	$1.89	181,566	184,470	$8,695,780	$47.89
3.....	73,350	33,272	145,264	1.98	284,912	289,471	3,177,582	11.15
4.....	42,352	19,211	86,925	2.05	268,234	272,527	4,308,410	16.06
5.....	70,048	31,802	139,377	1.99	296,251	300,991	5,765,238	19.46
6.....	41,084	18,652	83,167	2.02	208,702	212,041	6,796,696	32.56
7.....	42,794	19,421	73,028	1.71	291,957	297,096	9,048,270	31.32
8.....	1,550	701	3,095	2.00	288,022	292,630	6,978,513	24.20
9.....	9,818	4,453	11,065	1.132	393,530	399,846	9,113,254	23.16
0.....	6,124	2,778	4,806	0.785	453,747	461,007	9,182,161	20.23
1.....	22,934	10,403	11,047	0.482	292,931	297,633	6,935,794	23.68

ear.	Copper, Ingots, Old, etc.				Cryolite.			
	Lb.	Metric. Tons.	Value.	Value. per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
2.....	103,129,568	46,778	$13,051,159	$0.126	6,188	6,287	$85,640	$13.83
3.....	136,707,995	62,011	17,262,148	0.126	7,708	7,831	102,879	13.35
4.....	142,344,433	64,567	18,374,959	0.129	959	974	13,708	14.30
5.....	160,619,385	72,876	22,103,741	0.137	1,600	1,623	22,482	14.05
6.....	176,558,390	80,069	30,416,578	0.172	1,505	1,529	29,683	19.72
7.....	192,901,267	87,523	38,658,754	0.200	1,438	1,461	28,920	20.10
8.....	162,224,144	73,604	22,851,134	0.141	1,124	1,142	16,445	14.63
9.....	240,713,721	109,186	30,529,425	0.127	1,278	1,299	18,427	14.42
0.....	259,210,796	117,609	31,620,689	0.122	36	.....	2,343	.....
1.....	265,940,760	120,630	31,540,827	0.119	2,007	2,039	47,093	23.46

ear.	Emery Grains.				Emery Rock.			
	Lb.	Metric. Tons.	Value.	Value per Lb.	Long Tons.	Metric. Tons.	Value.	Value per L. T.
2....	1,665,737	756	$60,079	$0.036	7,166	7,281	$151,959	$21.21
3....	3,595,239	1,630	109,272	0.030	10,885	11,059	188,985	17.36
4....	2,281,193	1,035	109,772	0.048	7,054	7,167	131,493	18.64
5....	3,209,915	1,456	143,729	0.045	11,073	11,250	185,689	16.77
6....	4,655,168	2,113	215,357	0.043	13,840	14,061	286,386	20.69
7....	4,282,228	1,942	186,156	0.043	11,235	11,415	211,184	18.80
8....	1,735,366	788	89,702	0.052	8,084	8,213	146,105	18.09
9....	2,696,960	1,224	132,264	0.049	9,836	9,993	186,930	19.00
0....	1,189,664	535	53,709	0.045	16,770	16,321	344,421	20.54
1....	712,134	323	35,384	0.050	10,232	10,396	176,890	17.29

ear.	Phosphates, Crude.				Pig Iron.			
	Long Tons.	Metric. Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
2....	137,386	139,584	$646,264	$4.70	619,354	629,264	$10,935,831	$17.66
3....	132,965	134,092	679,112	5.11	599,574	609,167	11,173,302	18.64
4....	130,214	132,297	745,744	5.73	79,590	80,772	1,765,107	22.20
5....	56,021	56,917	273,289	4.88	212,465	215,864	5,185,784	24.41
6....	23,281	23,653	147,547	6.34	379,828	385,905	11,851,210	31.20
7....	25,876	26,290	163,944	6.34	489,475	497,305	13,418,982	27.42
8....	26,734	27,161	175,365	6.56	92,202	93,677	2,886,339	31.35
9....	11,903	12,094	97,277	8.18	176,442	179,265	5,112,045	28.97
0....	19,384	19,688	235,040	12.12	237,233	241,029	6,549,938	27.61
1....	16,153	16,412	157,394	9.74	148,459	150,842	4,380,334	29.50

Year.	Fuller's Earth.		Gold.		Iron Ore.			
	Long Tons.	Value.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per l. T.
1902...	.....	.....	$22,710,957	$21,482,360	1,165,470	1,184,118	$2,583,077	$2.22
1903...	15,267	$120,671	44,054,902	21,212,794	980,440	996,127	2,261,008	2.31
1904...	9,126	78,006	75,646,128	9,157,106	487,613	495,415	1,101,384	2.26
1905...	13,001	105,997	38,564,328	11,729,007	845,651	859,181	2,062,161	2.44
1906...	13,238	108,696	139,705,887	15,873,493	1,060,390	1,007,356	2,967,434	2.80
1907...	14,648	122,221	130,605,413	12,792,659	1,229,168	1,248,835	3,937,483	3.20
1908...	10,963	92,413	38,346,267	11,930,026	776,898	789,326	2,224,248	2.86
1909...	11,406	101,151	30,648,147	13,438,819	1,694,957	1,722,161	4,579,078	2.70
1910...	13,775	132,545	47,212,754	12,009,764	2,591,031	2,632,617	7,832,225	3.02
1911...	16,272	143,594	46,623,766	10,821,418	1,811,732	1,840,810	5,412,636	2.99

Year.	Scrap, Iron and Steel.			Bar Iron.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902.....	109,510	111,262	$1,606,720	28,844	29,307	$1,286,238
1903.....	82,921	84,248	1,273,941	43,392	44,090	1,904,469
1904.....	13,461	13,676	189,506	20,905	21,247	917,254
1905.....	23,731	24,111	370,328	37,294	37,891	1,522,434
1906.....	19,091	19,397	248,106	35,793	36,366	1,590,302
1907.....	27,652	28,094	368,842	39,746	40,382	1,774,441
1908.....	5,090	5,171	61,981	19,671	19,980	837,585
1909.....	63,504	64,523	781,426	19,210	19,518	806,862
1910.....	72,764	73,928	928,002	38,231	38,783	1,565,999
1911.....	17,272	17,549	190,285	26,730	27,159	1,202,351

Year.	Rails.			Hoop, Band or Scroll.			Ingots, Blooms, Slabs, Billets, etc.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902...	63,522	64,538	$1,576,679	3,362	3,416	$131,052	289,318	293,965	$7,943,818
1903...	95,555	97,083	2,159,273	1,525	1,550	74,898	261,570	265,932	7,331,299
1904...	37,776	38,380	808,775	2,135	2,169	60,934	(m)10,807	10,980	1,537,531
1905...	17,278	17,554	409,807	4,772	4,848	137,612	(m)14,641	14,875	2,072,606
1906...	4,943	5,022	137,104	10,231	10,395	256,836	(m)21,337	21,678	3,010,589
1907...	3,752	3,812	104,958	1,508	1,532	82,706	(m)19,334	19,643	3,004,178
1908...	1,719	1,752	53,128	1,110	1,127	75,920	11,212	11,391	1,437,514
1909...	1,542	1,567	36,963	(§)	(§)	(§)	19,289	19,599	2,695,630
1910...	(§)	(§)	(§)	(§)	(§)	(§)	46,578	47,323	4,075,038
1911...	(§)	(§)	(§)	(§)	(§)	(§)	29,205	29,687	2,772,614

Year.	Sheet, Plate and Taggers Iron or Steel.			Tin Plates, Terne Plates and Taggers Tin.			Wire Rods.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902...	7,156	7,270	$545,739	60,115	61,080	$4,023,421	21,382	21,725	$1,033,074
1903...	11,557	11,741	540,272	47,360	48,118	2,999,252	20,836	21,169	1,028,977
1904...	4,165	4,232	302,500	70,652	71,782	4,354,761	15,313	15,558	707,779
1905...	2,236	2,272	242,955	65,740	66,792	4,090,523	17,616	17,898	800,027
1906...	3,231	3,283	325,276	56,982	57,894	3,883,225	17,799	18,084	876,270
1907...	3,749	3,809	367,140	57,773	58,697	4,462,522	17,076	17,349	851,371
1908...	2,628	2,669	377,549	58,492	59,426	3,651,576	11,208	11,387	543,170
1909...	4,711	4,787	536,841	62,593	63,598	3,782,952	10,544	10,613	531,652
1910...	6,107	6,205	461,632	67,086	68,159	4,502,862	20,374	20,700	1,024,531
1911...	2,492	2,532	274,945	14,089	14,324	1,082,417	13,728	15,980	731,291

Year.	Wire and Articles Made from.			Total Iron Imports. (e)	Lead in Ore and Base Bullion.		
	Long Tons.	Metric Tons.	Value.		Short Tons.	Metric Tons	Value.
1902	3,469	3,525	$606,724	$41,468,826	105,186	95,425	$4,424,511
1903	5,018	5,098	728,430	41,258,864	103,384	93,790	3,596,635
1904	3,956	4,019	624,892	21,621,970	104,127	94,464	3,517,691
1905	3,978	4,042	705,465	26,401,283	92,657	84,081	3,565,282
1906	6,610	6,716	1,079,868	34,827,132	72,371	65,640	3,490,750
1907	(f)	(f)	1,551,415	38,789,992	70,538	64,019	3,579,990
1908	(f)	(f)	1,003,973	19,957,385	109,315	99,168	4,384,904
1909	(f)	(f)	1,117,812	30,571,542	110,605	100,339	4,121,380
1910	(f)	(f)	1,468,741	47,115,112	105,117	106,805	4,056,722
1911	(f)	(f)	1,270,426	28,995,600	87,319	79,215	3,452,695

Year.	Lead in Pigs and Old.			Lead, Sheet, Pipe, Shot, etc.			Other Lead Mfres.	Total Lead.
	Short Tons.	Metric Tons.	Value.	Lb.	Metric Tons.	Value.		
1902	2,529	2,294	$132,500	224,208	102	$7,765	$18,918	$4,533,694
1903	3,023	2,742	164,528	17,008	8	810	8,071	3,770,044
1904	8,724	7,914	461,316	69,581	32	2,441	7,755	3,989,203
1905	5,720	5,812	367,106	54,779	25	2,638	4,580	3,939,606
1906	11,763	10,669	910,417	346,177	157	17,250	20,681	4,401,167
1907	9,277	8,414	846,166	734,418	333	39,210	12,736	4,426,156
1908	2,759	2,504	182,503	42,376	19	2,026	46,486	4,567,407
1909	3,576	3,244	230,347	40,434	18	2,056	33,892	4,351,727
1910	3,491	3,166	202,376	(f)	.....	.....	7,073	4,259,098
1911	2,632	2,388	190,543	(f)	.....	.....	9,834	3,642,238

Year.	White Lead.				Litharge.				Red Lead.			
	Lb.	Metric Tons.	Value.	Value per lb.	Lb.	Metric Tons.	Value.	Value per lb.	Lb.	Metric Tons.	Value.	Value per lb.
1902	506,423	230	$25,320	$0.050	88,115	40	$2,908	$0.033	1,075,839	488	$37,383	$0.035
1903	453,284	206	24,595	0.054	42,756	19	1,464	0.034	1,152,715	523	40,846	0.035
1904	587,338	266	33,788	0.058	44,541	20	1,500	0.034	836,077	379	30,115	0.036
1905	597,510	271	34,722	0.058	117,757	53	4,139	0.035	704,402	320	26,553	0.038
1906	647,636	294	41,233	0.064	87,230	40	3,737	0.043	1,093,619	497	50,741	0.046
1907	584,309	265	37,482	0.064	90,475	41	4,388	0.048	679,171	308	35,959	0.053
1908	540,311	245	30,451	0.056	96,184	44	3,327	0.035	645,073	292	28,155	0.034
1909	694,599	315	39,963	0.057	90,655	32	3,740	0.041	760,179	345	30,428	0.040
1910	686,052	311	38,919	0.057	48,693	22	2,252	0.046	822,289	373	32,750	0.039
1911	741,071	336	46,213	0.062	24,662	11	1,196	0.048	1,163,533	528	46,170	0.040

Year.	Orange Mineral.				Magnesite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L.T.
1902	997,494	452	$49,060	$0.049	45,157	45,880	$373,928	$8.28
1903	756,742	343	36,407	0.048	49,684	50,479	461,399	9.29
1904	766,469	348	37,178	0.049	35,106	35,668	286,828	8.17
1905	628,003	285	31,106	0.049	66,405	67,566	638,619	9.46
1906	770,342	350	42,519	0.055	80,711	82,002	863,492	10.70
1907	615,015	279	37,793	0.061	88,400	89,814	875,359	9.90
1908	485,407	220	26,645	0.055	75,442	76,648	736,763	9.80
1909	496,231	225	27,562	0.056	102,045	103,683	985,019	9.66
1910	600,461	269	32,199	0.053	152,078	154,511	1,542,800	10.14
1911	504,734	229	28,515	0.056	200,180	203,483	1,114,520	5.57

Year.	Manganese Ore.				Mica.	Nickel. (h)	Nickel Ore and Matte.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.			Long Tons.	Metric Tons.	Value.
1902.....	235,576	239,345	$1,931,282	$8.20	$466,332	.....	(f)	.....	.....
1903.....	146,056	148,393	1,278,108	8.75	317,969	$207,954	11,936	16,191	$1,285,93
1904.....	108,519	110,255	901,592	8.31	269,808	206,021	8,549	8,685	915,47
1905.....	257,033	261,146	1,952,407	7.60	403,755	335,211	13,451	13,666	1,626,92
1906.....	221,260	224,800	1,696,043	7.67	1,042,608	86,336	15,156	15,398	1,816,61
1907.....	209,021	211,236	1,793,143	8.59	915,259	90,153	16,888	17,158	2,153,97
1908.....	178,023	181,054	1,350,223	7.59	264,755	101,398	16,322	16,582	2,396,21
1909.....	212,765	216,180	1,405,329	6.60	493,978	104,019	18,578	18,876	2,927,97
1910.....	242,348	246,226	1,711,131	7.06	721,541	139,772	28,519	28,975	4,085,07
1911.....	176,852	179,690	1,186,791	6.71	469,089	104,160	23,993	24,378	3,918,53

Year.	Oil, Mineral.			Platinum, Unmanufactured.				Platinum Mfrs.
	Gal.	Value	Value per Gal.	Lb. Troy.	Kg.	Value.	Value per Lb. Troy.	
1902....	3,578,393	$207,310	$0.058	8,670	3,235	$1,950,362	$224.96	$37,61
1903....	4,266,974	261,199	0.061	9,540	3,561	1,921,772	201.44	1,727,83
1904....	4,846,681	277,399	0.057	8,648	3,230	1,812,242	209.55	165,69
1905....	13,725,720	672,127	0.049	8,681	3,240	1,985,107	228.67	188,13
1906....	21,045,316	1,061,076	0.050	13,928	5,198	3,601,021	258.54	187,60
1907....	20,505,197	1,037,728	0.051	7,515	2,805	2,509,926	333.99	175,65
1908....	9,289,376	393,050	0.042	4,155	1,551	1,096,615	263.92	134,11
1909....	3,862,445	198,540	0.051	9,904	3,696	2,557,574	258.30	410,96
1910....	24,323,829	1,438,609	0.059	10,009	3,736	3,320,699	331.77	1,717,38
1911....	69,019,304	2,406,581	0.035	2,868	1,070	1,278,293	445.71	3,587,90

Year.	Potassium Salts.											
	Chlorate.			Chloride.			Chromate and Bichromate.			Nitrate.		
	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.
		$	$		$	$		$	$		$	$
1902....	1,209,148	60,429	0.050	180,980,460	2,141,553	0.015	231,009	15,161	0.066	10,505,474	299,415	0.0
1903....	468,042	19,308	0.041	169,337,673	2,550,478	0.015	41,229	2,784	0.067	13,835,668	367,721	0.0
1904....	95,889	4,209	0.044	174,865,872	2,832,554	0.016	26,053	1,817	0.069	14,184,287	376,931	0.0
1905....	42,510	2,876	0.067	214,207,064	3,326,748	0.016	59,650	4,225	0.070	9,911,534	304,596	0.0
1906....	45,873	3,103	0.068	223,203,387	3,858,895	0.017	30,098	2,102	0.080	11,326,256	371,595	0.0
1907....	12,980	959	0.074	252,303,441	4,175,353	0.017	18,171	1,307	0.072	18,291,890	574,977	0.0
1908....	17,607	1,447	0.082	214,338,887	3,415,326	0.016	216,080	15,453	0.072	16,118,160	470,116	0.0
1909....	22,425	1,837	0.082	298,854,649	4,780,106	0.016	640,623	31,798	0.050	14,883,849	437,660	0.0
1910....	410,770	26,313	0.064	381,873,875	5,252,373	0.014	406,791	19,569	0.048	11,496,904	333,854	0.0
1911....	27,856	2,450	0.088	509,119,193	7,651,684	0.015	22,408	2,159	0 096	7,945,747	265,061	0.0

Year	Potassium Salts. All Other.		Precious Stones.			Pyrites. (i)			
	Lb.	Value.	Uncut.	Cut, not Set.	Jewelry.	Long Tons.	Metric Tons.	Value.	Value per L.
1902.	91,857,009	$1,820,585	$8,282,760	$18,494,288	(f)	437,319	444,316	$1,623,430	$3.7
1903.	70,205,850	1,593,380	10,374,877	15,428,819	$954,456	427,319	434,156	1,636,450	3.8
1904.	74,720,241	1,678,699	10,316,615	16,934,090	803,952	413,585	420,202	1,533,564	3.7
1905.	82,935,632	1,891,081	10,206,350	26,699,670	801,566	515,722	520,926	1,780,800	3.4
1906.	30,302,735	763,513	11,937,542	32,201,949	988,766	597,347	606,903	2,138,746	3.5
1907.	91,299,496	2,220,685	8,740,278	23,706,975	1,069,373	256,479	666,981	2,637,485	4.0
1908.	69,382,278	1,721,626	2,367,189	11,660,442	720,502	668,115	678,804	2,624,339	3.9
1909.	100,180,417	2,445,526	9,230,287	34,340,269	1,267,457	692,385	703,498	2,428,638	3.5
1910.	116,820,873	2,777,396	9,426,647	32,424,471	1,907,148	806,590	819,495	2,773,627	3.4
1911.	155,140,643	3,909,361	9,963,393	30,224,826	1,470,167	1,001,944	1,018,025	3,788,632	3.7

Year.	Salt.				Silver.		Sodium Nitrate.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1902	188,775	167,481	$654,990	$3.47	$ 8,502,614	$17,900,321	205,245	208,529	$5,996,205	$29.21
1903	157,201	142,494	489,179	3.11	7,935,844	16,038,664	272,947	277,314	8,700,806	31.88
1904	167,295	151,810	515,822	3.08	11,865,805	14,221,237	228,012	231,660	9,333,613	32.41
1905	158,449	143,783	491,079	3.10	16,472,911	19,466,224	321,231	326,371	11,206,548	34.89
1906	170,505	154,648	502,583	2.95	20,402,738	23,825,103	372,222	378,178	14,115,206	37.92
1907	153,435	139,166	452,227	2.95	17,652,679	28,259,681	364,610	370,444	14,844,675	40.71
1908	156,608	142,043	440,484	2.81	14,169,524	28,054,606	310,713	315,684	11,385,393	36.68
1909	158,487	143,777	447,983	2.83	15,728,756	30,458,946	428,429	.....	13,608,195	31.43
1910	142,549	129,320	388,015	2.72	15,913,668	29,964,500	529,172	605,938	16,601,328	31.37
1911	137,759	124,974	378,083	2.74	16,351,154	27,395,417	544,878	553,623	16,814,256	30.86

Year.	Sodium Hydroxide (Caustic).			Soda Ash and Carbonate.			All other Sodium Salts.	
	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.
1902	3,334,697	$ 77,482	$0.020	31,889,252	$284,634	$0.009	17,151,682	$283,745
1903	2,970,426	73,647	0.025	25,313,370	228,041	0.009	14,272,646	268,738
1904	2,570,984	64,405	0.025	23,631,832	205,496	0.009	10,399,711	281,527
1905	2,245,789	56,515	0.025	15,754,979	146,812	0.009	11,257,629	247,413
1906	1,209,053	35,262	0.022	6,800,288	71,013	0.010	.....	.....
1907	1,297,070	37,894	0.029	6,198,136	66,521	0.011	8,481,979	258,262
1908	874,813	26,079	0.029	3,515,933	38,372	0.011	.....	296,777
1909	942,982	29,771	0.032	153,928	3,543	0.023	13,805,869	350,396
1910	2,973,522	70,901	0.027	314,396	4,269	0.014	35,895,668	406,643
1911	990,562	34,281	0.035	3,214,129	36,252	0.011	22,202,469	303,934

Year.	Sulphur.									
	Crude.				Flowers.			Refined.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902	176,951	179,782	$3,360,562	$19.00	738	750	$19,954	14	15	$369
1903	188,888	191,910	1,649,756	19.32	1,854	1,883	52,680	189	192	7,254
1904	128,885	130,947	2,463,779	19.12	1,332	1,353	39,133	204	207	9,776
1905	83,201	84,532	1,522,005	18.29	572	581	16,037	778	709	19,960
1906	72,404	73,562	1,282,873	17.72	1,100	1,118	29,565	709	720	17,928
1907	20,399	20,725	355,944	17.45	1,458	1,481	41,216	606	616	14,589
1908	20,118	20,441	318,577	15.83	793	804	22,562	692	700	17,227
1909	26,914	27,346	458,954	17.05	770	782	23,084	966	982	26,021
1910	28,647	29,105	495,988	17.31	915	930	30,180	990	1,004	25,869
1911	24,250	24,639	436,725	18.01	3,891	3,953	83,491	986	1,002	24,906

Year.	Talc.				Tin.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1902	2,859	2,594	$35,336	$12.35	85,043,353	38,575	$21,263,337	$0.250
1903	1,790	1,623	19,635	11.00	83,133,847	37,702	22,265,367	0.268
1904	3,268	2,964	36,370	11.13	83,168,657	37,718	22,356,896	0.270
1905	4,000	3,630	48,225	12.06	89,227,698	40,507	26,316,023	0.294
1906	5,643	5,118	67,818	12.02	101,027,188	45,816	37,446,508	0.371
1907	10,060	10,221	126,391	12.56	82,548,838	37,436	32,075,091	0.389
1908	7,429	6,738	97,296	13.08	82,503,190	37,433	23,932,560	0.290
1909	8,377	7,599	102,964	12.29	95,350,020	43,250	27,559,937	0.289
1910	8,378	7,560	106,451	12.72	105,137,740	47,703	33,921,449	0.322
1911	7,113	6,453	88,050	12.38	106,936,872	48,506	43,390,639	0.406

Year.	Zinc.							
	Blocks, Pigs and Old.				Oxide. (*j*)		Sulphide.	
	Lb.	Metric Tons.	Value.	Value. per Lb.	Lb.	Value.	Lb.	Value.
1902	1,238,091	561	$46,713	$0.038	3,434,466	.....	1,247,936	$32,879
1903	728,614	330	30,900	0.042	3,653,076	$188,495	1,229,806	33,077
1904	933,474	423	44,455	0.048	2,809,905	165,110	1,228,875	31,382
1905	1,042,081	473	51,052	0.048	3,779,311	196,220	1,235,360	33,308
1906	4,407,481	1,999	253,310	0.057	4,494,014	288,065	1,286,469	40,112
1907	3,555,890	1,613	210,322	0.059	5,311,318	323,551	1,570,073	51,435
1908	1,762,627	799	85,885	0.049	4,635,101	262,876	1,048,109	46,733
1909	19,340,029	8,772	826,588	0.043	6,654,352	397,084	1,263,316	44,873
1910	6,904,389	3,133	289,689	0.042	6,470,097	391,670	3,726,135	99,954
1911	3,275,340	1,486	170,634	0.052	5,561,016	357,466	6,355,312	166,199

'(*a*) From Summary of Commerce and Finance of the United States. (*b*) Includes arsenic sulphide. (*c*) 400 lb. (*e*) Not including iron ore. (*f*) Not reported, (*g*) Includes pig and old. (*h*) Includes nickel o in which nickel is the principal constituent and manufactures of nickel. (*i*) Containing more than 25 per ce (*j*) Includes white pigments containing zinc but not lead, dry and in oil. (*m*) Includes bars of steel and not elsewhere specified. The high value is due to the value of "high-speed" steel. (*n*) Crude. (*o*) Crude alloys in which aluminum is chief component. (*p*) Crude and refined. (*q*) Also included in all other salts ash only.

EXPORTS OF DOMESTIC PRODUCTS. (*a*)

Year.	Aluminum and Mfrs. of.	Asbestos and Mfrs of.	Cement.			
			Bbl.(*i*)	Metric Tons.	Value.	Va
1902	$116,052	$130,437	340,821	61,838	$526,471	
1903	157,187	158,360	285,463	51,748	433,984	
1904	166,876	223,096	774,940	140,898	1,104,086	
1905	290,777	.....	1,016,236	185,345	1,387,706	
1906	364,251	259,760	583,299	105,811	944,886	
1907	304,938	200,371	900,550	163,360	1,450,841	
1908	330,092	296,890	846,785	153,638	1,249,229	
1909	567,375	322,523	1,056,922	191,764	1,417,534	
1910	949,215	348,716	2,475,957	449,314	3,477,981	
1911	1,158,603	448,395	3,135,409	568,886	4,632,215	

Year	Coal.								
	Anthracite.				Bituminous.				C
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.
1902	907,977	922,505	$4,301,946	4.73	5,218,969	5,302,472	$13,927,063	$2.66	392,491
1903	2,008,857	2,040,999	9,780,044	4.86	6,303,241	6,404,093	17,410,385	2.76	416,385
1904	2,228,392	2,264,046	11,077,570	4.97	6,345,126	6,446,648	17,160,538	2.74	523,090
1905	2,229,983	2,265,663	11,104,654	4.98	6,959,265	7,070,613	17,867,964	2.56	599,054
1906	2,216,969	2,252,441	10,896,200	4.91	7,704,850	7,828,128	(*k*)19,787,459	2.57	765,190
1907	2,698,072	2,741,241	13,217,985	4.90	(*m*)10,454,677	10,621,950	(*m*)26,982,111	2.54	874,689
1908	2,752,358	2,796,394	13,524,595	4.92	(*m*) 9,100,819	9,246,431	(*m*)23,361,914	2.53	620,923
1909	2,842,714	2,888,340	14,141,468	4.97	(*m*) 9,693,843	9,849,429	(*m*)24,300,050	2.51	895,461
1910	3,021,627	3,070,124	14,785,387	4.89	(*m*)10,784,239	10,957,326	(*m*)26,685,405	2.47	879,073
1911	3,553,999	3,611,041	18,093,285	5.09	13,878,754	14,101,508	34,499,989	2.49	914,042

Year.	Copper.								
	In Ore and Matte (*b*).				Ingots, Bars, Plates and Old.				Mfrs.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.	Value.
1902	18,035	18,321	$1,326,131	$73.53	354,668,849	160,876	43,392,800	$0.122	$2,092,798
1903	12,291	12,488	855,367	69.59	310,729,524	140,920	41,170,059	0.132	2,339,729
1904	18,927	19,230	1,202,537	63.54	554,550,030	251,497	71,488,116	0.129	3,328,818
1905	37,688	38,291	1,531,429	40.63	534,907,619	242,699	80,693,232	0.151	4,184,070
1906	47,619	48,380	1,760,140	36.96	454,752,018	206,239	84,728,400	0.186	4,284,611
1907	99,141	100,727	2,452,562	24.74	508,929,401	230,799	94,912,185	0.186	5,888,170
1908	63,149	64,158	1,254,172	19.87	661,876,127	300,302	87,393,200	0.132	3,162,303
1909	59,880	60,841	1,335,316	22.29	682,846,726	309,734	89,367,455	0.131	3,217,185
1910	43,784	44,585	1,049,086	23.96	695,107,043	315,384	88,949,799	0.128	5,132,287
1911	57,915	58,844	2,352,080	40.61	786,553,208	356,778	98,168,182	0.122	5,159,664

Year.	Gold.		Iron.							
			Ore.				Pig.			
	In Coin and Bullion. (c)	In Ore. (d)	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
02	$35,722,835	$307,756	88,445	89,860	$294,168	$3.32	27,487	27,927	$502,947	$18.30
03	43,765,360	581,474	80,611	81,901	255,728	3.17	20,379	20,705	384,334	18.86
04	120,226,424	985,403	213,865	217,287	458,823	2.14	49,025	49,809	764,543	15.60
05	46,099,580	694,887	208,017	211,345	530,457	2.55	49,221	50,009	762,899	15.50
06	46,068,451	640,707	265,240	269,484	771,831	2.91	83,317	84,650	1,506,774	18.08
07	54,869,688	345,993	278,208	282,659	763,422	2.74	73,703	74,879	1,508,938	20.43
08	80,778,091	437,365	309,099	314,043	1,012,924	3.29	46,696	47,441	789,318	16.92
09	132,340,610	540,211	455,934	463,251	1,365,325	2.99	61,989	62,994	1,030,267	16.66
10	58,305,510	469,312	644,875	655,193	2,074,164	3.22	127,385	129,423	2,113,036	16.51
11	36,683,309	499,265	768,386	780,719	2,653,448	3.45	120,799	122,738	1,874,401	15.52

Year.	Iron, Bar.				Iron, Band, Hoop and Scroll.			Billets, Ingots and Blooms.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L. T.
02	22,249	22,605	$869,519	$39.08	1,674	1,701	$82,322	2,409	2,447	$74,938	$31.11
03	19,380	19 690	796,631	41.11	1,241	2,175	101,839	5,445	5,532	141,924	26.07
04	29,582	30,055	1,133,128	34.93	3,435	3,489	162,039	314,324	319,353	6,150,035	19.56
05	32,025	32,537	1,255,418	39.20	4,426	4,497	182,431	237,738	241,542	4,701,909	19.79
06	56,024	56,920	2,575,905	45.98	5,405	5,491	242,776	192,616	195,698	4,094,659	21.26
07	24,190	24,577	1,092,631	45.17	8,587	8,724	395,758	79,991	81,271	2,013,319	25.17
08	8,224	8,355	362,909	44.12	4,334	4,402	223,073	112,177	113,390	2,674,524	23.84
09	13,536	13,755	538,436	39.77	3,856	3,918	200,379	104,862	106,545	2,401,091	22.90
10	18,045	18,234	726,300	40.25	(n)	(n)	(n)	58,230	59,162	1,274,732	21.89
11	17,683	17,967	653,320	36.94	3,731	3,791	163,853	234,267	238,027	5,150,518	21.99

ear.	Iron, Nails and Spikes, Cut.				Iron, Nails and Spikes, All Other.				Iron, Plates and Sheets.		
	Lb.	Metric Tons.	Value	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Met. Tons.	Value.
02	16,122,775	7,312	$339,227	$0.021	64,565,650	29,287	$1,456,768	$0.022	3,434	3,489	$229,887
03	19,912,563	9,031	424,985	0.021	75,654,532	34,310	1,698,500	0.024	4,782	4,858	273,618
04	20,772,049	9,422	416,389	0.020	80,279,746	36,403	1,949,908	0.024	4,728	4,804	247,694
05	17,674,099	8,019	352,405	0.020	89,976,088	40,506	2,118,836	0.024	8,004	8,132	460,995
06	16,951,893	7,688	340,526	0.020	116,310,428	52,747	2,731,021	0.024	17,054	17,327	1,139,526
07	15,521,208	7,042	354,802	0.023	111,670,147	50,642	3,014,863	0.027	40,651	41,301	2,902,025
08	15,721,898	7,133	364,202	0.023	71,427,124	32,407	1,813,784	0.025	44,100	44,805	2,985,538
09	22,256,458	10,095	456,635	0.021	85,387,006	38,731	1,993,142	0.023	75,305	76,513	4,706,592
10	18,208,116	8,261	361,838	0.020	118,881,375	53,939	2,693,753	0.023	102,534	104,175	6,412,458
11	25,585,379	1,161	470,515	0.018	148,875,384	67,529	3,279,105	0.022	134,497	137,113	8,353,089

ear.	Steel, Sheets and Plates.			Iron Rails.				Steel Rails.			
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
02	14,866	15,104	$725,547	211	214	$4,639	$22.02	67,455	68,534	$1,902,396	$28.09
03	13,312	13,525	657,713	181	184	8,808	48.67	30,656	31,146	937,779	30.59
04	50,477	51,278	2,064,241	1,405	1,427	23,870	17.00	414,845	421,482	10,661,222	25.72
05	67,093	68,166	2,889,084	*Nil.*	.....	.....	.....	295,023	299,473	7,310,029	24.78
06	93,601	95,099	4,081,915	*Nil.*	.....	.....	.....	328,036	333,285	8,903,411	27.14
07	82,045	83,358	4,262,582	*Nil.*	.....	.....	.....	338,906	344,328	10,411,072	30.72
08	60,893	61,865	3,422,031	*Nil.*	.....	.....	.....	196,510	199,654	6,021,549	30.62
09	104,742	106,423	4,627,614	*Nil.*	.....	.....	.....	299,540	304,533	8,519,793	28.44
10	171,982	174,734	7,514,832	*Nil.*	.....	.....	.....	353,180	358,831	10,162,522	28.77
11	237,428	241,239	9,800,215	*Nill*	.....	.....	.....	420,874	427,629	12,229,045	29.05

Year.	Structural Iron and Steel.				Wire.				Steel Wire Rods.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T
1902 .	53,859	54,721	$2,828,460	$52.52	97,843	99,414	$5,140,702	$52.54	24,613	25,007	$831,067	33.7
1903 .	30,641	31,131	1,788,556	58.37	108,521	110,258	5,528,726	50.94	22,360	22,718	713,718	31.9
1904 .	55,514	56,402	2,777,768	50.04	118,581	120,478	5,935,093	50.05	20,073	20,394	695,448	34.6
1905 .	84,234	85,582	4,357,186	51.73	142,601	144,883	7,061,442	49.52	6,514	6,618	277,651	42.6
1906 .	112,555	114,356	6,140,861	54.56	174,014	176,798	8,770,042	50.40	5,896	5,990	221,679	37.6
1907 .	138,442	140,657	7,784,618	56.23	161,228	163,808	9,164,829	56.84	10,653	10,823	465,757	43.7
1908 .	116,878	118,746	6,289,610	53.80	136,167	138,344	7,270,794	53.35	7,412	7,530	277,694	37.5
1909 .	90,830	92,288	4,488,197	49.42	149,341	151,738	7,836,564	52.49	20,142	20,465	635,409	31.5
1910 .	146,721	149,069	7,127,673	48.58	171,928	174,679	9,198,005	53.49	22,869	23,235	714,553	31.2
1911 .	223,493	227,080	10,270,977	45.96	229,762	233,452	11,637,596	50.65	22,641	23,003	659,066	29.1

Year.	Lead and Mfres. of.	Nickel. (e)	Petroleum products. (In Thousands of Units.)*								
			Crude.			Naphtha.			Illuminating Oil.		
			M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.
1902..	$696,010	$924,579	145,234	$6,331	$0.042	19,683	$1,393	0.071	778,801	49,079	$0.06
1903...	491,362	703,550	126,512	6,782	0.054	12,973	1,519	0.118	691,837	51,356	0.07
1904...	616,126	2,130,933	111,176	6,351	0.057	24,989	2,322	0.093	761,358	58,384	0.07
1905...	511,699	2,894,700	126,185	6,086	0.048	28,420	2,215	0.078	881,450	54,901	0.06
1906...	600,057	3,493,643	148,045	7,731	0.052	27,545	2,488	0.090	878,284	54,858	0.06
1907...	686,096	2,845,663	126,306	6,334	0.050	34,625	3,676	0.106	905,924	59,635	0.06
1908...	599,640	3,297,988	149,190	6,520	0.044	43,887	4,543	0.103	1,129,005	75,988	0.06
1909...	509,542	4,101,976	186,305	6,568	0.035	68,759	5,800	0.084	1,046,401	67,814	0.06
1910...	614,158	4,704,088	180,111	5,404	0.029	100,695	8,407	0.083	940,247	55,642	0.05
1911...	680,419	8,283,777	201,843	6,165	0.030	137,295	11,483	0.084	1,112,295	61,055	0.05

Year.	Petroleum Products. (In Thousands of Units.)*									
	Lubricating Oil.			Residue, Etc. (g)			Paraffin.			
	M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.	M Lb.	M Metric Tons.	M Value.	Value per Lb.
1902 ...	82,201	$10,872	$0.133	38,316	$922	$0.024	175,269	79.5	$8,398	$0.048
1903 ...	95,622	12,690	0.133	9,753	282	0.029	204,120	92.6	9,596	0.047
1904 ...	89,738	12,389	0.138	34,904	1,174	0.034	174,582	79.2	8,273	0.047
1905 ...	113,730	14,312	0.126	70,728	2,128	0.030	160,836	73.0	7,873	0.049
1906 ...	151,269	18,690	0.124	64,645	1,971	0.030	173,504	72.9	8,463	0.049
1907 ...	152,029	19,210	0.126	75,775	2,528	0.033	207,504	94.1	10,209	0.049
1908 ...	147,769	18,971	0.128	77,552	2,793	0.036	141,667	64.2	6,923	0.049
1909 ...	161,640	20,076	0.124	107,999	3,640	0.034	181,328	82.6	7,609	0.042
1910 ...	163,832	20,291	0.125	117,605	3,732	0.032	199,913	90.7	7,329	0.037
1911 ...	183,320	23,337	0.127	133,979	3,882	0.029	214,018	97.1	7,048	0.033

Year.	Crude Phosphates.				Quicksilver.			Silver.	
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Lb.	Metric Tons.	Value.	In Coin and Bullion (c)	In Ore
1902 . .	802,086	814,919	$6,193,372	$7.73	1,013,434	459	$575,099	49,228,303	44,8
1903 . .	785,259	797,823	6,109,230	7.78	1,344,615	610	719,119	40,531,095	73,2
1904 . .	842,484	855,964	6,521,555	7.74	1,611,365	731	847,108	49,975,370	159,8
1905 . .	934,940	949,899	7,465,592	7.91	1,009,446	458	497,470	54,133,721	3,379,3
1906 . .	904,214	918,681	7,373,945	8.16	484,151	219	243,914	57,012,104	266,6
1907 . .	1,018,212	1,034,503	8,387,176	8.24	384,913	174	192,094	61,202,024	422,8
1908 . .	1,196,175	1,215,374	9,371,649	7.83	224,692	102	124,960	51,554,414	283,2
1909 . .	1,020,556	1,036,936	7,644,368	7.49	510,241	231	266,243	56,876,292	716,0
1910 . .	1,083,037	1,100,366	8,234,276	7.63	144,237	65	91,077	56,984,109	376,8
1911 . .	1,246,577	1,266,585	9,235,388	7.41	21,841	10	13,995	65,470,010	194,6

Year.	Zinc Ore.				Zinc Pigs, Bars, Plates and Sheet.			
	Long. Tons.	Metric Tons.	Value.	Value Per L.T.	Lb.	Metric Tons.	Value.	Value. per Lb.
1902	49,762	50,558	$1,449,104	$29.12	6,473,135	2,936	$ 300,557	$0.046
1903	35,188	35,751	987,000	28.05	3,041,911	1,380	163,379	0.053
1904	32,063	32,576	905,782	28.25	20,145,942	9,204	1,094,490	0.053
1905	27,630	28,072	848,451	30.71	11,031,815	5,005	682,254	0.062
1906	24,750	25,146	733,300	29.63	9,340,455	4,236	583,526	0.062
1907	18,171	18,462	579,490	31.89	1,126,753	511	75,194	0.067
1908	23,311	23,683	877,745	37.61	5,280,344	2,396	250,254	0.047
1909	11,121	11,299	412,300	37.01	5,131,360	2,328	263,010	0.051
1910	17,599	17,881	649,425	36.90	7,979,457	3,620	426,500	0.053
1911	16,322	16,584	642,884	39.39	13,744,212	6,234	810,099	0.059

Year.	Zinc Oxide.			
	Lb.	Metric Tons.	Value.	Value per Lb.
1902	10,716,364	4,861	$433,722	$0.040
1903	14,429,885	6,544	578,215	0.041
1904	16,313,826	7,399	628,494	0.039
1905	22,559,625	10,236	810,203	0.036
1906	31,156,616	14,129	1,149,297	0.037
1907	26,512,920	12,023	1,069,924	0.040
1908	24,016,254	10,893	845,070	0.035
1909	26,691,347	13,468	1,026,377	0.035
1910	26,333,993	11,949	943,968	0.036
1911	29,236,779	13,262	1,051,311	0.036

RE-EXPORTS OF FOREIGN PRODUCTS. (*a*)

Year.	Antimony.			Antimony Ore.			Asphaltum, Crude.		
	Lb.	Metric Tons.	Value.	Short Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902	37,184	16.9	$2,710	104	94.6	$4,602	2,930	2,977	$23,564
1903	79,917	36.0	4,478	*Nil.*	.....	.....	1,605	1,631	13,894
1904	31,077	14.0	1,734	214	194.0	10,775	1,887	1,917	26,272
1905	*Nil.*	.....	.....	*Nil.*	.....	.....	1,081	1,098	18,190
1906	24,892	11.2	4,939	*Nil.*	.....	.....	1,765	1,793	22,324
1907	47,999	21.8	9,064	6	5	273	8,288	8,421	31,749
1908	1,763	0.8	125	4.8	4.3	663	4,262	4,290	21,419
1909	6,648	3.0	475	0.25	0.23	56	6,867	6,977	48,375
1910	339,685	154.1	20,392	*Nil.*	.....	.....	5,830	5,948	29,942
1911	160,844	73.0	9,737	*Nil.*	.....	.....	3,402	3,457	48,552

Year.	Cement.			Chemicals.					
				Salts of Potassium. (*f*)			Chloride of Lime.		
	Bbl. (*i*)	Metric Tons.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.
1902	32,594	5,913	$48,797	1,266,145	574,323	$59,789	198,794	90,172	$2,997
1903	25,362	4,601	32,156	1,299,905	589,637	33,264	836,411	379,696	7,609
1904	39,711	7,186	54,486	1,262,222	572,544	33,358	1,434	650	13
1905	31,874	5,782	40,583	3,053,191	1,386,149	83,652	100	102	3
1906	16,216	2,941	19,487	2,264,175	1,027,935	77,043	*Nil.*	.....	.....
1907	20,697	3,754	30,435	2,675,248	1,285,892	75,470	*Nil.*	.....	.....
1908	9,552	1,734	11,455	1,046,689	570,445	34,505	121,511	55,116	912
1909	4,198	762	6,312	2,332,386	1,058,251	66,352	13,964	6,334	292
1910	17,914	3,251	24,878	2,187,787	922,645	60,648	496	225	6
1911	8,529	1,547	14,095	2,731,792	1,239,133	67,297	17,804	8,076	241

Year.	Chemicals. (*Continued.*)											
	Nitrate of Sodium.			Caustic Soda.			Soda Ash and Carbonate.			Sodium Salts, All Other.		
	Long Tons.	Metric Tons.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.
1902..	3,675	3,734	$144,650	1,343,132	609,246	$28,704	62,653	28,419	$931	115,491	52,386	$1,628
1903..	4,417	4,488	184,657	1,116,354	506,378	23,227	30,030	13,622	464	42,540	19,294	637
1904..	6,076	6,173	279,864	1,115,600	506,036	23,608	40,351	18,303	593	1,778,616	806,780	25,212
1905..	8,991	9,135	420,613	1,087,772	493,848	22,728	32,221	14,628	473	16,748	7,604	177
1906..	6,660	6,767	324,915	(1)	.....	.....	2,486	1,128	41	1,032,372	468,180	21,624
1907..	7,159	7,274	370,048	(1)	.....	.....	3,100	1,406	53	742,201	336,662	16,088
1908..	9,955	10,113	514,799	(1)	.....	.....	4,645	2,104	77	834,207	378,726	18,285
1909..	8,233	8,365	377,571	(1)	.....	.....	(1)	.....	....	1,053,410	477,816	21,777
1910..	5,784	5,878	250,550	(1)	.....	.....	(1)	.....	....	333,427	151,282	11,767
1911..	6,787	6,896	269,274	(1)	.....	.....	(1)	.....	....	611,922	277,566	14,479

Year.	Coal, Bituminous.			Copper.						Graphite.	
				Ore and Matte.			Pigs, Bars, Ingots, Old and All Unmanufactured.				
	Long. Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Lb.	Metric Tons.	Value.	Long Tons.	Value.
1902....	7,559	7,680	$22,153	14,446	14,657	$2,229,912	11,629,877	5,275	$1,604,522	12	$834
1903....	88,468	89,883	453,613	5,750	5,232	852,726	2,093,103	949	261,413	63	4,223
1904....	7,250	7,366	21,910	*Nil.*	....	.....	1,088,672	494	140,695	8	455
1905....	3,945	4,008	10,974	*Nil.*	....	.....	1,718,584	780	272,945	5	91
1906....	2,541	2,582	13,062	71	72	29,791	1,567,782	711	309,605	3	362
1907....	1,947	1,978	12,199	....	....	.....	995,555	451	199,828	1	41
1908....	4,759	4,832	16,313	2	2	50	718,541	326	93,148	....	....
1909....	3,128	3,178	8,532	434	441	5,600	1,058,528	480	135,952	16	976
1910....	3,734	3,894	13,761	*Nil.*	....	.....	55,857	25	6,443	34	3,192
1911....	3,718	3,778	10,296	*Nil.*	....	.....	200	0.1	52	*Nil.*	....

Year.	Iron and Steel.											
	Pig Iron.			Scrap.			Bar Iron.			Rails.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1902.....	250	254	$6,286	1,542	1,567	$25,020	22	22	$1,875	297	302	$7,184
1903.....	1,863	1,893	33,996	262	266	2,862	16	16	2,108	739	751	17,560
1904.....	1,646	1,672	25,910	190	193	2,367	7	7	765	96	98	2,305
1905.....	1,010	1,026	29,047	4,270	4,338	80,623	22	22	2,556	31	31	1,132
1906.....	6,750	6,858	236,957	5,111	5,193	101,886	61	62	7,207	*Nil.*	....	....
1907.....	2,921	2,968	86,420	157	160	3,378	38	39	3,959	*Nil.*	....	....
1908.....	1,827	1,855	52,079	288	293	3,597	26	26	1,271	*Nil.*	....	....
1909.....	720	732	25,936	....	....	.....	20	20	1,500	*Nil.*	....	....
1910.....	579	588	20,706	....	....	.....	9	9	356	*Nil.*	....	....
1911.....	1,837	1,866	73,987	100	102	1,373	12	12	635	*Nil.*	....	....

Year.	Iron and Steel. (*Continued.*)									Lead and Mfrs.
	Steel, Ingots, Blooms, Etc.			Sheets, Plates, Rods, Wire.			Tin and Terne Plates, Taggers Tin.			
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	
.....	106	108	$6,774	236	240	$14,221	98	100	$7,471	$3,553,144
.....	60	61	5,316	55	56	5,532	2	2	184	2,917,957
.....	40	41	6,208	108	110	6,482	81	82	5,306	2,880,907
.....	86	87	15,570	161	164	8,019	26	26	3,014	2,441,166
.....	196	199	14,104	318	323	27,631	0.4	0.4	28	2,307,345
.....	292	297	25,974	14	14	1,220	42.2	42.9	1,813	2,416,082
.....	33	34	9,822	66	69	3,441	4.7	4.7	351	3,101,953
.....	60	61	10,389	42	43	2,630	14	14	6,273	3,139,908
.....	121	123	24,867	96	98	6,122	.....	.....	.....	2,511,850
.....	108	110	19,721	.....	.....	34,177	.....	.....	.....	4,067,574

ear.	Salt.			Sulphur—Crude.			Tin in Blocks, Pig and Granulated.		
	Lb.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
.....	3,699,411	1,678	$7,155	207	210	$5,086	939	954	$562,350
.....	2,310,759	1,048	4,544	1,253	1,273	28,024	479	486	286,897
.....	7,804,215	3,585	26,636	967	982	22,658	512	520	317,805
.....	2,089,234	948	2,814	2,493	2,533	58,887	519	527	322,234
.....	611,912	278	893	1,713	1,741	36,858	557	567	375,763
.....	1,462,413	663	1,129	403	409	8,475	807	820	650,411
.....	1,166,049	529	1,686	301	306	5,759	562	571	492,415
.....	2,525,945	1,146	9,352	380	386	8,500	244	248	156,761
.....	1,617,705	734	1,700	16	16	284	441	448	294,649
.....	2,124,983	964	2,173	5	5	145	620	630	467,526
.....	449,025	204	646	*Nil.*	.....	.....	1,010	1,027	958,481

For convenience in tabulating, the quantities of all petroleum products and their gross values have been led by 1000.

(*a*) From Summary of Commerce and Finance of the United States. (*c*) Total exports of coin and bullion; is, includes both foreign and domestic. (*d*) Only approximately correct. The Bureau of Statistics rts only the value of silver ores exported, but a much larger amount of silver leaves the country in copper e, which is classified as copper ore, and no record is kept of its silver contents. The gold in copper matte rted is not included in the exports of gold given in the above table. These figures include ore of both estic and foreign origin. (*e*) Includes nickel oxide and nickel matte. (*f*) Includes chlorate, chloride, te and all other salts of potassium. (*g*) Reported in barrels, but calculated to gallons, on a basis of 42 ns to the barrel. (*i*) Barrel of 400 lb. (*l*) Included in all other salts of soda (*m*) Does not include used for fuel on vessels for foreign trade. (*n*) Included in other manufactures.

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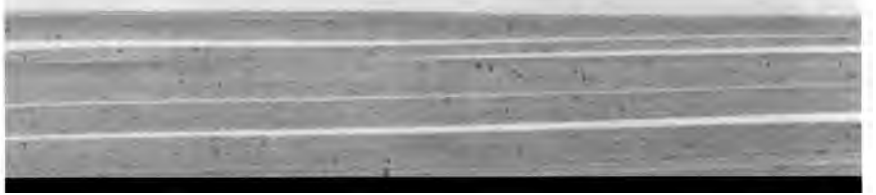

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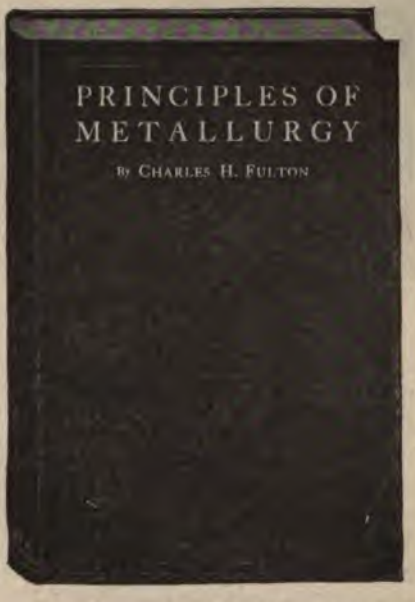

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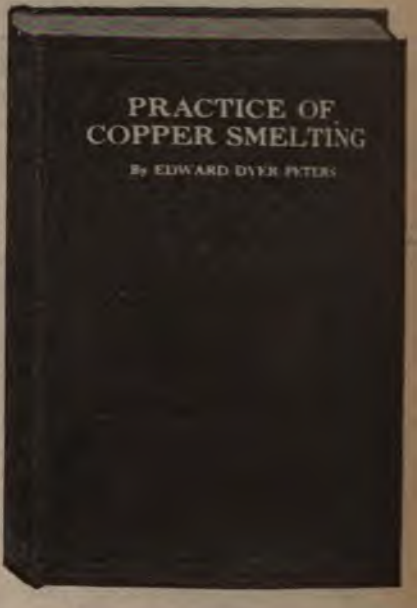

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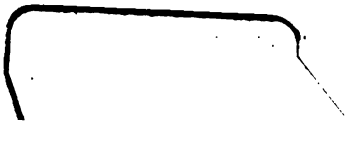
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